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THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA.

VOLUME XIV.

JULY, 1913 — JUNE, 1914.

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JOHANNESBURG
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CONTENTS OF VOLUME XIV.

PAGE
Papers and Notes Read before the Society ... ... iv.
Officers and Council of the Society ... ... v.
Journal No. 1.—July, 1913 1
,, ,, 2.—August ,, ... 49
,, ,, 3.—September ,, ... 113
,, ,, 4.—October ,, ... 169
,, ,, 5.—November ,, ... 233
,, ,, 6.—December ,, ... 289
,, ,, 7.—January 1914 337
,, ,, 8.—February ,, ... 365
,, ,, 9.—March ,, ... 394
,, ,, 10.—April ,, ... 421
,, ,, 11.—May ,, ... 445
,, ,, 12.—June ,, ... 469
Index to *Journal* 505

ERRATA.

Page 134. First column, line 30: for "ore" read "one."

Page 135. Second column, line 8: for "scientific" read "specific."

Page 135. Second column, line 13: between "usually" and "towards" insert the word "slowest."

PAPERS AND NOTES READ BEFORE THE SOCIETY.

Accounts, Costs and Metallurgical Records, A System of Keeping Mine and Mill, by M. W. Maclachlan ... 138-148

Annual Report, 1913-1914, Council's ... 469-472

Assaying Concentrates and Battery Chips for Gold and Platinum Metals, by Andrew Crosse, 373-374; Discussion 374-375, 422-423 ... 483-484

Black Sand, and Floating Particles of Metallic Minerals, Recovery of, by J. M. Neill (Vol. XIII, pp. 418-421; Discussion 421, 542-544), 72-73. Reply to Discussion ... 133-136

Charcoal, The Effect of, in Gold Bearing Solutions with Reference to the Precipitation of Gold, by M. Green, 257-263; Discussion 341-343, 371-373; Reply to Discussion ... 452-455

Classifiers and Classification, Hydraulic, Notes on, by G. H. Stanley, 445-452; Discussion ... 484

Copper Ores, Mining, at Messina, by J. Allan Woodburn, 53-64; Discussion 192-195, 244, 306; Reply to Discussion ... 401-402

Crown Mines Visit to, with full description of Reduction Plant ... 412-414

Cyanide Practice in India, by H. M. Leslie (Vol. XIII, pp. 407-417; Discussion 417-418, 541-542); Reply to Discussion 66. Addendum ... 66-67

Cyanide Solution, Analysis of a, from the Meyer & Charlton G. M. Co., by H. E. Williams ... 475-476

Cyaniding of Gold and Silver Ore at Caveira Mine, Portugal, by J. Hutton (Vol. XIII, pp. 312-313; Discussion 313-314, 421, 540-541) ... 1-2

Dust in Underground Workings, Prevention of, by B. C. Gullachsen, 337-341; Discussion 433, 459-463 ... 483

Electric Blasting, by W. Cullen, T. Donaldson, and W. Waters, 174-189; Discussion 189-192, 248-252, 307-308, 375-377. Reply to Discussion ... 431-433

Electric Parting Apparatus, Description of, by F. Wartenweiler ... 367

Explosives in Hand and Machine Labour Stopes, by T. D. Delprat, 241-243; Discussion 243-244 ... 308

Falls of Roof in Collieries on the Middelburg Coalfield Relation of, to Weather Changes, by Chas. J. Gray, 395-400; Discussion 400-401 ... 433-434

Gold, The Determination of, in the Presence of Iridium and Allied Metals in Materials such as Black Sand, by James Gray and Chris. Toombs, 2-9; Discussion 119; Reply to Discussion ... 292-294

Gold Production in Relation to Humanity: A Study of the Question: Has Gold Depreciated in Value? By E. M. Weston (Vol. XIII, pp. 472-481; Discussion 481, 611-618), 64-65, 136-138; Reply to Discussion ... 252-257

Graphite Deposits of Madagascar and Africa, Some Observations on the, by E. R. Bawden (Vol. XIII, pp. 258-259; Discussion 471-472). Reply to Discussion ... 1

Inaugural Address, by Alex. Richardson (President) ... 51-53

Iodine, The Solubility of, in Sodium Iodide Solution, by H. W. Gill 290-292; Discussion 292, 343-344, 378-379. Reply to Discussion ... 476-478

Launder Grades for Sand Pulp, by W. A. Caldecott ... 486-487

Mine Samples, Notes on the Assay of, by R. Dures (Vol. XIII, pp. 608-610); Discussion 68-72, 173-174. Reply to Discussion ... 263-264

Mining Exhibition ... 169, 367, 393, 421, 486-499

Nitrous Fumes in Firing ("Cheesa") Sticks, Note on the Quantitative Determination of, by Dr. L. Heymann (Vol. XIII, pp. 464-466; Discussion 466-467). Reply to Discussion 118. Correspondence ... 200

Oxidisers, The Action of, in Cyaniding, by Morris Green (Vol. XIII, pp. 355-359; Discussion 359-360, 422, 541). Reply to Discussion ... 65-66

Patents Bill, The Union, by A. L. Spoor and W. E. John, 296-302; Discussion 302-303, 379-381, 408-409, 428-430 ... 482-483

Platinum Metals in Cupellation Beads Detection of the, by means of the Microscope, by C. O. Bannister and G. Patchin, 478-482; Discussion ... 482-483

Precipitation by Zinc, A Method of, by J. S. MacArthur (Vol. XIII, pp. 310-311; Discussion 311-312, 421, 540-541), 1-2. Reply to Discussion ... 265-266

Primer Clip for Explosives, by Wm. Cullen ... 366

Referee Assays, by A. Whitby ... 367

Rock Temperatures, by E. J. Moynihan ... 455-459

Sand-Filling of Mines, The, by Dr. W. A. Caldecott and O. P. Powell, 119-132; Discussion 131-133, 264-265, 303-306 368-371, 409. Reply to Discussion ... 409-411

Screening, Fine, by A. McA. Johnston ... 394

Soda Deposits of Africa; The Natural, with some notes on the Alkali Trade, by J. Watson 235-241; Discussion 377-378. Reply to Discussion ... 463-464

Tin Ores, Proposed Method of Assay of, by A. Whitby 170; Discussion 171 ... 235

Vacuum Filtration at the Victoria Mill of the Waihi G.M. Co., Ltd., New Zealand, by W. Macdonald (Vol. XIII, pp. 527-539; Discussion 539), 67-68, 171-173. Reply to Discussion ... 430-431

Ventilation of the Mines of the Rand: The Problem of Obtaining Healthier Conditions, by G. H. Blenkinsop 148-151; Discussion 151, 195-200, 244-248, 306-307. Reply to Discussion 402-404

Waste Rock Values, Notes on, by R. Dures ... 233-235

Waters, The Determination of the Acidity or Alkalinity of, A Study in Indicators, by Dr. J. Moir 114-118
Discussion 118, 294-296. Reply to Discussion ... 404-407

Witwatersrand Earth Tremors, The, by H. E. Wood, 423-427; Discussion 466 ... 485

LIST OF OFFICERS FOR THE YEAR ENDING JUNE, 1914.

Hon. President:

H.E. The Right Hon. Viscount Gladstone, P.C., G.C.M.G.

Hon. Vice-Presidents:

Hon. F. S. Malan, M.L.A., Minister of Mines.
John Munro (President Transvaal Chamber of Mines).
Sir Lionel Phillips, Bart., M.L.A.
Max Elkan.

Past-Presidents:

Wm. Bettel, the late
Andrew F. Crosse,
W. R. Feldtmann,
Charles Butters,
Julius Löevy,
John R. Williams.
S. H. Pearce,
W. A. Caldecott,
William Cullen,
Edward H. Johnson,
John Yates,
R. G. Bevington,
A. McA. Johnston,
James Moir.
C. B. Saner,
Walford R. Dowling

President:

Alex. Richardson.

Vice-Presidents:

G. H. Stanley.
J. E. Thomas.
H. A. White.

Ordinary Members of Council:

T. Donaldson,
James Gray,
K. L. Graham,
H Meyer,
E. Pam,
A. Salkinson,
C. Toombs,
F. Wartenweiler,
F. W. Watson,
John Watson,
A. Whitby,
J. A. Wilkinson.

With the Past-Presidents, who are Members of the Council *ex officio*.

Corresponding Members of Council:

E. G. Banks,
Clement Dixon.
G. T. Holloway,
Walter Neal,
W. J. Sharwood,
T. B. Stevens,
Ralph Stokes,
G. M. Taylor,
Gerard W. Williams.

Hon. Treasurer:

James Littlejohn.

Hon. Auditor:

Alex. Aiken, F.S.A.A., F.C.I.S.

Hon. Legal Advisers:

Messrs. van Hulsteyn, Feltham & Ford.

Editorial Committee:

A. Richardson (*Chairman*),
H. Meyer,
J. Moir
E. Pam,
G. H. Stanley,
A. Whitby.
H. A. White.

Finance Committee:

R. G. Bevington,
J. Littlejohn,
A. Salkinson,
F. W. Watson,
J. A. Wilkinson.

Secretary:

Fred. Rowland, Fellow, Chartered Inst. Secretaries.
(P. O. Box 1183; Telephone 1404; Tel. Add.: "Scientific")
9, S.A. School of Mines & Technology Building, Eloff Street, Johannesburg.

CONSTITUTION AND RULES.

As amended and adopted at Special General Meetings of Members, held on 20th April, 1907, and further amended and added to, on the 18th April, 1908, 19th December, 1908, and the 17th April, 1909.

1.—The name of the Society is " THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA."

2.—The chief office of the Society shall be in Johannesburg, Transvaal.

OBJECTS.

3.—The objects of the Society are:—

(*a*) To advance Technical and Metallurgical Chemistry and the Science and Practice of Mining.
(*b*) To cultivate all branches of pure Chemistry.
(*c*) To extend the knowledge and practical application of improvements and discoveries in matters bearing on the Chemical, Metallurgical and Mining professions, and to afford Members opportunities for interchange of ideas on Chemistry, Metallurgy and Mining.
(*d*) To do all things which may be desirable for the attainment of the above-described objects, or any of them.

MEMBERSHIP, ADMISSION AND SUBSCRIPTIONS.

4.—The Society shall consist of Honorary Members, Life Members, Members, Associates, and Students.

5.—Honorary Members shall be such persons as are so elected by the Council. Their membership shall be for the current year only, and they shall enjoy all the privileges and rights of members, except those of voting and holding office.

6.—Life Members are Members who have commuted their subscriptions by a single payment of Twenty Guineas, or persons on whom the Society in General Meeting has conferred Life Membership, in consideration of services rendered to the Society, Science, or Industry. Life Members shall have all the privileges of Members.

7.—Members.—Every candidate for admission as a Member shall be proposed and seconded by two Members in accordance with the form in annexure " A " hereto. The proposal shall be submitted to the Council, and, if approved, the candidate shall stand for election at the Ordinary General Meeting next following. The election shall be by ballot, one adverse vote in ten to exclude. Notice of the names of the candidates for election shall be sent to all Members prior to the Meeting.

8.—Associates.—Applications for Associateship shall be made in the form provided in annexure " B " hereto. Associates shall be admitted by the Council. Should an Associate desire to become a Member of the Society, such Associate shall be proposed, approved, and elected in the manner provided for in Clause 7. Associates shall have all the privileges of Members, with the exceptions of holding office and voting.

9.—Students.—(*a*) Any *bona fide* student of a recognised technical school, college or similar institution, or (*b*) any man engaged as a *bona fide* apprentice or learner in mining or metallurgical operations, shall be eligible to be admitted a student member upon making application in the form provided for in annexure " C " hereto, and being duly vouched for by his instructor or by the head of the department in which he is employed. Students shall be admitted by the Council. Student membership ceases, *ipso facto*, under (*a*) on the completion of the course of studies; and under (*b*) at the end of each financial year, but it may be renewed on making a fresh application on the form " C." Students may be admitted as Associates or elected Members of the Society by complying with the provisions of Rules 7 and 8. Students shall have all the privileges of Associates.

10.—Every candidate shall on his election, or admittance, be duly notified of the fact by the Secretary, and no election shall be confirmed until the candidate shall have paid the fees to the Society on election or admittance.

11.—The Entrance Fee for Members, and for Associates and Students who may be subsequently elected to Membership, shall be One Guinea, payable on election.

12.—Each Member shall pay an annual subscription of Two Guineas; each Associate shall pay an annual subscription of One and a Half Guineas; and each Student shall pay an annual subscription of Half a Guinea. All subscriptions shall become due on the first day of July in each year, and shall be payable in advance.

13.—Should any Member, Associate, or Student be elected on or after January 1st of any year, his subscription for the remainder of the current financial year shall be one-half of the usual subscription.

14.—The Council shall have power to strike the names of Members, Associates, and Students, whose subscriptions are in arrear, from off the Roll of the Society, or to deal with such defaulters as in their discretion they see fit.

15.—Members, Associates, and Students may at any time resign from the Society by giving written notice to the Secretary; and on payment of all liabilities to the Society then existing on their part, such resignations shall be accepted by the Council.

16.—Every person ceasing to be a Member of the Society from any cause whatever shall thereupon forfeit all right, title, and interest in and to the funds, property, and assets of the Society.

17.—A complete list of the names of Members, Associates, and Students, with their addresses, shall be kept, and shall be printed and circulated annually with the Society's *Journal.*

18.—Should not less than Six Members or Associates address to the Council in writing a complaint against the continued connection with the Society of any Member, Associate, or Student, the Council shall institute an inquiry, and should it resolve that the complaint is justified, it shall take such measures as it may deem expedient for the suspension, resignation, or expulsion of such Member, Associate, or Student. Provided, however, that it shall be competent for such Member, Associate, or Student, in the event of suspension or expulsion, to appeal against the action of the Council to the Society in Special General Meeting, whose decision shall be final.

ELECTION, POWERS AND DUTIES OF COUNCIL.

19.—The Management of the affairs of the Society shall be vested in the Council subject to the Constitution and Rules; which Council shall consist of a President, three Vice-Presidents, an Hon. Treasurer, and twelve Ordinary Members. Past-Presidents of the Society shall be Vice-Presidents and Members of Council.

20.—The Council of the Society shall be elected annually from the members by the members. The declaration of election shall be made at the Annual General Meeting in June of each year, and the Council thus elected shall hold office until their successors are elected.

21.—Candidates for the offices of President, Vice-Presidents, Hon. Treasurer, and Members of Council shall be proposed and seconded by members, and all such nominations shall be lodged with the Secretary not less than 30 days prior to the date of the Annual General Meeting. On receipt of the nominations the Secretary shall notify each nominee, and request within seven days his acceptance or otherwise of such nomination, and failing acceptance such nomination shall be void. Nominations for the offices of President, Vice-President and Hon. Treasurer shall be *ipso facto* nominations for ordinary Membership of Council.

22.—At least seven days prior to the Annual General Meeting the Council shall cause a list of all accepted nominations, which shall form a Ballot List, to be forwarded to each member whose subscription for the current year is paid.

23.—In voting, Members shall vote for a President, for not less than two nor more than three Vice-Presidents, an Hon. Treasurer, and for not less than eight nor more than twelve Members of Council.

24.—The Ballot List duly completed shall be returned to the Secretary so as to reach him not later than noon on the day of the Annual General Meeting, after which hour the Scrutineers shall commence their duties.

25.—At the Ordinary General Meeting next preceding the Annual General Meeting in each year there shall be elected from amongst the members not less than three Scrutineers whose duties shall be to open and examine the Ballot Lists, and report the result of their scrutiny, which shall be final, to the Chairman at the Annual General Meeting.

26.—The votes for any candidate who may not be elected President, Vice-President, or Hon. Treasurer, shall count for him as Vice-President, Hon. Treasurer or Member of Council respectively, but in no case shall any candidate receive more than one vote from any voter. In the event of the number of votes being found to be equal for any two or more candidates, all of whom cannot be declared elected, the retiring President shall determine the election by his casting vote.

27.—The office of any of the Twelve elected Members of Council shall be vacated by non-attendance at Council Meetings for three consecutive months without leave.

28.—The Council shall have power to fill any casual vacancy in their number.

29.—The Council shall have power to appoint Honorary Officials and Honorary Members for the current year.

30.—The Council shall have power to elect each year "Corresponding Members of the Council" from among full Members of the Society resident abroad. Each such official shall hold office during the year for which he is elected, provided, however, that his tenure of office shall cease as soon as he shall cease to reside and be in the country which he represents. The Council may refer to such officials matters concerning the country they respectively represent for consideration and advice.

31.—The Council shall have power to appoint a Secretary and other officers and servants, and to remunerate them. Such Secretary, officers, and servants shall hold office during the pleasure of the Council.

32.—At Meetings of the Council, five Members shall form a quorum, and all matters shall be decided by a majority, the Chairman having a casting vote. The Council shall have power to appoint committees for such purposes, and with such powers as the Council may determine. Such committees may consist of Members, Associates, or others.

33.—The Council shall cause Minutes to be kept of the proceedings at all meetings of the Society and of the Council.

34.—All Funds, Property, and Assets of the Society shall be vested in the Council for the time being in trust for the Members of the Society. The Treasurer shall administer the funds of the Society in accordance with the directions of the Council, to whom he shall be responsible.

35.—The Council shall cause true accounts to be kept of all moneys received and expended by the Society, the matters in respect of which such receipts and expenditure take place, and the Assets and Liabilities of the Society.

36.—The Council shall submit to each Annual General Meeting a report on the position of the Society, and an audited financial statement and balance-sheet made up to a date not more than one month prior to the meeting.

37.—The accounts of the Society shall be audited at least once a year by an Auditor, or Auditors, who may be appointed by the Members at each Annual General Meeting for the ensuing year; no Member of Council or officer of the Society shall be eligible as Auditor. If any casual vacancy occurs in the office of Auditor it shall be filled at the next ensuing ordinary General Meeting.

38.—Subject to these presents, the Council may from time to time frame Bye-laws for the conduct of the business of the Society and the management of its affairs; such Bye-laws may at any time be added to or amended by the Council. Copies of current Bye-laws shall be supplied to all Members and Associates, or published in the Journal of the Society.

LOCAL SECTIONS.

39.—A local section may be formed by the Council whenever at least twenty members or associates residing in or identified in interest with any locality, district, colony or country, whether in South Africa or not, desire to have local facilities for meetings, and sign and forward a requisition to the Council, asking to be permitted to form a section for that purpose. The consent of the Council being obtained, the signatories shall elect a local committee and officers and formulate Bye-laws, subject to the approval of the Council of the Society, for conducting the business of the Section.

40. The Chairman of the local committee shall *ex officio* become an Hon. Vice-President of the Society, and the Hon. Secretary shall *ex officio* become a Corresponding Member of Council, during their respective terms of office.

41.—Places and dates of Sectional Meetings and the business to be transacted thereat shall be arranged by the local committee of the Section.

42.—The Sectional subscription, if any, for defraying local expenses shall be fixed by the local committee, but shall not exceed 10s. 6d. per annum for each member or associate.

43.—Members and associates resident in or identified in interest with the district included in the Section shall, upon application to the Hon. Secretary, be entered in the register of the Section and be entitled to the privileges thereof. The local committee and all officials of the Section shall be elected annually by the members on the register of the Section.

44.—Papers read before any Section and the discussions thereon shall, subject to the approval of and editing by the Editorial Committee of the Society, be printed in the Journal of the Society under the heading of the Section.

45.—The expense incurred in the publication in the Journal, of Sectional papers and contributions to discussion, shall be borne by the general funds of the Society, and a contribution to the local expenses of the Section may be made from the general funds of the Society by the Council.

46.—The Society as a body shall not be responsible for any local expenses or obligations incurred by sections, although it has discretion to contribute thereto.

47.—Amendments in, alterations of, or additions to the bye-laws of a Local Section shall be submitted to the Council, and shall not become operative or of any effect unless and until they have been allowed by the Council, and notice of such allowance has been given by the Secretary.

48.—The Council may, at a special meeting called for that purpose, declare by resolution that a Local Section shall be suspended in operation, or shall be dissolved:

(*a*) If it shall appear that the further continuance of the Section is not in accord with or conducive to the interests of the Society, or the furtherance of its objects.

(*b*) Whenever three-fourths of the Members of the Local Section vote in favour of a resolution, proposed at a meeting called for the express purpose, that it is desirable such Section shall be suspended or dissolved.

MEETINGS.

49.—Annual General Meetings of the Society shall be held once in every year on dates to be appointed by the Council, to receive and consider the Report of the Council, with the Financial Statement and Balance Sheet, to elect Officers and Council in place of those retiring, and to transact such other business as, in the discretion of the Council, may be transacted at ordinary General Meetings.

50.—Ordinary General Meetings of the Society shall be held on such dates as may be appointed by the Council for the purpose of reading papers and considering matters connected with the objects of the Society, and for the election of Members.

51.—All other Meetings of the Society shall be Special General Meetings, at which such business only shall be transacted as shall be specified in the notice convening the Meeting.

52.—The President, or in his absence a Vice-President, or in the absence of all of them, a Member elected by the Meeting shall take the chair at all Meetings.

53.—The Council may at any time, and they shall, upon a requisition made in writing by not less than ten members, convene a Special General Meeting. Every such requisition shall specify the objects of the Meeting required, and shall be signed by the Members making the same, and shall be deposited with the Secretary; it may consist of several documents in like form signed by one or more of the requisitionists.

54.—In case the Council shall fail or neglect for seven days after such deposit to convene a Special General Meeting to be held within thirty days of such deposit, the requisitionists or any other Members may themselves convene a meeting to be held at any time within six weeks of such deposit by giving the notice required for Special General Meetings in the manner hereinafter provided.

55.—Fourteen days' notice at the least, specifying the place, day, and hour of all Annual General and Special General Meetings, shall be given, either by advertisement or by notice sent through the post, and in the case of Special General Meetings the nature of the business to be transacted thereat shall be stated.

56.—The accidental omission to give such notice, or the fact that such notice shall merely state generally the objects of the Meeting, shall not invalidate any resolution passed at such Meeting.

57.—All Meetings of the Society shall be held in Johannesburg, or at such other place in South Africa as the Council may determine.

58.—The quorum for all General Meetings of the Society shall be twenty-five members personally present. Subject to the provision of Article 69, a majority of votes shall carry any business before the meeting.

59.—Except as otherwise provided for in these presents, every question submitted to any Meeting shall, in the first instance be decided by a show of hands, but a poll may be demanded by any six Members, and shall thereupon be taken at once. The Chairman of of any Meeting shall have a casting vote in the event of an equality of votes.

60.—Members unable to be present at any General Meeting may vote by proxy; the person appointed as proxy must be a Member or the Secretary of the Society, and his appointment must be in writing.

61.—Two Scrutineers, appointed by the Members present at any General Meeting, shall conduct the ballot at such Meeting.

62.—The Chairman of any Meeting may, with the consent of a majority of Members present, adjourn the same from time to time and from place to place.

MISCELLANEOUS.

63.—Every Member, Associate, or Student who is not in arrear with his subscription shall be entitled to a copy of the monthly proceedings, if and when the same are published.

64.—Authors shall be entitled to twenty copies of their papers free of cost, if and when the same are printed.

65.—A notice may be served by the Society on any Member, Associate, or Student, either personally or by despatch through the post to the registered address of such Member, Associate, or Student. Every Member, Associate, and Student shall name an address to be registered as his address for service, and if he does not name such an address he shall be deemed to have waived the service of the notice upon him. All notices which may be given by advertisement shall be published in the Journal of the Society, or in such newspaper or newspapers as the Council may determine

66.—All contributions communicated to the Society, with their illustrative drawings, shall become the property of the Society unless stipulation be made to the contrary; and Authors shall not be at liberty, save by permission of the Council, to publish or cause to have published such contributions until they have either appeared in the Journal of the Society, or a period of three months shall have elapsed since the date of their being handed to the Secretary.

67.—No Member, Associate, or Student of the Society shall make use of any portion of the Society's Proceedings in a published document connected with a commercial undertaking, without the written authority of the Council, and of each individual speaker or writer concerned, previously obtained. The infraction of this Bye-law will be regarded by the Council as justifying action under Rule 18.

68.—All Deeds, Documents and Writings requiring execution on behalf of the Society, shall be signed by the President and Treasurer, or in the absence of either, or both, by Members of the Council duly authorised by a Minute of the Council.

69.—Alterations in or additions to the Constitution and Rules shall only be made by a majority of not less than two-thirds of the votes of Members present in person or by proxy at a special General Meeting convened for that purpose.

ANNEXURE A.

We, the undersigned Members of The Chemical, Metallurgical and Mining Society of South Africa, hereby recommend

Candidate's Name in full

as a fit and proper person to become a Member of the above Society.

For Names *only* of Members recommending the Candidate } Proposer
Seconder

And I, the said Candidate for Membership, hereby declare that if elected I will do my best to support the Society, and will observe its Rules and Bye-Laws.

Signature of Candidate

Address of Candidate to be registered.

Profession or Business Occupation

Date

Passed for Ballot at Council Meeting 191 .

Elected 191 .

Secretary.

ANNEXURE B.

I hereby apply for admission as an Associate of The Chemical, Metallurgical and Mining Society of South Africa, and declare that if admitted, I will do my best to support the Society, and will observe its Rules and Bye-Laws.

Signature of Applicant

Applicant's Name in full

Address to be Registered

Profession or Business Occupation

Date

Admitted by Council

191 .

Secretary.

ANNEXURE C.

I hereby apply for admission as a Student of The Chemical, Metallurgical and Mining Society of South Africa, and declare that if admitted I will do my best to support the Society and will observe its Rules and Bye-Laws.

Signature of Applicant

Applicant's Name in full

Age Monthly Salary (if any) Nature of Occupation

Name of Institution or Mine

Address to be Registered

Date of Entry

I hereby certify that the above application is bona-fide.

Signed

Instructor or Head of Department.

Date of Application 191

Admitted by Council 191

Secretary.

ANNEXURE D.

FORM OF BEQUEST.

Every person desirous of bequeathing to the Society any sum or sums of money, manuscripts, books, maps, plans, drawings, instruments, specimens, or other personal property, is requested to make use of the following form in his will:—

> "I give and bequeath to The Chemical, Metallurgical and Mining Society of South Africa (here enumerate and particularize the effects or property to be bequeathed); and I hereby declare that the receipt of the Treasurer of the said Society for the time being shall be an effectual discharge to my executors for the said legacy or bequest."

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. XIV. JULY, 1913. No. 1.

Notice to Members.

Owing to labour disturbances it was regarded as inadvisable to hold a meeting of the Society in July. A large amount of matter being in type it was decided to issue the *Journal* as usual, which contains in addition to the ordinary notices and abstracts, Mr. E. R. Bawden's reply to discussion, a closing contribution to the discussion on the papers by Messrs. MacArthur and Hutton, and the prize papers dealing with the Determination of Gold in the Presence of Iridium.

SOME OBSERVATIONS ON THE GRAPHITE DEPOSITS OF MADAGASCAR AND AFRICA.

(*Read at December Meeting, 1912.*)

By E. R. Bawden, M.I.M.E. (Member)

REPLY TO DISCUSSION.

Mr. E. R. Bawden (*Member*): Owing to the fact that practically no serious attempt has been made to mine graphite in Africa, it was only natural that the discussion was of a limited nature.

The contribution by Mr. Ralph Stokes is very much to the point, but unfortunately I am not privileged to give the results of my sampling of these deposits mentioned in detail. Large areas of the disseminated deposits contain no more than 4% to 7% graphite, but at several points I have seen an enrichment going as high as a 25% graphite. Deposits of quite another type I have seen which carried 40% to 50% graphite carrying about 15% "chip" in a matrix of amorphous plumbago. I also know of an instance where 1,000 tons, carrying 60% of fibrous and flaky graphite were raised from 12 months preliminary prospecting operations. I think that the majority of failures in graphite mining that I know of have been caused, first: by over capitalisation; secondly: by being controlled by people unacquainted with the nature of the graphite industry; and thirdly: by being managed by engineers who have had no special training in this particular branch of mining

Nearly all the mines in Ceylon are worked by small syndicates, and rarely have been originally capitalised in more than £10,000. Madagascar has already commenced to produce graphite of the LL quality on a commercial scale; this production is controlled by people having an intimate knowledge of the mining, cleaning and marketing of the mineral. No one should attempt to exploit graphite deposits without having first taken expert advice from those who have had a long standing experience of all the conditions that govern the mining, cleaning, marketing and occurrence of the mineral.

A NEW METHOD OF PRECIPITATION BY ZINC.

By John S. MacArthur, M.Inst.M.M. (Hon. Member).

CYANIDING OF GOLD AND SILVER ORE AT CAVEIRA MINE, PORTUGAL.

By James Hutton (Associate).

(*Read at January Meeting, 1913*).

DISCUSSION.

Mr. J. H. G. Banks (Waihi, N.Z.) (*Associate*): I regret that any remarks which I have to make on the subject of zinc wafer precipitation are so much delayed in being put before you, but delay was unavoidable owing to late arrival of the *Journal* and also owing to the time taken to make the tests, which extended over a period of two months. In view of the fact that experiments are being made on the Rand, and will doubtless be made on many other fields, a summary of the results obtained and the discussion thereon will be very interesting. It is rather difficult entering upon a discussion of this nature from such a distance and I only wish I could be present to put my views more fully before you, however, this cannot be, so I must be content with giving you result of tests as carried out by the Waihi Company and trust they will give additional interest to the discussion. The wafers used in our tests were cut from waste zinc from the lathe, and, although not very uniform, yet were sufficiently so to enable us to get a good idea as to the merits of the method. We took one precipitator of eight compartments and fitted up eight trays of wafers. The depth of wafers was 3 in., weight approximately 40 lb. The result of a number of tests shows an average extraction of gold 50%, silver 65%, whereas with the compartments filled with zinc shaving the extraction is 95% to 99%. The solutions in the bulk of the tests assayed before entering the boxes gold 4 dwt., silver 32 dwt., the solution strength varying from 0·023% to 0·07% KCN. Rate of flow 80 tons per 24 hours. It is possible that want of uniformity of the wafers was responsible for a slightly poorer result than would be obtained if the method were finally adopted. Compared with the Caveira Mine, our head values were higher and solution strength lower, but even stronger solution (0·07% KCN) failed to improve the extraction.

Mr. Hutton claims in favour of the wafer method that the bullion slime is very high grade, requiring no acid treatment, reduced zinc consumption and less labour in handling the slime. As regards the high grade slime, Mr. MacArthur states (p. 311) that the bullion precipitate contains 3% zinc and 12% foreign matter, therefore 85% bullion. This is ideal, but Mr. Hutton states (p. 313) that the precipitate averages 35% fine metal, which is very low. Seeing that there is such a difference between the figures there must surely be a misprint, or have I not read them correctly? The precipitate which we obtained from the wafers melted 66·5% bullion 950 fine which is very good although not a great deal better than we can obtain from zinc shaving by mechanical means.

Consumption of Zinc.—It is only reasonable to suppose that the consumption of zinc would be lower with the wafer method, for reasons pointed out by Mr. MacArthur, but the zinc consumption at the Caveira Mine using wafers is higher than with our use of shavings.

The following are the figures:—

	Caveira.	Waihi.
Bullion extracted per ton of ore	3·06 oz.	2·8 oz
Zinc to fine metal recovered ...	2·18 to 4·8 to 1	2·1 to 1
Zinc per ton of ore	8 to 16 oz.	5·835 oz. troy

The Waihi figures were taken over the past four months.

At Waihi we have selenium and copper to contend with, certainly they exist in very small quantities, but the last named is at times present in the bullion slime to the extent of 7% or 8%.

Regarding labour for cleaning out the extractors, we did not find that the slime was freed from the wafers with the ease which Mr. Hutton would lead one to suppose, although by using a trommel or some such mechanical device, the removal of the slime, and cleaning up generally could no doubt be very much simplified. If we had been able to extract say 80% or 90% of the value in our three compartments of wafers there would have been an advantage in the method, but it is unreasonable to suppose that such was possible even allowing that per square feet of surface, the wafers have a greater efficiency than shavings. Seeing that it is the surface exposed which must be reckoned on, it is a simple matter to calculate the weight of zinc wafers necessary to meet our requirements, the disadvantage of having such a weight discounting anything to be gained from the production of high grade slime. In the matter of the production of high grade slime and also lessened zinc consumption, I think the method of washing through a fine screen and returning the shorts to a series of superimposed trays set in the first compartment of the extractor is good. This method was used at several plants and was tried by us early in 1911, but was not persevered with.

To sum the matter up, our first point is good precipitation and this we did not get, but we will await with interest the result of experiments on the Rand and elsewhere, and no doubt we will be further enlightened on the subject before finally coming to a conclusion.

THE DETERMINATION OF GOLD IN THE PRESENCE OF IRIDIUM AND ALLIED METALS IN MATERIALS SUCH AS BLACK SAND.

As reported at the June meeting of the Society, the prize of twenty guineas offered by the Witwatersrand Co-Operative Smelting Works, Ltd.,* has been divided and awarded to "Cuprous" (Mr. Chris. Toombs) and "Propria" (Mr. Jas. Gray). The following are the prize papers, which are printed as received:—

Paper Submitted

By James Gray, F.I.C. (Member of Council).

Most of the methods which have been suggested for the estimation of gold and the platinum group of metals have reference to material in

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*See this *Journal*, Vol. XIII., Dec. 1912, p. 242.

which a large proportion of these metals is present and under these circumstances the classical methods of Deville and Stas (1), and the more recent methods of Leidie and Quennessen (2), are applicable without any preliminary treatment of the ore. The metals in the material under discussion must be concentrated by means of an assay before the above methods can be applied. Unfortunately, a considerable amount of analytical skill as well as time is necessary (besides an extensive equipment, which is usually beyond the average Rand assay office), in an assay by these methods and, therefore, they need not be considered further.

The other methods which have been suggested can be roughly divided into two classes which, however, depend to some extent upon one another. They are :—

1st. Processes depending on the selective action of acids followed by precipitation of the metals obtained in solution by this means.

2nd. Those involving direct precipitation of gold from solutions of the chloride and subsequent estimation of same.

Mietzschke's (3) method, which is a type of the first class is :—"Four quantities of 0·25 gm. of the alloy are cupelled with silver. Two of the resulting cornets are boiled with concentrated sulphuric acid and two with nitric acid and the residues weighed. The gold is submitted to quartation a second time and then boiled with nitric acid by which means silver and platinum are removed. Part of the iridium will be found floating in the silver solution, which is filtered to recover it. The gold-iridium is dissolved in *aqua regia* when most of the iridium will be left undissolved. The gold is precipitated with ferrous sulphate and the remainder of the iridium obtained by evaporation of the filtrate."

Rossler (4) treats one cornet with nitric acid and one with sulphuric acid and the excess of weight of the first over the second is reckoned as platinum. For large quantities, the silver is first removed with sulphuric acid and the residue dissolved in dilute *aqua regia*, avoiding excess of nitric acid. Rhodium and iridium are left undissolved. The gold in the filtrate is precipitated with ferrous chloride. He also states, that if ferrous sulphate is used platinum may be precipitated with the gold.

The methods in the second class consist in the removal of the silver by sulphuric acid and solution of the residue in *aqua regia*, filtration to remove any iridium or allied metals which may be insoluble and then precipitation of the gold which is collected and weighed. The platinum metals are estimated by difference.

Many methods have been suggested with regard to the precipitation of gold, it being of course essential that the reagent should only precipitate gold. On reference to the literature on the subject it is found that the following reagents, among others, have been proposed :—Hydrogen peroxide (5), formaldehyde (6), hydrazine hydrochloride and hydroxylamine hydrochloride (7), dextrose (8), acetylene (9), etc., etc. It would, however, appear that hydrogen peroxide is the most satisfactory, both from the point of view of cost and of precipitating gold only, for with regard to the other reagents some authorities are of opinion that other metals are precipitated along with gold.

The details of the method which I have adopted in carrying out an estimation of this kind are as follows :—

0·5 to 2 A.T. of the material are assayed in the usual manner and the lead button cupelled. The bullion thus obtained is weighed—silver, gold and platinum metals. Owing to the tendency of the platinum metals to cause the bead to scatter over the cupel it may be necessary, when a large proportion of these metals is present, to add silver before cupelling. The bullion is then cupelled with a sufficient excess of silver and the alloy after being well flattened, preferably in the mill, is treated with hot concentrated sulphuric acid and after washing, the residue is weighed—gold and platinum metals. The residue is then dissolved in *aqua regia* and the solution filtered to remove any insoluble platinum metals. The filtrate is then made alkaline with caustic soda, the change from deep yellow to almost white being easily noticed. From 10 to 20 cc. hydrogen peroxide (20 vols quality), are then added when the gold comes down immediately as a black precipitate according to the following equation :—

$2AuCl_3 + 3H_2O_2 + 6KOH = 2Au + 3O_2 + 6KCl + 6H_2O$. The mixture is then boiled for about ten minutes and filtered. After washing it is run down in a crucible with borax, litharge and reducing agent or scorified and the resulting lead button cupelled and the gold thus obtained weighed. The difference in weight between the weight of the gold + platinum metals and the gold gives the platinum metals.

It is advisable to run a blank on pure gold simultaneously and the loss is added to the final weight of gold recovered. It will be found that with ordinary care 99·5% of the gold is easily recovered.

In the case of dilute solutions it is advisable to acidulate after the oxygen has been expelled by boiling.

The platinum metals may be recovered from the filtrate by means of sulphuretted hydrogen and determined according to the methods of Leidie and Quennessen. The residue insoluble in *aqua regia* as stated above may be recovered in a convenient form by cupelling the filter paper.

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Paper Submitted

By Chris. Toombs (Member of Council).

LINES OF INVESTIGATION.

The following are the lines on which the research was carried out:—

(*a*) Research on a simple and rapid method for the recovering of total gold dissolved in aqua regia.

(*b*) Research on the operation of flattening, rolling, annealing and parting small silver-gold buttons.

(*c*) Research on methods of fluxing and treatment of black sand so as to obtain the highest result for both gold and iridium contents.

(*d*) Research on separation and estimation of gold and iridium.

(*e*) Examination for platinum.

A

Before proceeding with the investigation of gold plus iridium, the first thing to be considered is the following:—

Starting with a piece of pure gold weighed accurately to 0·01 mgm., dissolving it in aqua regia, what is the possibility, and what is the best method of recovering it again as metallic gold, to within 0·01 mgm.?

If one looks up the text books on this subject it would appear to be a very simple matter, but when one comes to try most of the methods on small quantities of about 20 mgm. pure gold it is very difficult to recover the full amount to within 0·01 mgm. or 0·02 mgm., and any method employed for assaying the gold contents of material handled by customs works, must be accurate to within at least 0·02 mgm., this representing 0·02 oz. on 1 assay ton, or approximately 1s. 8d. per ton.

EXPERIMENT I.

A weighed quantity of pure gold was dissolved in dilute aqua regia in small covered porcelain capsules heating gently on water bath. Solution transferred to lead capsules, evaporated to dryness on water bath, heated to about 120° C., cupelled with silver, parted and the recovered gold weighed.

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·14 mgm.	19·07 mgm.	1·07 mgm.

EXPERIMENT II.

Repeated Experiment I., using long test tubes for the dissolving of the gold in aqua regia.

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·18 mgm.	19·57 mgm.	0·61 mgm.

EXPERIMENT III.

Repeat of Experiments I. and II., using very long necked parting flasks for the dissolving.

Average of Three Determinations.

Original.	Recovered.	Deficit.
19·99 mgm.	18·98 mgm.	1·01 mgm.

EXPERIMENT IV.

Evaporated gold aqua regia solution in lead dish with litharge, scorified, cupelled, etc., etc.

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·49 mgm.	19·47 mgm.	1·02 mgm.

EXPERIMENT V.

Evaporated gold aqua regia solution in lead dish to dryness, added water, then oxalic acid, took to dryness, heated to 120° C., scorified, etc., etc

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·36 mgm.	19·41 mgm.	0·95 mgm.

EXPERIMENT VI.

Took gold aqua regia solution to dryness in lead dish with formaline, added water and formaline and evaporated to dryness again, scorified, cupelled, etc., etc.

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·35 mgm.	19·89 mgm.	0·46 mgm.

N.B.—The individual determinations except in Experiment VI. were very erratic.

EXPERIMENT VII.

Dissolve gold in 2 cc., 10% aqua regia, transfer to lead dishes, neutralise with NaOH, add little NaCN then some Cu_2Cl_2 evaporate to dryness, heat to 120° C., scorify, cupel, etc., etc.

Average of Six Determinations.

Original.	Recovered.	Deficit.
20 36 mgm.	20·10 mgm.	0·26 mgm.

From the results of above experiments it is evident that an evaporation-in-lead-dish-method is not sufficiently accurate to be of any value, the precipitation by cuprous chloride as in slime residue assaying, giving the best results, but repeat in triplicate giving the same deficit as the original experiment done in triplicate.

EXPERIMENT VIII.

Dissolve gold in 2 cc., 10% aqua regia, neutralise with NaOH, add NaCN then Cu_2Cl_2 in excess, filter, fuse, cupel, etc., etc.

Average of Three Determinations.

Original.	Recovered.	Surplus.
20·27 mgm.	20·41 mgm.	0·14 mgm.

The treatment after cupelling in all above experiments were as follows:—

Silver gold bead removed from cupel cleaned from any adhering cupel matter, tapped with hammer on anvil, annealed, put through bullion rolls, annealed and again through rolls, then parted in routine weak parting acid of sp. gr. 1·15, 2½ silver to 1 gold being used in each case.

EXPERIMENT IX.

Cupellation Checks.

The flattening and parting being carried out as in Experiment VIII. and previous experiments.

Average of Three Determinations.

Original.	Recovered.	Surplus.
19·95 mgm.	20·08 mgm.	0·13 mgm.

This gives practically the same surcharge as in Experiment VIII. so that one might conclude that cuprous chloride used as above gives a full precipitation of the dissolved gold.

EXPERIMENT X.

Repeat Experiment VIII., but boil in a second strong parting acid.

Average of Three Determinations.

Original.	Recovered.	Deficit.
20·23 mgm.	20·17 mgm.	0·06 mgm.

The individual determinations agreed in having a loss.

B

Research on treatment of silver-gold beads after cupellation.

EXPERIMENT XI.

2½ silver to 1 gold were cupelled together in this experiment, the beads cleaned and rolled out.

Length after rolling	...	30 mm.
Breadth after rolling	...	6 mm.
Thickness after rolling	...	0·25 mm.

The parting was done in a silica-ware parting apparatus to ensure uniform acid treatment, a crystal of $AgNO_3$ being added to the parting acids.

	Original mgm.	Recovered mgm.	Difference mgm.
A Unannealed, 1 acid treatment ...	20·32	20·50	+0·18
	20·37	20·57	+0·20
B Annealed, 1 acid...	20·37	20·40	+0·03
	20·43	20·47	+0·04
C Unannealed, 2 acids	20·04	20·12	+0·08
	20·36	20·46	+0·10
D Annealed, 2 acids	20·43	20·35	−0·08
	20·38	20·32	−0·06

EXPERIMENT XII.

Repeat (B) and (D) Experiment XI. Gave (B) longer time in the boiling acid, about 8 minutes in all.

	Original mgm.	Recovered mgm.	Difference mgm.
Annealed, 1 acid	20·64	20 66	+0 02
	20·01	19·99	+0 02
Annealed, 2 acids	20·09	20·05	−0·04
	20·29	20·22	−0·07

EXPERIMENT XIII.

Repeat (D) Experiment XI., *i.e.*, annealed and boiled in weak and strong parting acids, giving longer time in the second boiling acid than in Experiments XI. and XII.

Original.	Recovered.	Difference.
19·00 mgm.	18·90 mgm.	−0·10 mgm.
19·47 ,,	19·34 ,,	−0·13 ,,
18·84 ,,	18·70 ,,	−0·14 ,,
19·16 ,,	19·04 ,,	−0·12 ,,

The parting acid employed was of the following strength :—

Weak acid .. 5 HNO_3 to 12 H_2O
Strong acid ... 1 HNO_3 to 1 H_2O

The weak acid being about the strength of ordinary routine parting acid, while the stronger is not so strong as generally used in bullion assay practice. Each of the acids was well boiled to drive off nitrous fumes and then cooled to about 85° C. before using, and also a precautionary crystal of $AgNO_3$ added in case of the presence of any Cl. With regard to nitrous fumes—the operation of parting produces them copiously, so it is quite possible that gold may be dissolved from that source.

EXPERIMENT XIV.

The weak and strong parting acids used in Experiments XIII. were filtered through a close filter paper, HCl added in sufficient quantity to precipitate the silver, which was filtered off; the filtrate was evaporated almost to dryness three times, adding HCl to drive off HNO_3, the residue taken up with water, neutralised, NaCN added, and cuprous chloride (Cu_2Cl_2), filtered, fused, cupelled, etc., etc.

Found in weak acid	Trace
Found in strong acid	Trace

The filter papers through which the acids were passed were also fused, and no gold found on them. Further work is required to clear up this point as the traces of gold found in solution do not account for the deficit.

C

Some time ago I made a series of experiments to determine the method giving the highest gold result on black sand, especially roasting before fusing, against fusing direct, utilising the sulphur present as the reducer. I almost invariably found that direct fusing gave the highest result. The experiments were made on black sand containing from 7 oz. to 10 oz. gold plus iridium, the iridium being present in some cases in only traces. For this investigation I have now gone into the subject closely, and the results are most interesting. The theory is held that if the iron in a slag is largely in the ferric state some of the gold is held up in the slag. A series of fusions were done to test this theory, wrought iron nails, or flour in excess being used in an endeavour to obtain a ferrous slag.

The following are the fluxes used :—

Raw black sand ...	1 A.T.
Soda carbonate ...	2½ ,,
Borax	1 ,,
Litharge	1½ ,,

with and without iron nails

Roasted black sand ...	1 A.T.
Soda carbonate ...	2 ,,
Borax	1½ ,,
Litharge	1½ ,,

as reducer 4 or 5 wrought iron nails or 4 gm. flour.

Five grams of a washed slime residue going 0·08 dwt. gold to ton was used to coat the roasting dishes to prevent the black sand adhering. The roasting was done at a scarcely visible red heat in muffle. (Of course black sand was weighed before roasting.)

EXPERIMENT XV.

	Gold oz.	Iridium oz.
Unroasted plus iron nails...	16·70	0·56
Unroasted no reducer ...	16·58	0·93
Roasted plus nails ...	16·60	0·25
Roasted plus flour ...	16·54	1·29

Fusions were done in triplicate, above being averages. These fusions and estimations were made before the possibility of a loss of gold in cupelling and parting was thought of, and there was a deficit of about 0·2 oz. in the sum of gold and iridium, against both weighed together before separation. The results of the above are instructive. The use of iron nails would appear to drive the iridium into the slag, and also would appear to bear out the theory stated early in this paragraph as in the case of the unroasted black sand plus iron nails a distinctly higher gold value was obtained than in any of the other fusions. The slag too on examination was most decidedly ferrous.

EXPERIMENT XVI.

In this experiment an endeavour was made by thoroughly fluxing with iron to drive all the iridium into the slag and obtain a clean gold bead. Some ordinary iron filings were obtained in the workshops, and the portion passing through a 30 × 30 sieve used as reducer in flux.

Black sand	1 A.T.
Soda carbonate ...	2 ,,
Borax	2 ,,
Litharge	1½ ,,
Iron filings	10 gm.

	Gold oz.	Iridium oz.
Unroasted	16·29	1·37
Roasted	16·43	1·50

The opposite result to what was expected has taken place. The gold values being lower than in Experiment XV., and opposed to them also, as a higher value is obtained from the roasted ore. As regards the iridium—this would appear to be a good method to flux for a high result in that metal. Further results on the use of iron and flour will be found in the next paragraph.

D

Separation and estimation of gold and iridium when obtained together after parting operations.

The information I have been able to obtain with regard to iridium has not been great, and I must say somewhat conflicting. However, there seems to be a consensus of opinion that iridium is insoluble in dilute aqua regia, and the results of my experiments seem to bear this out. To be on the safe side a solution of 10% aqua regia has been used in the following work.

The method employed was to weigh the gold and iridium together on the assay balance, then separate by dissolving the gold in dilute aqua regia, obtaining and weighing the two metals separately, and thus finding out how close the sum of the two came to the two weighed together originally. My first results were very disappointing, deficits of from 0·1 to 1 mgm. being found, with the result that the investigation had to be made in the complete manner already described, and it was not till the parting operation, and loss in the operation, was understood, that success to my satisfaction was obtained.

The following is the method used for separation of the gold and iridium :—

Parting from the silver is best done in porcelain crucibles. The first trouble met with is, that the iridium often separates from the gold in a very finely divided state, very difficult to settle for the necessary decantation of acid, and washes to follow. An endeavour to overcome this was tried with varying success as follows :—The acid was brought to the boil and then cooled to about 80° C., when the cornet previously rolled and annealed was dropped in. It was left standing till action ceased, and then put on a hot plate and gradually heated till just at boiling, without any ebullition taking place however, after 5 min. it can be removed, decanted, washed with three washes very carefully, dried and annealed. If this operation is done with care the iridium will be found adhering to the gold.

Personally I don't see why the parting acid and subsequent washes, cannot be passed through a very small ashless filter (ash = 0·03 mgm) the paper ignited and added to the bulk of gold and iridium. However, I did not go further into this. Transfer the gold and iridium to the balance pan, if—as happens when the iridium comes away from the gold—iridium is found adhering to the crucible, it can be removed as follows :—A scrap of ashless filter paper about the size of a shilling is damped with distilled water, and with it on the tip of the finger wipe out the crucible, ignite and add to the main bulk on the balance pan ; the error introduced by the scrap of paper being negligible. Weigh the gold and iridium together, then transfer to a long narrow test tube on which a spout has been formed by heating the edge in a burner, and when soft, drawing the edge of a penknife blade along it. A set for routine use can be quickly prepared. Test tubes have the advantage over porcelain crucibles in preventing concentration of the aqua regia. A beaker with a piece of thick rubber insertion, having holes to just take the test tubes, as a cover, makes a good water bath. To the gold and iridium in test tube add 1 cc. 10% aqua regia, place in the water bath and heat to 60–70° C., action ceases, and the gold is dissolved under half an hour. Use very small funnels and ashless filter papers. The papers used were Schleicher and Schull's No. 590 ; the 9 cm. having an ash of 0·05 mgm. were cut down to 5 cm. so that the ash would be from 0·02 to 0·03 mgm. Smear the spout of the test tube with a film of grease, and filter. Wash thoroughly till free from gold. As a last precaution against any of the very heavy iridium remaining in the test tube, fill test tube full up with distilled water, then carefully invert funnel over it and quickly turn upside down, the water will run through quickly leaving any little iridium on the paper. This last operation must not be done till everything is washed gold free. As a precautionary measure a wash with dilute NH_3 to remove traces of AgCl should be given. The filter paper is then dried, ignited and weighed, result being the iridium. The filtrate is next neutralized with 2·5 cc. normal soda solution, 2 cc. of a 10% NaCN added, then 5 cc. Cu_2Cl_2 solution, stirring well while adding the Cu_2Cl_2. After 2 min. add a second 5 cc. Cu_2Cl_2 which seems to clarify the solution : filter, sprinkle the paper with litharge, add the necessary silver, fuse, cupel, flatten, roll and anneal, part by boiling in weak parting acid, anneal and weigh. If these operations are carried through carefully the sum of the separate estimation of the gold and iridium will agree with the original to within ± 0·02 mgm.

With every series I would strongly advise that a blank experiment with fine gold (in duplicate might be safer) be carried out, starting at the cupellation of the buttons obtained from the original black sand fusions. As soon as the assayer has obtained confidence in himself, and the method, it will be no longer necessary to precipitate and recover the gold which is a lengthy process. He will simply weigh the gold and iridium together, dissolve out gold and filter off the iridium wash and weigh, deducting from the total weight to obtain the correct gold value. Personally I would now have no hesitation in doing this.

Sample Submitted by the Witwatersrand Co-Operative Smelting Works.

The fusions and fluxing were carried out as in Experiment XV. so as to obtain further information on the use of iron in fluxing, and also as to the advantage of treating raw or roasting.

EXPERIMENT XVII.

		Au + Ir.	Au.	Ir.	Differ.
		oz.	oz.	oz.	oz.
A	Unroasted + nails...	19·80	19·10	0·68	– ·02
		19·76	19·05	0·70	– ·01
B	Unroasted no reducer	20·48	19·06	1·32	– ·10
		20·39	18·98	1·30	– ·11
C	Unroasted + flour...	20·02	18·46	1·56	nil
		20·14	18·64	1·47	+ ·03
D	Roasted + nails ...	18·69	18·29	0·38	– ·02
		18·83	18·29	0·58	+ ·04
E	Roasted + 1 gm. flour	20·61	19·01	1·60	—
		20·54	18·99	1·59	+ ·04

Checks cupelled and treated with above.

Original.	Recovered.	Difference.
18 90 mgm.	18·87 mgm.	– ·03 mgm.
18·96 ,,	18·95 ,,	– ·01 ,,

To be added to gold value 0 02 oz.

Filter ash = 0·03 mgm. to be deducted from iridium value.

Slags run down with excess litharge flour and soda carbonate, (C) and (D) showed gold, others none.

In the above there seems to be a tendency for a direct fusion to give the highest gold value, but the difference is so small in its favour that one might put it down to experimental error. There is no doubt, however, about the method giving the highest iridium value, viz., roasting plus excess flour as reducer, although the unroasted plus flour method runs it close but gives low gold results. I would report the gold value of the above sample from the average of the following:

	19 10 oz.
	19·05 ,,
	19 06 ,,
	18·98 ,,
	19·01 ,,
	18·99 ,,
Average ...	19·03 ,,
plus deficit on checks	0 02 ,,
Total ...	19·05 ,,
Iridium	1·60 ,,
	1·59 ,,
Average ...	1·59 ,,
Less filter ash	0·03 ,,
	1·56 ,,
Gold ...	19·05 oz. per ton
Iridium ...	1·56 ,,

EXPERIMENT XVIII.

A further test of the above method was made on a black sand residue low in gold and iridium.

Au + Ir.	Ir.	Au.	Difference.
oz.	oz.	oz.	oz.
0·43	0·11	0·34	+ ·02
0·43	0·10	0·34	+ ·01
0·43	0·11	0·34	+ ·02

There are several points still which require further research, in particular the operation of parting, especially on very thinly rolled out cornets, as I undoubtedly found a loss on boiling in the second strong nitric acid. Time does not permit me to investigate this point further at present.

The question of which method of fusion gives the highest gold value also requires further investigation, and both these points could be more easily handled by the senior students at the College, than by busy professional men.

E

Examination for Platinum.

EXPERIMENT XIX.

The parting acids used in dissolving out the silver from the gold and iridium in 10 cornets, had the silver precipitated with excess HCl, filtered and evaporated to dryness, taken up with HCl and any further silver filtered off. Excess HCl driven off, and potassium chloride and alcohol added.

Result—platinum absent.

EXPERIMENT XX.

10 gold cornets obtained from the black sand submitted were dissolved in dilute aqua regia leaving a trace of residual matter which proved to be AgCl. The solution was taken almost to dryness several times with water, and finally some ammonium chloride added with alcohol.

Result—platinum and iridium absent.

Method for General Use.

Cover three roasting dishes with a layer of fine silica, or washed slime residue, and on each spread out 1 assay ton of the black sand. Roast at a scarcely perceptible very dull red heat in the muffle till sulphur free. Flux as follows:—

Black Sand...	1 A.T.
Soda carbonate	2 A.T.
Borax	1½ A.T.
Litharge	1½ A.T.
Flour	4 gm.
Silver	q.s.

Black sand of course weighed before roasting.

Mix thoroughly in mortar. Transfer to paper bag, and roll into a very compact parcel, which will fit well into the pot. Have pots in the fire, and when at a good red heat, place bags in pots, closing off draught for a minute till fusion has taken place, then open up draught finishing at a nearly white heat. Wash with mixture of litharge and flour; pour and cupel. With the series cupel

1 or 2 pieces of pure gold, with the necessary silver. Remove beads from cupels, clean off any adhering cupel matter, flatten and anneal: roll and anneal. Part in porcelain crucibles, using dilute nitric acid ($5HNO_3$ to $12H_2O$). Bring the acid to the boil, cool to about 80° C., throw in cornet. Action should be brisk but not vigorous, so as to keep the iridium adhering to the gold. Put on hot plate and heat to about boiling, without any actual ebullition: decant off acid, wash, and anneal. I would strongly advise annealing over a Bunsen burner (the writer uses a spirit Bunsen) as there is a tendency for the iridium to stick to the porcelain crucible if heated too strongly. Weigh gold and iridium together; dissolve out gold with 1 cc. 10% aqua regia in a long test tube. Filter off the iridium, wash, dry and ignite over a Bunsen burner (not in muffle), weigh, make a deduction for filter ash, the remainder being iridium. Subtract this from sum of gold and iridium, this gives the weight of fine gold in the 1 A.T. fused. Weigh the parted pure gold checks and make an addition or subtraction as the case may be.

In the writer's opinion the foregoing is all that is necessary to obtain very accurate gold and iridium results.

The roasting plus excess of flour method has been selected as giving good gold results, with a maximum value for iridium over any of the other methods tried, and it is most essential that the assayers employed by the mines (the sellers) should use the same method (roasting, fluxing, etc.) as the assayer at the smelting works, otherwise an iridium result of anything from a trace up to 2 oz. may be obtained.

If it is desired to actually recover and weigh the gold separately, the precipitation by cuprous chloride from an alkaline cyanide solution will be found to give maximum results in a minimum time.

Preparation of Cuprous Chloride.

Fill a Winchester quart bottle about one-third with copper sulphate crystals, fill up with strong hydrochloric acid and shake occasionally till the copper sulphate is dissolved. Place about 8 or 10 pieces of stout copper wire almost the length of the bottle, inside. In about 4 or 5 days the solution becomes almost colourless, and is then in a good healthy cuprous state, a good test for it being to let one drop fall into a beaker of water when a milkiness should be produced.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Delicate Reactions for Gold.—"When one volume of a 0·1% solution of certain aromatic reducing agents, such as quinol, pyrogallol, etc., is added to 10 volumes of a solution contains 0·002% of auric chloride, a violet colloidal solution of gold is produced in the course of two minutes (*Analyst*, p. 54, 1913; abstr. *Journ. Soc. Chem. Ind.*, Mar. 15, 1913) The colours vary according to the reagent used and the strengh of the solution, those yielded by metol (mono-methyl-para amido phenol) and quinol being particularly fine. *p* phenylenediamine gives a dark yellowish-green solution, which after a time undergoes further colour changes, and this reaction, being specially delicate, is suggested as a test for traces of gold salts, for which purpose one volume of a 0·1% solution of *p*-phenylenediamine is added to 10 volumes of a solution containing 0·0005% of gold chloride; a distinct greenish-yellow colouration is observed even with a solution containing only 0·0001% of the gold salt."—*Engineering and Mining Journal*, April 19, 1913, p. 804. (H. A. W.)

Temperatures of Decomposition of Carbonates.—"The author subjected a number of pure mineral carbonates in the form of fine powder to the heating action of an electric current, which was so regulated as to raise each specimen to a temperature of 1130° C. within an hour. The periods of time, during which heating was in progress, and the corresponding temperatures were plotted on curves, which indicated for each substance under examination the point at which absorption of heat began, and the point at which it reached a maximum. The former indicated the temperature of incipient dissociation (*a*) and the latter the temperature of maximum decomposition (*b*) respectively, in all cases except those of barium and strontium carbonates, which were found to undergo molecular transformations at temperatures about 300° C. below their temperatures of decomposition. The values of (*a*) and (*b*) for the different minerals examined are as follow: for cerussite ($PbCO_3$), (*a*) about 315° C., (*b*) 335° C.; smithsonite ($ZnCO_3$), (*a*) about 395° C., (*b*) 440° C.; siderite ($FeCO_3$), (*a*) about 400° C. (?), (*b*) 460°—500° C.; basic lead carbonate, (*a*) 430° C., (*b*) 460° C.; rhodochroisite ($MnCO_3$), (*a*) 510° (?)—525° C., (*b*) 550°—570° C.; magnesite ($MgCO_3$), (*a*) 570° C., (*b*) 600° C.; calcspar and aragonite, ($CaCO_3$), (*a*) 895° C., (*b*) 910° C.; strontianite ($SrCO_3$), (*a*) over 1130° C.; witherite ($BaCO_3$), (*a*) over 1100° C."—K. Friedrich, *Journal of the Society of Chemical Industry*, April 15, 1913, p. 361. (J. A. W.)

Detection of Arsenic by Bettendorf's Reagent.—"Most text-books direct that Bettendorf's reagent should only be used cold, since, on heating, the stannous chloride reduces any sulphuric acid and sulphides are formed. In the author's experience this is not the case. For the preparation of the reagent, 100 gm. of clear crystals of stannous chloride are dissolved in sufficient of the strongest commercial hydrochloric acid (36—38%) to make 1 litre of solution. The solution becomes slightly brown on standing, owing to the presence of traces of arsenic in the acid, and must not be used until the liquid is completely clarified. Clarification requires several days in a stoppered bottle standing in a warm place. It may be assisted after the first day by adding to a litre of the solution about 1 gm. of very finely powdered glass and shaking. The clear colourless liquid is decanted off for use. For the test, about 2 cc. of the solution to be tested, prepared as far as possible with hydrochloric acid, are treated in a test-tube with 10 cc. of the reagent, heated to boiling, and laid aside for half an hour. In testing concentrated sulphuric acid for arsenic, about 1 cc.

is diluted with an equal volume of water and then mixed with 10 cc. of the reagent. The test is apt to fail if an insufficient quantity of the reagent be employed. The limit of sensitiveness of the test, using the above quantities, is about 0·001 gm. of arsenious oxide per litre: 0·01 gm. per litre shows a strong reaction after half an hour. The test as described above has been adopted in the Hungarian Pharmacopœia, Ed III."—L. W. WINKLER, *Journal of the Society of Chemical Industry* (from *Z. angew. Chem.*, 1913, *26*, 143-144), April 15, 1913, p. 383. (F. W. W.)

HYDRAULIC LIME.—"In calculating the hydraulic modulus $\frac{CaO + MgO}{SiO_2 + R_2O_3}$ of a sample of hydraulic lime, any undecomposed calcium carbonate and insoluble silica must be taken into consideration by subtracting the amount of lime combined with carbon dioxide, and the amount of silica which is not soluble, from the respective constituents as they appear in the formula. The nearer its hydraulic modulus approximates to that of Portland cement (1·7 to 2·1), and the smaller the amounts of insoluble residue and calcium carbonate which it contains, the greater is the value of a hydraulic lime. It was found that by raising the temperature of calcination, so as to decompose the calcium carbonate completely without dead-burning the lime, a stronger-setting hydraulic lime was obtained, yielding sand-mortars which had 2½ times the crushing strength of sand-mortars prepared from the same raw materials calcined at a lower temperature. The addition of 2% of stucco plaster, or of 2½% of raw gypsum, greatly increased the crushing strength of hydraulic lime mortars, especially when these were exposed to atmospheric conditions only." G. HENTSCHEL (*Tonind. Zeit.*, xxxvi, 1843), *Journal of the Franklin Institute*, vol. clxxv, May, 1913, No. 5, p. 510. (J. G.)

HELIUM IN GASES FROM SPRINGS.—"While the rare gases argon, krypton, xenon, and neon seem to occur in natural springs approximately in quantities of the same order as in the atmosphere, the percentage of helium is sometimes far higher, and some springs in the Côte d'Or contain 10% of helium in their total gas content, consisting of nitrogen and the rare gases. The Source Carnot at Santenaz yields 17.815 litres of helium per annum. This helium might be 'young' (*i.e.*, liberated as it is formed) or 'fossil' (*i.e.*, liberated after long accumulation). The first assumption would require the disintegration of 91 tons of radium or of 5·0 million tons of pitchblende per year for the production of the helium, hence this seems out of the question."—C. MOUREU and A. LEPAPE (*Comptes Rendus*, clv. 197), *Journal of the Franklin Institute*, clxxv, May, 1913, No. 5, p. 495. (J. G.)

A NEW CARBON OXIDE. "Tertiary polycarboxylic acids of appropriate structure, if completely anhydrised, should be capable of yielding binary compounds of carbon and oxygen. Thus furanetetracarboxylic acid should yield a substance of formula, C_8O_7, and pyronetetracarboxylic acid one of formula, C_9O_8. The authors have succeeded in obtaining such an anhydride from mellitic acid. When the acid is boiled for a long time with thionyl chloride or heated with it at 160° C., four of the six carboxyl groups are anhydrised, and a 1:4 carboxylic acid, of formula $(CO{\cdot}O{\cdot}CO)_2C_6(CO_2H)_2$ is produced. This is a white crystalline powder, which dissolves in hot water to re-form mellitic acid, and decomposes if heated dry, yielding pyromellitic anhydride and carbonaceous products. It cannot be further anhydrised: but if mellitic acid be heated for six hours or more under a reflux condenser with a large excess of benzoyl chloride, part goes into solution, but part is converted into the anhydride; the latter separates out as shining colourless crystals, which can be recrystallised from hot benzoyl chloride. It is insoluble in cold water, but takes up water when heated for some time with it, to form mellitic acid. It can be dried at 160° C. without decomposition, and sublimed in a vacuum, but decomposes above 320° C. Its analysis gave results corresponding with the formula $C_{12}O_9$, or $C_6(CO{\cdot}O{\cdot}CO)_3$."—H. MEYER and K. STEINER, *Journal of the Society of Chemical Industry*, April 15, 1913, p. 362. (J. A. W.)

VOLUMETRIC DETERMINATION OF TIN WITH POTASSIUM BROMATE BY THE METHOD OF H. ZSCHOKKE.—"The method, which is based on the fact that stannous chloride, in presence of excess of hydrochloric acid, reduces bromates quantitatively to bromides in accordance with the equation: $3SnCl_2 + 6HCl + HBrO_3 = 3SnCl_4 + HBr + 3H_2O$, is carried out as follows: 0·15 gm. of aluminium wire (in pieces) is added to 20 cc. of a slightly acid solution of stannous or stannic chloride, containing about 5·95 gm. of tin per litre, the mixture is allowed to stand for a short time, and the tin, which is thus precipitated (aluminium going into solution), is redissolved, as stannous chloride, by adding a mixture of 30 cc. of concentrated hydrochloric acid and 20 cc. of water, closing the containing flask with a Bunsen valve, and gradually heating to boiling; the valve is then closed, the flask cooled, and the cold solution titrated as quickly as possible with a potassium bromate solution, containing in the litre 1/60 gm.-mol. = 2·7837 gm. $KBrO_3$, until a yellow colour appears (indicating liberation of bromine by interaction of hydrogen bromide and the excess bromic acid). The bromate solution is standardised iodometrically. The process may be employed in the presence of phosphoric acid and is said to be both rapid and accurate; although devised specially for use in silk-dyeing works, it may be applied to the analysis of alloys and of tin compounds generally."—F. FICHTER and E. MULLER, *Journal of the Society of Chemical Industry* (from *Chem.-Zeit.*, 1913, *37*, 309), April 15, 1913, p. 382. (F. W. W.)

THE DETERMINATION OF PHOSPHORUS IN STEELS CONTAINING VANADIUM.—"In order to precipitate phosphorus quantitatively as phosphomolybdate from steels containing vanadium, it is necessary to reduce the latter to the quadrivalent state, otherwise precipitation is incomplete and there is contamination of the phosphomolybdate by vanadium. In the present method ferrous sulphate in slight excess is used as the reducing agent, and the following conditions should be observed:

(1) The temperature of precipitation should be held at a point (15° to 20°) where the nitric acid does not oxidize the excess of ferrous salt or the reduced vanadium before complete precipitation of phosphorus takes place; (2) the partial neutralization with ammonia, frequently used when phosphorus is precipitated as phosphomolybdate, must be made before reduction of the vanadic acid, otherwise the heat of neutralization causes the oxidation of most of the ferrous iron and reduced vanadium by the nitric acid; (3) care must be taken to prevent the

action of oxides of nitrogen, formed by interaction of ferrous salt and nitric acid, on the reduced vanadium, since these substances seem to catalyze the oxidation of the vanadyl salt and may in some cases completely prevent precipitation of phosphorus, owing to the large amount of vanadic acid produced; (4) efficient means for shaking or agitation of the solutions in which precipitation is to take place must be provided."—CAIN and TUCKER (Notes from the U.S. Bureau of Standards), *Journal of the Franklin Institute*, vol. clxxv, May, 1913, No. 5, p. 531. (J. G.)

METALLURGY.

ELECTROMETALLURGY OF ALUMINIUM.—"As a preliminary to an investigation of the binary system, NaF—AlF$_3$, the authors have prepared pure anhydrous aluminium fluoride by sublimation from the commercial salt. This is heated to 1100° C. in a platinum crucible in a kryptol electric furnace and the pure salt sublimes in large crystals on the crucible lid. By thermal analysis of the system, NaF—AlF$_3$, the authors have established the melting point of cryolite as 1000° C. Cryolite is a very stable substance and can be melted frequently without decomposition. In addition to this there is another compound, 5NaF,3AlF$_3$, corresponding to the mineral, chiolite, and having a transition temperature of 725° C. There are also eutectics at 885° C. (14 mols. per cent. AlF$_3$) and 685° C. (46·4 mols. per cent. AlF$_3$). When alumina is added to cryolite a binary system is produced in which there is an unbroken series of solid solutions up to about 21 % by weight of alumina which represents the limit of solubility at ordinary pressures. The minimum melting point is reached at 935° C. The solubility of the alumina is increased by the addition of sodium fluoride, probably owing to the formation of an aluminate. Between the eutectic, NaF—Na$_3$AlF$_6$, and pure sodium fluoride the solubility of alumina again diminishes. Addition of aluminium fluoride to cryolite lowers the solubility of alumina. The decomposition voltage of alumina in molten cryolite is 2·1 volts to 2·2 volts, but there is a break in the voltage curve at 1·2 volts to 1·3 volts, caused by the formation of aluminium carbide (which is strongly exothermic) at the cathode. Investigations of the sparking phenomena at the anode during electrolysis show that it can be caused not only by the electrolyte becoming poor in alumina, but also by the presence of large quantities of alumina accumulating at the anode. Like other metals, aluminium when exposed to the action of a molten salt, forms a 'metal mist' or fine suspension which diffuses to the surface and there volatilises or oxidises. Experiments were made to measure this 'solubility' and showed that the current losses in the electrical manufacture of aluminium are largely to be attributed to this cause. The action of the molten electrolyte on carbon is also of importance. An addition of sodium fluoride to a solution of alumina in cryolite increases the rate of disintegration of the carbon crucibles very considerably. It was also established that aluminium and carbon under a layer of a molten mixture of cryolite and alumina at 1000° C.—1100° C. may form a considerable amount of carbide without the intervention of the electric current. Actual electrolytic experiments were carried out and showed the possibility of obtaining current yields of 60%—70%, the limit being set by the 'solubility' of the aluminium referred to above. It is found that the addition of sodium fluoride to the molten cryolite leads to unfavourable conditions and a low yield. Although the melting point is lower, the concentration of alumina at the anode becomes so raised during electrolysis that sparking takes place, the anodes are attacked, and carbide formation is plentiful. The difference between the specific gravity of the bath and that of the metal is also not sufficient for proper working. Other alkali fluorides and chlorides have probably a similar action. With chiolite it is possible to get a normal electrolysis when the bath contains not more than about 7·5 % of alumina, while for cryolite about 10 % is best. The anode current density must not be more than 5 amp. per sq. cm. The advantage of an excess of aluminium fluoride over that present in cryolite is that the temperature of the bath is lowered, but it is not desirable to reduce this below about 900° C., otherwise the specific gravity of the bath becomes too near that of aluminium. The temperature is often lowered by the addition of calcium fluoride, but at the same time the solubility of the alumina is lowered and the specific gravity of the melt is increased. Mixtures of calcium fluoride and aluminium fluoride form an eutectic at 815° C.—820° C. (37·5 mols. per cent. AlF$_3$), but the solubility of alumina in this mixture is very slight and its specific gravity is greater than that of aluminium."—FEDOTIEFF and ILJINSKY, *Journal of the Society of Chemical Industry*, April 15, 1913, p. 367. (J. A. W.)

HEATING MILL SOLUTIONS. "Through the courtesy of the Liberty Bell Gold Mining Company, Telluride, Colo., we are able to present some interesting data regarding the effect of cold and warm solutions, particularly referring to amalgamation in cyanide solution. Metallurgical practice at the Liberty Bell mill includes stamping in cyanide solution, with subsequent amalgamation, after both stamps and tube mills. Prior to April, 1912, cyanide solution was used at ordinary temperature, say 60° F. Beginning with April, 1912, there was a marked increment in solution temperature, and a notable increase in percentage extraction of silver. The natural assumption was that the increased saving was due to better cyanide extraction. But upon tabulating the percentage recoveries from the different mill departments, some unexpected figures were obtained, which did not wholly confirm the first assumption. The following table gives typical percentages of recovery in the different departments over a period of one year, with an opportunity to compare the results obtained during the first half of that time, when cold solutions were used, with those of the last half when the solutions were warm.

A marked gain is seen in total silver recovery, amounting to 10·57 %. 40 % of this gain, however, is by amalgamation, and another 40 % by concentration, with only 20 % by cyanide.

Percentage Recovery.

Period.	Amal.	Cyan.	Conc.	Total.
	Gold			
First half ...	48·98	35·71	7·52	92·21
Second half ...	50·20	34·40	8·40	93·00
Gain second half	1·22		0·88	0·79
Loss second half	—	1·31	—	—
	Silver			
First half ...	7·12	36·67	17·24	62·03
Second half ...	11·30	39·70	21·60	72·60
Gain second half	4·18	2·03	4·36	10·57
Loss second half	—	—	—	—

Some laboratory tests were conducted to check the results obtained in the mill and to explain the distribution of the gain in silver recovery in the different departments. The following amalgamation tests with hot and cold pulp (barren solution) are suggestive.

Amalgamation Tests.

Test No.	Mg. Silver in Ore.	Mg. Silver Recovered in Amalgam. Hot 85°.	Mg. Silver Recovered in Amalgam. Cold 60°.	Per Cent Silver Extracted by Amalgamation. Hot 85°.	Per Cent Silver Extracted by Amalgamation. Cold 60°.
1	61·8	2·18	2·90	3·5	4·7
2	,,	5·41	1·65	8·7	2·7
3	,,	5·50	2·95	8·9	4·8
4	,,	2·68	1·58	4·3	2·5
5	,,	5·95	3·46	9·6	5·6
6	,,	4·22	3·94	6·8	6·4
7	,,	6·86	2·90	11·2	4·7
8	,,	9·62	3·16	15·5	5·1
9	,,	4·80	2·60	7·8	4·2
10	,,	7·96	4·47	12·8	7·2
11	,,	8·20	4·50	13·2	7·3
12	,,	8·16	3·70	13·2	6·0
13	,,	5·42	4·35	8·7	7·0
14	,,	5·59	5·62	9·0	9·0
Mean	61·8	5·86	3·42	9·45	5·52

The figures show nearly 90% increase in amalgamation at a temperature of 85° F. as compared with 60° F. Similarly the following agitation tests in cyanide solution show an increased extraction of both gold and silver at the higher temperature, although the gain is not so marked in the case of gold as with silver.

Agitation Tests.

Ore Assay: Au. 0·20; Ag. 2·00.

Test No.	Temperature 85°. Tail Assay. Au.	Tail Assay. Ag.	Extraction. Au.	Extraction. Ag.	Temperature 60°. Tail Assay. Au.	Tail Assay. Ag.	Extraction. Au.	Extraction. Ag.
1*	0·018	0·58	91·0	68·5	0·020	0·53	90·0	74·2
2	0·012	0·65	94·0	71·5	0·022	1·32	89·0	35·7
3	0·012	0·75	94·0	70·3	0·022	0·92	89·0	55·2
4	0·012	0·65	94·9	72·7	0·022	0·72	89·0	65·0

It will be noted from the above tabulated amalgamation tests that the increase in silver extraction with warm pulp is fairly consistent. The gold increase was more erratic. This lead to the suspicion that silver was being precipitated on and amalgamated with the mercury after first being taken into solution by cyanide. To confirm this view the following amalgamation tests were then made, using only mercury and zinc box head solution, with no ore. The same quantities of mercury solution were used as in the preceding tests with ore and barren solution, and the test was conducted for the same length of time, with the following results:—

Test No.	Mg. Silver Precipitated on and Amalgamated with Mercury. Hot 85°.	Mg. Silver Precipitated on and Amalgamated with Mercury. Cold 60°.
1	3·54	1·10
2	4·20	0·80
3	3·46	0·86

The increase in silver precipitated on and amalgamated with the mercury in the case of the warm solution has 8·1% of the dissolved silver in the solution.

The average silver recovery in mill amalgamation for the period October, 1911, to March, 1912, inclusive, when cold solutions were in use, was 7·69%. The same average for the following six months with warm solutions was 11·30%, showing a gain of 4·21%. The average silver content of the mill ore during the latter period was 2·45 oz. per ton. The apparent increased silver recovery by amalgamation during this period was, therefore, 0·103 oz. per ton of ore treated.

The average quantity of dissolved silver in battery solution during the latter period was 0·297 oz. per ton solution. Assuming 5 tons solution used in the batteries per ton of ore treated, and 8·1% more of its dissolved silver to have been precipitated and amalgamated on the plates than during the preceding six months, the increased quantity amalgamated amounts to 0·1202 oz. per ton of ore, or an apparent increased silver extraction of 4·91%.

The indications are, therefore, that the apparent increase in silver extraction by amalgamation during the last six months is in reality an increased recovery by cyanide, probably due, as shown by the accompanying agitation tests, to the increase in solution temperature. The total increase in silver recovery should, then, more properly be credited to cyaniding and concentration, including in the former item that increase which is shown in amalgamation.

The foregoing results seem to indicate that, at least in this instance, heating is beneficial, and that the processes of amalgamation, cyaniding and concentration are materially improved when performed in warm solutions. The reason for the gain in silver recovery by concentration probably is explained by the effect of heat on the settling process."—*Metallurgical and Chemical Engineering*, Feb., 1913, p. 98. (A. McA. J.)

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* Temperature on this cold test may have exceeded 60°.

STANDARD SCREENS FOR SCREEN ANALYSES.—"Great interest has been shown in recent years, and rightly so, in the importance of having standard sizes of screens, or sieves, in laboratories where sampling and screen-sizing has to be done. The need of such a standard is occasioned by the fact that without it the worker on mill or furnace products in one part of the country can not read the article written by the worker in another part of the country, and translate the figures obtained into the measures that he is used to in making his own experiments, unless he knows the sizes of grain that are dealt with in the experiments described.

Some efforts have been made in this direction up to the present time, and the writer will refer to two. The sieve scale adopted by Rittinger, in his work on ore dressing, started with 1 mm. and laid out the different sizes for the screen holes for sizes above 1 mm., by multiplying each lower size by 1·414, or the square root of 2, in order to obtain the size next larger; going up in this way until he reached the largest size that in all probability would be needed, namely 64 mm., which is about 2½ in. in size, and going down to sizes below 1 mm., by dividing the size of the hole in each screen by 1·414, in order to obtain the size of the hole in the screen next smaller, and so on down the series until the smallest possible size of screen hole was reached. In bringing out this sieve scale so far as we know Rittinger simply worked it out as a very desirable plan, but so far as we know no manufacturer took up the making of the screens, and no standard set of screens were to be had.

The second instance to which the writer will refer is the standard set of sizing screens adopted by a commission appointed by the Institution of Mining and Metallurgy of London, and presented at their 20th session 1910-11. This commission took the ground that miners, mill men, and metallurgists are so wedded to the use of the word 'mesh' as description of the size of a sieve that it was unwise to try to introduce any other mode of designation except 'mesh.' It is known to all who have given any thought at all to the matter that the word 'mesh' does not define the size of the sieve hole, and therefore the size of the product which the sieve makes; for example, the writer has before him the catalogue of a manufacturer of screens, and in it he sees that a 14 mesh screen, with the coarsest wire that is ever used by this company would have a hole 0·030 in. in dia., while the 14 mesh screen, with the finest wire that is ever put into that size of screen would have a hole of 0·061 in. in dia.; and there are sieves made by this manufacturer that have 14 different sizes of wire, ranging between the above two extremes: 14 mesh sieve therefore has 16 different sizes of hole according to the 16 different sizes of wire used. Therefore the writer who speaks of a 14 mesh sieve in his descriptive article may mean to his reader that he is talking about grains anywhere from 0·030 in. to 0·061 in., failing utterly to convey a scientific fact to his reader.

The commission, aforesaid, in making their screen set, known as the I. M. M. standard screens, fully realised this difficulty, and met it by arranging to use wires of the same diameter as the width of the holes. This gives the reader definite information as to the size of the grain that is meant when a particular mesh of the sieve is used. These screens, however, do not have a constant ratio between two adjacent sizes: in fact the ratio between the width of 100 mesh holes and 90 mesh holes is 1·095, while the ratio between 30 mesh and 20 mesh is 1·508. Other lesser variations occur all the way along the series, and in fact there is no attempt at a uniform ratio. There is another difficulty which this method runs against, and that is, that in order to have screens of uniform size of opening, we must use double-crimp wire cloth, and a prominent manufacturer of screens informs the writer that it is quite difficult to make double-crimp wire cloth where the wire is as large as the size of the holes, particularly when one gets to the coarser sizes, and if single-crimp wire cloth is used, the wires are not fixed, and the sizes of holes will vary to a ruinous extent by the slipping of the wires.

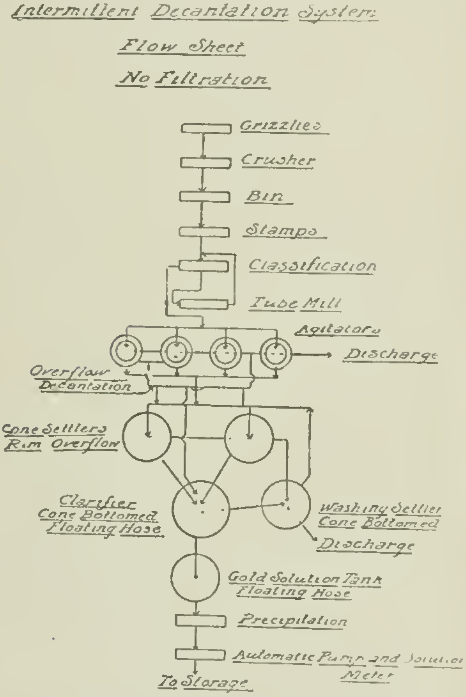
In regard to the use of a constant ratio between the sizes of holes of the adjacent sieves, the writer believes there are very strong reasons in favour of using a constant ratio. If one is making a series of sizing tests from a crushing machine, one wishes to know the proportional amount of the sizes at different points down the series, and will probably wish to draw a curve of the same. If one pair of sieves has a very small ratio of sizes of holes, and another has a very large ratio, then no regular statement of quantities of products made by the oversize of the successive sieves will give its full meaning, nor will it be possible to make a curve which will truly represent the result of that crushing operation. Also if the series of products obtained are to be subjected to any concentrating operation, with a view to ascertain the ability of the ore to be concentrated, there will be no fair comparison between two products, using the above illustration, where in one instance the coarsest grain in the product is 1·5 times the size of the finest, while in the second case the coarsest grain in the product is only 1·09 times the size of the finest grain in the product. The work of concentrating on two products differing so completely in their limits of size as those two products would differ, would fail entirely of being comparable one with the other. Again, the very fact of using 'mesh' as the standard of size of sieve invites the man who has not obtained the I. M. M. standard sieve to talk about his products by use of the word 'mesh,' when his products are made by sieves of different make, and therefore have different sizes of holes from the I. M. M. standard screens.

On these three accounts (1) the difficulty of making double-crimp wire cloth of such coarse wire; (2) the difficulty of drawing deductions from curves, and of comparing results; and (3) the difficulty of having others not provided with the I. M. M. standard entering into the discussion and using the word 'mesh' it seems to me that a standard set of screens, which is standardized according to the size of its hole, instead of according to the number of meshes, and that has a constant ratio from one end to the other, for example, Rittinger's ratio of 1·414 or the square root of 2, and having the liberty to use wires that are smaller than the size of the hole, admitting therefore of the use of the most perfect kind of double crimp wire cloth, cannot fail to have great advantages over the series of standard screens based upon mesh.

Such a set of standard screens has just now been prepared by W. S. Tyler Co. of Cleveland. This set of screens begins with 200 mesh as its starting point, which is a size of screen that has been standardized at Washington by the U. S. Government, and uses the ratio upwards from that point of 1·414, running all the way up to the maximum size. These screens are made with the greatest care in order that they shall be always standard in size of hole, and in order that the various experimenters in the different parts of the country who make screen analyses of their products can read each other's results, and know exactly what is meant by the results, and since the ratio between the width of the hole between the next small screen and the next larger is always the same, namely 1·414, a sizing test made by this set of screens will give the proportional quantity of material that belongs between each two sieves according to a perfectly definite law, that is to say the largest grain in any product will always be 1·414 times the size of the smallest grain in that product. As a consequence of this the series of products obtained will have their quantities, or weights, proportional to a perfectly definite law, with the variable ratio entirely eliminated. They will yield curves which truly represent those quantities according to that definite law, and if concentrating tests are desired upon these products, the results of concentrating will be comparable all the way down the line of products, ranging from the coarsest to the finest.

The writer warmly commends this effort to make a valuable standard set of screens for use in screen-sizing tests, and hopes that they will generally be adopted by the mining engineers, the mill men, and the metallurgists of the country, in order that the papers written on these various subjects may have the greatest uniformity in the description of the products entering into such discussions." ROBERT H. RICHARDS. *Mining and Engineering World*, Feb. 15, 1913, p. 341. (A. R.)

INTERMITTENT CYANIDING. "The tendency in the cyaniding of fine sand and slime is invariably to adopt the continuous system, as any stoppage for settling and decantation causes a clogging of the apparatus and pipes. On the other hand, the continuous system has a great drawback, due to the individual particles not receiving the same length of treatment. There is no guarantee that the particles will stay in the apparatus long enough, or that they will be discharged when they have parted with their gold. In fact, the heavier and coarser particles pass through in the shortest time; thus the part of the ore requiring the longest treatment gets the shortest, and vice-versa. The *Mining and Scientific Press* for February 8th reprints a paper published in the *Colorado School of Mines Magazine*, in which Leon P. Hills, manager of the United States cyanide plant at Tuolumne, California, describes a form of agitator, by the use of which the intermittent system of treatment is successfully conducted. By this method the fine particles receive a shorter treatment and the heavier and coarser the longer. The agitator is a modified Pachuca, the central fixed column being surrounded by a vertically adjustable column of a few inches greater diameter. This outer column, by means of a screw at the top of the vat, can be moved from a position where the lower end of the column is in contact with the sides of the cone, to any predetermined point above that, the upper extremity of the column always being below the surface of the solution. During the time agitating is suspended and settling is taking place, the outer column is in its lowest position, thus excluding the pulp from settling round the inner column and air jet. The air jet, on being started, will institute circulation of the solution down between the two columns and up through the central column, whereupon the outer column is raised, allowing the pulp to enter into the circulation. This agitator has never failed to get the pulp into a condition of suspension and without the aid of additional air.



In the system under consideration, a thickener to precede the agitators is not necessary. The agitator is provided with a baffle which gives a circumferential quiet zone when pulp is being run in and when agitating is taking place, the overflow going to cone settlers. When a charge is sufficiently thickened in the agitator, the pulp is switched to another agitator, and agitating started in the charged vat. During agitating, barren solution may be run in to lessen the gold contents and also to vitalise the solution in the vat. The extraction being completed, the outer column is screwed down, agitating suspended, pulp settled, solution decanted, and washes repeated as many times as is advisable, depending on the richness of the ore. The settlings from the settlers and clarifier are drawn off intermittently to the washing settler, or agitator, and washed."—*Mining Magazine*, March, 1913, p. 219. (A. R.)

EXTRACTING GOLD BY VOLATILISATION.—"In our December issue we made a short reference to the process adopted at the Gwalia Consolidated, in the East Murchison district, Western Australia, for extracting gold by volatilisation. Mr. Howe describes the process in the December issue of the *Monthly Journal* of the Chamber of Mines of Western Australia. It will be remembered that the Gwalia Consolidated ore is refractory, and that it is not of sufficiently high grade to stand the cost of roasting and cyaniding. Mr. Howe's process has the advantage of great simplicity and low working cost, and moreover an efficient extraction is obtained, usually from 92% to 94%.

The process consists in roasting the ore with a small percentage of salt, whereby the gold is volatilised. The fume is led through chambers and drenched with water, which dissolves the lime, arsenic, and iron salts, while the gold is left suspended in the solution as a black powder. The solution is pumped through a filterpress and returned to the chambers to be used again. It is not absolutely necessary to grind the ore very fine before roasting. Ore as coarse as 20 mesh has been successfully treated; but the finer the ore the quicker the volatilisation. Thus the same ore ground to pass a 100 mesh screen is treated in ten minutes, the 40 mesh in thirty minutes, and to 30 mesh in fifty minutes. It will, therefore, be found, in most cases, cheaper to grind the ore fine, than to keep it for a prolonged time in a furnace at a high temperature. The salt may be mixed with the ore dry, or the ore may be wet-crushed in salt water from the mines or salt lakes, and subsequently dried. If the ore is given a rough preliminary roast before the addition of salt, 1% to 2% is sufficient for most ores; but if mixed with the raw ore more salt, say, 3% to 4%, is necessary.

The furnace used is the simple brick-lined rotary type without ribs or other projections inside, with a constant feed at one end and a discharge at the other. The temperature required is about 1,000° C. (orange to yellow heat); the temperature, in fact, attained in a muffle-furnace during cupellation. A higher temperature than this sinters the ore. This is no disadvantage if it occurs at the discharge of the furnace, but if too high a temperature is carried back into the furnace the ore sinters at this stage and volatilisation ceases before the ore is ready for discharge. Sintering for even the last quarter of the furnace length only reduces the gold volatilised by about 5% to 7%. A low temperature is fatal to good results, as little action takes place below a bright red heat. It is advisable to use oil or gas firing in the furnace; in the first place because the temperature is more easily under control; and again, because the fume, under this mode of firing, is not contaminated with soot as it would be from a wood fire. Only a small draught, just sufficient to carry the fume through the furnace, is required. In this way the heat is well conserved within the furnace, and the great loss of heat carried into the flues, so characteristic of the many roasting furnaces, is avoided. A rabbled furnace of the reverberatory type is not suitable for this process. The water used in catching the fume becomes charged with sulphurous and a little hydrochloric acid, both of which are helpful in taking the salts of the base metals into solution. Owing to the constant circulation of water, no great heat is experienced in the absorption chambers, and wood can be used in their construction.

The laboratory tests to try the suitability of an ore to this treatment are simple. Two assay-tons of the finely-ground ore, of which the gold content is known, are mixed with 5% of salt. Either common salt or salt evaporated from mine waters may be used. It is advisable to commence experiments with 5% of salt and gradually work back to find the minimum amount of salt necessary for any particular ore. The mixture is spread evenly to cover a 4 in. or 5 in. roasting dish, and then placed at the back of the muffle already heated to a good cupellation temperature. It should remain at this temperature for half an hour without rabbling, or any further attention, and can then be removed, cooled, mixed with the usual fluxes, and assayed in the ordinary manner. The percentage of volatilisation of the gold is high, generally over 90%. The charge, on being removed from the muffle, should present the reddish appearance of roasted ore. It will probably have caked slightly by the action of the salt, but can be readily crushed with a spatula. Should it appear hard and sintered, the temperature has probably been too high, and a low volatilisation result will ensue. Care must, however, be taken to keep the muffle hot enough.

Mr. Howe records the origin of the discovery of the process. In 1910, A. T. Fry, a chemist and assayer at the mine, while experimenting with various processes of possible application, observed that the ore lost its gold by roasting with salt. This volatilisation was considered a serious disadvantage from the point of view of the object of the experiment, and it was not until Mr. Howe suggested that the 'loss' was really a 'gain' that advantage was taken of the discovery." *Mining Magazine*, March, 1913, p. 219. (A. R.)

ALLOYS OF COBALT WITH CHROMIUM AND OTHER METALS. "As in organic nature certain animals and vegetable forms have undergone modifications, and thus, as it were, fitted themselves to live in a new environment, so it has been found possible in certain instances to form new metallic combinations which are practically immune to the natural conditions that exist on the earth at the present time. A few of the metals, such as gold, silver and the metals of the platinum group, are found native, but the commoner heavy metals are nearly always found in the form of oxides or sulphides.

For a number of years I have made experiments with a view to finding certain metallic combinations or alloys which would not only resist oxidation and other harmful influences, but would also possess valuable physical properties, which would render them fit for special service. The first decisive step made in this direction was the discovery of an alloy of nickel and chromium in 1898. The properties of this alloy have already been described in a previous paper.* Immediately following the discovery of the nickel chromium alloy, I produced an alloy of cobalt and chromium, which has likewise been described in the reference given, but the range of proportion between the cobalt and chromium is so great, and the combination is modified to such a marked degree by the introduction of other substances, that I have felt justified in reading another paper on this subject, describing more fully my researches in this particular direction.

As early as 1907 and 1908, I made alloys or combinations of cobalt, chromium and tungsten; cobalt, chromium and molybdenum; and cobalt, chromium, tungsten and molybdenum. I have made alloys of cobalt and chromium containing zirconium, tantalum, thorium, titanium, vanadium, etc. I have also added to the cobalt-chromium alloys, the non-metallic elements, carbon, silicon and boron. Some exceedingly interesting results have been obtained from these various combinations, and while further investigation is necessary in order fully to determine their chemical and physical properties, a number of them have shown interesting economic possibilities.

The preliminary fusions were made in graphite crucibles by means of a furnace using natural gas. I was later obliged to use crucibles of a special composition, not only to avoid the contamination of the metal by carbon from the crucible, but also because they proved more reliable under long-continued heating than those made of graphite.

The metal tungsten alloys readily with chromium and cobalt in all proportions. When added in small quantities to the cobalt-chromium alloy, it seems to have little influence on the properties of the combination, but if the proportion rises to 2% or 3%, a notable effect is produced. Generally speaking, the cobalt chromium alloy becomes harder and more elastic, especially if it contains a small amount of carbon, boron or silicon.

The following experiment shows the effect of melting the alloy in a graphite crucible: 90 gm. of cobalt, 6·3 gm. of tungsten, 18 gm. of chromium, together with a small quantity of calcium silicide, were introduced into a graphite crucible. The resulting alloy was very hard, and the crucible much eroded on the inside. The bar could be slightly flattened at one end, and after being made into a cold-chisel, showed remarkable qualities. It would not only scratch glass, but also quartz crystal. It was quite tough at ordinary temperatures, and would cut small chips or shavings from a piece of stellite. At a bright yellow heat it showed signs of fusion, and became covered with a skin of oxide.

An alloy was made by melting the following in a special crucible: cobalt rondelles, 80 gm.; chromium, 20·8 gm.; tungsten, 7 gm.; calcium silicide, 10 gm.; calcium carbide, 5 gm. As soon as these materials were melted, the crucible cover was removed, and 15 gm. of an alloy of cobalt and boron was introduced. The crucible lid was then replaced and the heating continued. A heavy, thick slag formed, which was removed before pouring the metal. The resulting bar was very hard and elastic, but only drew slightly under the hammer, and then broke. A cold chisel made from the cast metal cut iron readily. The bar was broken up and remelted with about one-third of its weight of an alloy of cobalt, chromium, tungsten and carbon. The result was a fine-grain alloy which was very elastic, and would draw out to a considerable degree under the hammer without cracking. Its elastic limit must have been very high, since, when it broke, the pieces were thrown violently.

Taking the alloy of chromium and cobalt as a basis, and holding the proportion of chromium at 15% of the entire mixture, it was found that the alloy gradually increased in hardness with the percentage of tungsten. When the quantity of tungsten rises to 5%, the alloy becomes distinctly harder, particularly when forged under the hammer. When the tungsten reaches 10%, the metal still forges readily, and a tool formed from the alloy takes a fine cutting edge. This alloy is suitable for both cold chisels and wood-working tools. When the tungsten rises to 15%, the metal can still be forged, but great care is necessary in order to avoid cracking. This alloy is considerably harder than that containing 10% tungsten, and is excellent for cold chisels. When the tungsten rises to 20%, the alloy is still harder, and can be forged to a small extent. It makes good lathe tools for cutting steel and other metals at moderate speeds. When the tungsten rises to 25%, a very hard alloy results, which cannot be forged to any extent, but casts readily into bars which may be ground to a suitable form for lathe tools. These tools have shown great capabilities, particularly for the turning of steel, since they are very strong, and retain their hardness at speeds which almost instantly destroy the cutting edge of a steel tool. The tungsten may be still further increased to 40%, and the alloy will retain its cutting qualities, and for turning cast iron, this alloy answers even better than that containing 25%. When the tungsten reaches 40% or more, the alloy becomes so hard that it will not only scratch glass, but will readily scratch quartz crystal. A small drill made of this material drilled a hole through the wall of a glass bottle without the use of any liquid or other lubricant. A ⅝ in. square cast bar, when ground to a suitable edge, was set in a tool holder attached to lathe. The workman who had operated the lathe had been able to turn to form 26 cast iron wheels in ten hours with a steel tool of the same size. The stellite tool turned 49 of these wheels to form in the same time. The steel tool was ground 30 times during the operation, while the edge of the stellite tool was dressed slightly by a carborundum whetstone, after its day's work was completed. A set of steel cutters placed in the boring head of a cylinder boring machine, was able to bore from 27 to 28 holes in ten hours These cutters were replaced by others made of stellite, which performed the work in three hours and twenty minutes, or a little more than one-third the time. Not only was the speed of the mill doubled, but the feed also, and notwithstanding this severe ordeal, the stellite cutters were only slightly worn, while it would have been necessary to regrind the steel cutters at least two or three times for the same service at lower speed., Some remarkable results were obtained in the turning of steel on the lathe For example, a cylindrical bar of annealed nickel chrome steel, about 2·5 in. in diameter, was placed in a lathe and turned with a steel tool at about as high a speed as the steel would permit without 'burning.' The steel tool was then replaced by one of stellite, and the speed at the same time increased to 2⅔ its former speed. The stellite tool retained its edge under these severe conditions, and produced a shaving weighing 1·2 lb. in 30 seconds. Just what the effect of the alloy will be in machine-shop practice, is at present somewhat difficult to determine. In my opinion, however, it will not fully supersede high-speed steel in the machine shop, but in cases where rapid work is the consideration, it will doubtless replace high-speed steel.

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* *Journal of Industrial and Engineering Chemistry*, Vol. ii., No. 10, pp. 397 to 401 (Oct., 1910).

When molybdenum is added to a 15% cobalt-chromium alloy, the alloy rapidly hardens as the molybdenum content increases, until the content of the latter metal reaches 40%, when the alloy becomes exceedingly hard and brittle. It cuts keenly and deeply into glass, and scratches quartz crystals with ease. It takes a beautiful polish, which it retains under all conditions, and on account of its extreme hardness, its surface is not readily scratched. When 25% of molybdenum is added to a 15% chromium alloy, a fine-grained metal results, which scratches glass somewhat readily, and takes a strong, keen edge. Its colour and lustre are magnificent, and it will doubtless find a wide application for fine, hard cutlery It cannot be forged, but casts readily, and its melting point, is not abnormally high.

If carbon, boron, or silicon be added to any of the above alloys, the alloy becomes much harder, though the addition is not always desirable, since it tends to make the alloys more brittle.

If either tungsten or molybdenum be added to a cobalt-chromium alloy containing 25% of the latter metal, the hardness of the alloy is rapidly increased. When the percentage of tungsten, for example, reaches 5%, the alloy can be readily forged, and forms an excellent combination for wood-cutting tools, such as chisels, pocket-knives, etc. When molybdenum is added to the same mixture of chromium and cobalt, much the same effect is produced, though, generally speaking, a smaller quantity of molybdenum is required to produce a given increase in hardness. In some instances I have found it advisable to add both molybdenum and tungsten to the cobalt-chromium alloys. Generally, the colour and lustre of these alloys, after polishing, are brilliant, and they seem to resist atmospheric influences equally as well as the binary alloy of cobalt and chromium." ELWOOD HAYNES. — *Mining Science*, March 6, 1913, p. 153. (A. R.)

SILVER CYANIDING AT TONOPAH.—"The ores of the Tonopah district have not as yet been subjected to thorough tests in order to determine the advisability of sorting the ore before milling it. Some of the plants have adopted sorting in a desultory way and two have taken some pains to accomplish a more or less thorough sorting. These latter are the West End Consolidated and the new Belmont. The first of these has installed a mechanical washing machine which screens and washes the ore in hot water and then delivers it upon a wide sorting belt running at slow speed. This belt is so arranged that the sorters work on one side of it, throwing the waste not off the belt, but to one side of it. At the end of this belt there is a splitter which divides the stream delivery, the milling ore going to its proper bin while the waste is delivered into a waste bin from which it can be readily discarded.

There are several advantages about this sorting system. The ore to be sorted is passed over a trommel with small holes and is given a thorough rasping which disloges much of the sticking clay and mud and also gets rid of the fine portion of the ore which does not require sorting, but which goes directly into the mill. The coarser ore thus is passed through into the washer proper, which is a large steel cylinder set on an incline with its lower or receiving end set into a tank of water which is heated by steam. The inside of the cylinder contains a spiral, or reverse screw arrangement, which obliges the ore to ascend through the cylinder until it is finally discharged at the higher, or discharge end. Near the discharge end there is a section of the cylinder perforated with small holes through which the washing water returns to the tank below. The coarse ore delivered to the sorting belt is in perfect condition for sorting, as the operator can see at a glance its character and can eliminate all absolute waste without hesitation. The mud and clay washed from the ore are collected in the washing tank and can be recovered at convenient intervals. It is usually of high grade and is sent to the mill along with the regular milling ore.

The practical utility of this system has been thoroughly tested and calculated at the West End plant and the conclusion is that there is a clear and definite saving by its use. A large portion of rock that will not pay its way through the mill is discarded, saving the milling cost of putting it through the process and enriching the portion of payable ore. In other words, the same amount of silver is concentrated into a smaller bulk for milling, the economies being evident. The new Belmont mill has installed a similar machine for washing the ore.

Sorting in some form is practiced by several of the operating companies at Tonopah, but only at the above mentioned plants is it carried out in thorough form. It has been stated that a great many of the Tonopah dumps would pay well for sorting them over if it were possible to lift the material economically and get rid of the waste. If this statement is true, it seems likely that sorting while the ore was originally in motion might have repaid the trouble necessary to do it. It is a question which requires some study and it is impossible to make any statement which would certainly cover all cases.

Tonopah Ores Crushed in Solution. All the operating mills at Tonopah use solution throughout the mill operations. At the same time there are certain of the operators who believe that better metallurgical results could be obtained if it were possible to do the crushing and grinding in water without incurring a loss of cyanide and dissolved metal. It is evident that in treating silver ores where solution of comparatively high cyanide content must be used, crushing in water is not usually practicable. The amount of water taken in unavoidably from the grinding system to the cyanide plant would have to be discharged as moisture with the residues, and at that time would contain both cyanide and metal in solution, a combination that could not be thrown away. The only system by which it might be managed would be to filter the pulp from the grinding system, whereby it could be introduced into the cyanide plant with approximately the same quantity of water as the residues would contain in moisture. Whether the advantages obtainable would justify the additional expense is a question which would have to be solved in each particular case. The procedure is followed at the mill of the Smuggler-Union Mining Co., at Telluride, Colo., as has already been mentioned in these papers.

Crushing in Water at Goldfield. The advantage of crushing in water is believed to be a saving of cyanide by removing cyanicides from the ores, resulting also in an improved extraction. This matter has received much attention from the operators at the Goldfield Consolidated mill and experiments covering several months' work have convinced them that water crushing has decided advantages over crushing in solution. Naturally enough, the conditions at the Goldfield mill are decidedly different from those obtaining at Tonopah and the results of experiments cannot be transferred bodily. The former is treating a gold-bearing ore and using solutions much weaker than those necessary for the ores of Tonopah.

At the Goldfield Consolidated it has been satisfactorily proved that when crushing in water is followed the consumption of cyanide is less and the extraction of metal better than when crushing in solution. It is shown, also, that when crushing in solution is practised, there is likely to appear in the mill a light gelatinous product which refuses to settle and causes a great deal of trouble in mill operating. The substance appears much like an alumina product, which it may be, but that has not been proved. On the contrary, when water is used in the crushing system, this product does not appear.

It is extremely difficult to account for this difference and many theories have been invented in the attempt to do so. The theory has been advanced that no matter how alkaline a crushing solution may be, and all solutions containing cyanide must be kept alkaline, there will be a certain length of time, however short, when the acid liberated from the ore will overbalance any alkalinity immediately available at any one point, and that during this short time a chemical change might take place which would render soluble some elements which would not be put into solution with water and which would be removed by washing before reaching the cyanide department, were water used in crushing. This momentary acid condition might also introduce certain combinations prejudical to high extractions. Of course, there has been no research which would prove that this condition does exist and is responsible for the results shown, but the results are plain. Commercial operators rarely have the opportunity to solve questions of this kind, but it is just such problems which most need solving in the cyanide field.

Whether or not similar conditions to those found at Goldfield would obtain at Tonopah is not certain, but with this example in mind there are many who would like to have the opportunity of trying it.

Stamp Crushing Universal.—The use of gravity stamps for crushing is universal at Tonopah and it seems that nothing else has been considered in the design of the plants. This seems unfortunate, for the ore does not appear to be too hard to give good results with rolls or Chilian mills, either of which might be applied with economy. It is true, however, that stamps have been considered the conservative crushing machine for a long time and the design and construction of plants at Tonopah has been eminently conservative. To instal a machine which will surely do the work, even though at a cost somewhat higher than that of one which might be satisfactory, is a sound basis upon which to reason and designers can hardly be censured for following that system. At the same time the results obtained by the other machines, rolls and Chilian mills, both as to efficiency and cost, are no less definite and clearly proved, the obstacle to their use being simply that the results are not so well known. One cannot say with certainty that either of the machines above mentioned would satisfactorily take the place of stamps at Tonopah, but it is not too much to say that I believe either of them would introduce economies.

Varied Lining for Tube Mills.—The matter of tube mills at Tonopah introduces some interesting variations which are worthy of attention. First attention is called by the shortness of the mills generally used, especially in the later installations. This progression has apparently being along conservative lines and there is reason to believe that the theory is sound. Certainly it has resulted in a reduction of power for the same production.

The mills recently installed are generally 5 ft. dia. and 18 ft. long, in one case the length has been reduced to 16 ft. with satisfactory results. It will be interesting to watch this development and see what it brings forth. It is not impossible that the tube mills of the future may be more like drums of large diameter and short length than the long, narrow tubes first used.

The lining of the tube mills has given rise to much argument. The contrast here to Mexican practice is noteworthy. In all the Tonopah installations I have not seen a single instance of the use of the El Oro ribbed lining, but everywhere there is the use of the smooth cast lining and the silex blocks, both of which had been tried and discarded in Mexico before the El Oro lining came into use. The smooth cast lining would seem, theoretically, to be the most inefficient of all possible linings, as it would appear that the pebbles would be more than likely to slide over it and reduce the ore by rubbing it between the lining and the pebbles. The matter of the dropping or tumbling of pebbles would apparently be reduced to a minimum, and while it is clear that the power requirements would be reduced to the lowest possible point, apparently the efficiency would go down with it. That this is not the case is what several of the operators declare positively. They claim to get better results, both as to character of product, its quantity and cost of production. And it is also stated positively that the pebbles do not wear flat in the smooth-lined mills. It is a fact that if one is careful he can avoid seeing flat pebbles on the scrap heaps of these mills.

The silex lining has its advocates also, the installation at the new Belmont mill containing silex lining except in experimental cases. Silex has proved to be a good lining in many cases and except for the item of cost has done good work in Mexico. The objections there to its use are the cost and the time necessary to reline the mill when it wears out. This usually puts a tube out of commission for from five to seven days and where there is no reserve capacity it seriously interferes with the mill operation.

The Komata lining has had trials at Tonopah and in the case noted in the Tonopah Extension mill, is doing satisfactory work. Komata linings have also been installed at the Goldfield Consolidated with satisfactory results and are now used altogether. It is claimed that these linings increase the power consumption, which is undoubtedly true, but that the ultimate efficiency is increased. This is even more definitely shown when the revolutions of a mill into which they are placed are reduced considerably. For instance, a 5 × 18 ft. tube mill using smooth or silex lining and revolving at 26 or 27 r.p.m. can be reduced, when the Komata lining is used, to about 22 or 23 r.p.m., resulting in a lower power consumption and no great reduction of efficiency.

The height at which the load of pebbles is carried also varies a good deal. The variation is from somewhat below the centre line of the mill to 6 or 8 in. above it, calculating with the mill not running, of course. In cases where the load is carried above the centre line of the mill a grating has to be used in the discharge in order to prevent the issuing of pebbles. At the West End mill a difference of pebble load made an enormous difference in the efficiency of the tubes so that the whole mill was effected. This question of lining will probably be solved in the near future when there will be some agreement among the operators as to which type best suits their needs. At present there seems to be a great difference of opinion.

Many Types of Agitators. Opinions in regard to agitators seem to differ as widely as they do in tube-mill linings. There are examples of the Pachuca, Hendryx, Trent and other systems in use. The original mills of the district, the Desert and Belmont mills at Millers, used mechanical agitators for their slimes, the Hendryx type is in use at the Montana-Tonopah mill and several installations are using the Trent system, while the new Belmont mill at Tonopah is using the Pachuca system.

Particularly the Trent agitator seems to be a bone of contention. Some users say it is perfect and others declare it has no merit in it, that it is a nuisance and not satisfactory in any way. The truth lies somewhere between these two extremes, no doubt, but at what exact point is impossible to state.

The Hendryx agitator has been previously mentioned in these columns, but it is a machine which has never become popular on account of its high power consumption. There is no doubt that it is a good agitator, but it will probably never become a serious factor in the agitation of slime for the reason of its high operating cost.

Pachuca vats have been the subject of investigation for some time and many operators claim that they offer no advantages in economy nor in additional extraction obtained.

Referring back to the new Nipissing mill at Cobalt, in which the slime agitators are the old-style mechanically moved arms, the designers stated that they had been able to find no device which would assure them either better results or lower costs. In such event the Pachua vat would be entirely out of the reckoning, for if it presents no such advantages, its high cost of installation and the additional cost of pumping in a mill, occasioned by the height of the vats, would certainly preclude its installation. I am of the opinion that the Pachuca vat is likely to be replaced by some other more satisfactory design.

Concentration Practised in Some Mills.—It will be noticed that some of the Tonopah mills practice concentration while others do not. The two at Millers, the Montana-Tonopah, the West End and the New Belmont practice concentration, while the Tonopah Extension and the MacNamara mills do not. In those mills which do concentrate, the object has been to make as little concentrate as possible consistent with removing the objectionable elements from the ore. With low or medium-grade ores concentration is not necessary but with the rise in silver content, attendant upon the increase in the proportion of sulphides, concentration must be resorted to. Many of the mills using the process have reduced the number of tables, the point being that it is more expensive to market concentrate than bullion; therefore as large a proportion as possible should be taken out in bullion form. This is sound reasoning, particularly where the concentrate has to be shipped to smelters where losses may be multiplied by many means.

There is no instance of local treatment of concentrate at Tonopah. In this connection it is interesting to review the situation at Goldfield, where the Consolidated treats all of its concentrate by an elaborate process. Experiments on concentrate showed that by treating the concentrate raw a certain percentage could be saved; that by first roasting the concentrate and then cyaniding it, a better extraction would result, but that by treating it raw, then roasting it and retreating by cyanide, an extraction about equal to the sum of the results of the other two methods could be obtained. That system has been installed and is followed with success. After the cyaniding of the roasted concentrate, the residue is delivered into the vat which feeds the filter and is discharged with the filtered tailing. By this means the concentrate tailing is reduced to a value nearly equal to that of the regular mill tailing. The process is interesting and unusual.

Heated Solutions Used.—Operators universally agree that the extraction of silver from the Tonopah ores is increased by heating the solutions. The point to which this is carried is generally about 120°.

There seems to be no difference of opinion on this point and all the mills are doing this with good results. The question naturally occurs as to whether the Mexican operators have been overlooking anything in the matter of heating solutions. I have seen the scheme tried several times and made experiments on it, but without any beneficial result. That this is not unusual will be shown in a later paper, when I will call attention to another silver mill which has made repeated experiments and found no increase of extraction through heating the solutions. Still, this is a point which ought to be worked out for each particular ore, no general rule governing it.

Heating solutions seems to have no effect whatever on the extraction of gold. This means, of course, solutions of normal temperature, as extraction of both metals falls off in abnormally cold solutions.

The use of lime and lead acetate at Tonopah does not vary widely from accepted practice at most other camps. The addition of these materials varies, and the practice is governed principally by convenience. The filters in use at Tonopah are all of the stationary vacuum type. The present condition of the filter problem makes it advisable for discussion to be postponed until there shall be more liberty to publish statements.

Calculation of Extraction.—At Tonopah, as in most of the mills at the present time, the calculation of extraction is performed by using the content of bullion produced plus the content of tailing discharged as the value of head samples. Some of the older operators look with envy on the mill men of the present day and think of the monthly recurrence of attacks of heart disease that came with the attempt to reconcile production and tailing content with head samples taken at or before the batteries. Extractions of 120% were not unusual in those days, nor were drops to 60% and 70% unexpected, but the mill man was continually in hot water about it, particularly if he differed from the general manager about the accuracy of the head samples. This later method is much simpler and tends to avoid worries. At the same time it is hard to agree with a method of solving a problem by means of which any answer is the right one. I am anxious to have this matter discussed and hope that there will be someone kind enough to take it up and extract opinions about it." H. A. MEGRAW, *Engineering and Mining Journal*, p. 503. (H. A. W.)

CHEMICAL EXTRACTION OF TIN FROM THE ORE.—"Whilst awaiting further returns from Mr. Arthur Richards' volatilisation process, Mr. Henry Bailey's neutralisation of tin containing colloids, or Mr. Gee's centrifugal separation, it is well to know that these do not exhaust the inventor's projects, but that another process has been evolved which appears, from the results at present to hand, to, at all events, be worthy of further investigation. It does not only recover 98% of tin contents as copper precipitate or oxide of iron respectively. Mr. James W. Chenhall, who for the past 42 years has devoted his life to metallurgical pursuits, for the last two or three years has been interesting himself in working out a chemical method for the extraction of the tin contents of an ore. His process necessitates three operations—viz., roasting, leaching and precipitation. The roasting of the ore (which has previously been reduced to the ordinary mesh coming from a battery), to which has been added a certain proportion of carbonaceous matter, is undertaken in an enclosed furnace heated to about 1,200° F., the resulting gases being passed and repassed through the mass until all the oxygen in the ore and contents has been absorbed. The resulting porous mass is now thoroughly leached in an air-tight receptacle by means of a prepared solution, which dissolves the tin, iron, or copper present. These three products are afterwards precipitated individually from this solution, the tin coming down as pure tin oxide, the copper as copper precipitate, and the iron as a sulphate or oxide of iron. Mr. Chenhall, who claims a 98% recovery of the tin content of the ore, states that the entire cost of the treatment would be more than met by the value of the by-products, and I am given to understand that certain managers of mines in the mining division have sent, or are sending, their own ore for treatment in Mr. Chenhall's experimental plant at his laboratory at Totnes, where it will be dealt with under their own supervision. The effect of this treatment on those ores, containing wolfram, of which several of the mines have considerable quantities disseminated more or less throughout, is somewhat anxiously looked for, and arrangements are, I am given to understand, being made for an early test of this class of ore. If the wolfram remains in the residue as a product amenable to the influence of the magnetic separator its final recovery may readily be assured, or possibly some chemical means may be evolved for securing the tungstic acid (WO_3) direct, which will add considerably to the value of the process, at all events as far as wolfram-bearing ores are concerned."—*London Mining Journal*, March 22, 1913, p. 281. (A. R.)

PRINCIPLES OF ORE SAMPLING.—"The problem of sampling the ores that come to the smelter yard differs in no way from that encountered at the mine and mill, except that there is more of it. The materials which go to feed the blast furnances may vary from barren limestone, as uniform as a solid substance can be, to fluffy quartz stuff carrying free gold in particles weighing a grain or more. All these materials must be sampled and assayed so as to secure a basis for fair settlement without loss to the smelter and therefore, that sampling has received more attention with satisfaction to the shipper. It is not strange, therefore, that sampling has received more attention from the smelter man than from the operator of mine or mill. It is a serious matter for a custom smelter to get a reputation for low assays and unreliable sampling.

Not a little of the suspicion with which miners are apt to regard the smelter is due to the natural difficulties of securing a fair sample, giving rise to errors entirely beyond control. In the nature of things it is impossible to guarantee an accurate sample, even when the utmost care is exercised and the most elaborate methods are employed. Few realise the difficulties of the problem. A lot of ore may contain any of the minerals in the catalogue, and may include ingredients varying in size from lumps as large as a man's head to the finest dust, and in specific gravity from that of talc to the weight of galena or even gold. In spite of the fact that these heterogeneous materials may present the physical characteristics of hard angular crystals mixed with soft, sticky clay, the sampler is expected to abstract from the mixture a small portion which shall contain all the ingredients in the original proportions.

The Theory of Sampling.—The difficulty of accomplishing this result is shown rather forcefully in the classic example of a gold coin lost in a ton of barren quartz. Suppose this lot to be sampled by any of the usual methods. Before the first division it will not be crushed smaller than the dimensions of the gold piece, and the division will be manifestly unfair whether a half, a fifth or a tenth is taken for the sample. A grab sample or a handful taken at random would contain all the value or, more probably, none at all.

This is, of course, an extreme case, but a lot of unknown ore may present all the difficulties of crushing, mixing and subdivision found in this suppositious case. As a matter of fact most ores are much nearer the other extreme, that of a single chemical compound in which every particle is precisely like every other particle To the sampler all lots of ore fall somewhere between these two extremes—a mixture of barren dirt carrying a single nugget of value, and a quantity of fine hematite or limestone as uniform as sea water.

A sample of ore is a portion so selected from a lot and so prepared that any small portion taken from it for assay shall fairly represent the whole; that is, contain all the original ingredients in exactly the original proportions. To fulfil these conditions the lot must be made sufficiently uniform that in any portion abstracted a particle more or less of any given ingredient will make no difference. The simplest way to secure uniformity would be to grind the whole mass into fine particles and to impart a varied motion among the particles until, according to a useful law of change, a thorough distribution had been made. In practice this is, however, impracticable. The expense would be prohibitive and the fineness of the ore would interfere with subsequent treatment. The common practice is, therefore, to subdivide and to reduce the size of the particles in a series of operations.

The chance of error depends upon the weight of the richer particles as compared with that of the entire lot. If a large fraction is taken for the first cut, there is comparatively little chance that a lump more or less of the richer ingredient will make any difference even though the material is rather coarse. When the fraction removed becomes smaller the size of the particles must also be reduced, otherwise the chance of error would increase with each step. It is customary at each successive stage to reduce the diameter of the coarser particles one-half, thus reducing the volume to one-eighth.

While the chances of error are reduced by successive crushing and by removing a relatively large fraction at each stage of the operation, it must not be supposed that the chances of error can be entirely eliminated. In the nature of things many factors in sampling must be left to chance. Chance governs the distribution of the particles and chance determines whether a given crystal of rich material shall find its way into the sample or into the reject. By repeating the operation a sufficient number of times the chance of error is further reduced, but repetition alone cannot insure an accurate sample. As a matter of fact this action of the laws of chance is far more reliable in making a fair distribution and division than human judgment, and it is, therefore, sought to remove the human factor as completely as possible, but the chance of error will always remain.

Crushing and Mixing. There is now much less likelihood of error due to insufficient crushing than formerly. Crushing machinery, especially rolls, has been so much improved that to secure the desired size of the coarsest particle is a simple matter. The main thing is to choose a size small enough for all ordinary cases, for it is impossible to study each lot of ore received and to vary the process to any extent to suit the needs of any particular lot.

The difficulty of securing a thorough distribution of the particles increases in a measure with the size of the lot. The mechanical difficulty of effecting a thorough distribution of the particles in a large volume of material makes it necessary to rely very largely upon repetition to secure a fair division. Instead of taking a large average portion of the whole, an average is taken of a large number of small portions with the hope that it will be the same. It is safe to depend upon repetition only when the size of the particles is relatively very small compared with the whole.

No method of mixing large lots by mechanical means has ever been employed in sampling. Shaking trays operate upon only a comparatively few pounds of material at a time, and the revolving cylinder mixer can handle at most only hundreds of pounds and not thousands. For mixing large volumes of ore as a whole, the only method available is the old Cornish process of coning. This process which consists of allowing the material to fall upon a single spot and to distribute itself by gravity upon the sides of the resulting conical heap, should be effective when the particles are small and fairly uniform. Unfortunately ore particles are never uniform, and the larger lumps having a greater momentum cause more or less segregation, thus defeating the object of the process. There are, besides, mechanical difficulties in forming a perfect cone, and chance for controlling the distribution of the rich and poor ore by 'drawing the centre,' which have caused it to be regarded with disfavour.

The method is, however, commonly used in connection with the quartering process of division in which an absolute distribution is not essential, a fair lateral distribution being sufficient to suit the conditions of division by vertical planes. As the size of lots increases and mechanical handling becomes more necessary, the tendency is more and more to substitute repetition and a multiplicity of cuts for coning in sampling, and for that matter for mixing of any kind.

Revolving drum mixers operating upon the principles of the concrete mixing machine would seem to be more effective in removing the segregation caused by motion in mechanical handling. Concrete is certainly mixed very satisfactorily by this means. It must, however, be borne in mind that this device employs a free fall to effect the distribution, a principle frequently used in ore dressing to induce segregation. Concrete differs from ore in that it has a binding material and that its particles are more or less uniform.

The old Cornish method of sampling by quartering—really halving—is still as satisfactory a process for securing a fair portion for assay as has ever been devised. It is the result of years of experience and friction between miner and purchaser. It removes the largest fraction possible and frankly meets conditions which are imperfect at best. Its chief disadvantages are its expense and the necessary intervention of human hands. It requires great care to remove the quarters cleanly and sharply. It is hard to teach men the importance of close attention to details. To the average labourer it makes no difference whether the fine material on the floor is swept against a pile or is picked up carefully on a shovel and placed on the top of the heap; yet it is upon such small things that accuracy in quartering depends.

Friends of the mechanical samplers now in use profess to have no faith in quartering; yet there is scarcely a sampling mill in the country in which quartering is not practiced at some stage of the process. If it is unreliable for large lots, why is it employed almost universally in the bucking room to cut down the final sample? At this final stage every error is magnified to many times the importance of an inaccuracy in the earlier stages.

Fractional selection, setting aside a tenth, fifth or any other proportional shovel, barrow or cartload of material for a sample, is only another way of substituting many repeated cuts for thorough mixing. It is usually done rather roughly and is not regarded with great favour. Still it is employed almost universally when very large lots are to be cut down to a less unwieldy bulk. Nor is there any apparent reason why it should not give accurate results when carefully carried out. All the mechanical samplers in use employ precisely the same principle with the added disadvantage that they all effect the division while the material is in motion. Fractional selection will always have a place in the economy of sampling, and deserves closer study than it has hitherto received.

The split shovel is a device which is much used and abused. It is the working element of the machine sampler operated by hand and has all the disadvantages of the machine and few of the advantages of other hand methods. It removes a portion by introducing a pair of cutting edges into a falling stream of the material, without much regard for the proportion actually removed. Its cutting edges introduce an element of uncertainty which it, however, shares with all sampling machines now in use.

Machine Samplers.—The best results in hand sampling are obtained when the operations are performed in the most mechanical manner, when human judgment interferes the least with the operation of the laws of chance. Hand work also introduces possibilities for discrimination and fraud. The substitution of mechanical agencies for the hands of men therefore seems to offer a chance to place the disagreeable process of sampling upon an ideal basis.

The first experimenter to put his ideas into practical operation hit upon the falling stream principle, and none of his successors have as yet departed from it. The pipe sampler, which was installed in one of the smelters of the West about twenty years ago, proved a dismal failure. It introduced vertical blades into a stream of falling ore, cutting it into quarters in the approved manner, taking two for the sample and rejected the others. The failure of this device was soon traced to the fact that if a heap of ore is not uniform a stream of ore is still less so. The falling ore striking the sides of the pipe and rebounding from the cutting edges, separated itself into rich and poor portions, the fines as likely as not falling into one bin, while the coarse lumps came to rest in another. Ore particles are not uniform, and so a stream of particles cannot be treated as such.

The successor to the pipe sampler is not so simple and inexpensive as the first device. It has some departures, but it retains the cutting edges and it retains the falling stream principle. The introduction of a scoop to take the entire stream for a part of the time probably suggested itself at once when it was discovered that a stream of falling ore is not uniform, and many of the resulting machines have been tried and a few are still in use.

Dissatisfaction with the results of the revolving scoop machine led to many innovations and modifications. A careful study of the falling stream showed that it was very far from uniform, and no pains have been spared to slice the descending ribbon cleanly and accurately. Several of these machines are giving complete satisfaction to the owners, but many shippers and not a few smelter men still prefer the old quartering method with all its failings and its expense.

Objections of the Falling Stream Principle.—The inventors who employ the falling stream principle in the construction of sampling machines seem to have assumed that while a stream is not uniform throughout its horizontal section, it must be so in every other respect. No account seems to have been taken of the fact that the particles may vary vertically as well as laterally, and that they move at different velocities because of varying size and specific gravity. 'Time samplers,' so called because they take the entire stream for a definite portion of the time, are at present regarded as the acme of perfection. Yet the character of the ore stream varies markedly every moment, even with the best feeding devices, as a careful observation will show.

But the assumption of uniformity where no uniformity can exist, is not the worst fault of the modern sampling machine. The great number of cuts and the fact that nature has done most of the work in sampling most ores produces a fairly good average after all. In sampling, good enough must rule, since accuracy is impossible and a fairly close approximation is extremely expensive. The worst fault of the mechanical sampler lies in the retention of the vertical cutting edges of the old discarded pipe. The use of the cutting edges assumes uniformity in size and speed in the falling particles. Theoretically the cutting edge should be mathematical planes and the particles which they are to divide should be mathematical points. In practice the edges interfere very materially with the free fall of the particles. The chance of a lump's striking the edge depends upon its dimensions; it varies directly as the diameter of the particle. A lump one inch in diameter has twice as much chance of an interrupted fall as one half that diameter in size.

The law of averages has been depended upon to compensate for this error also. It is assumed that the shape of the lumps will be about the same, and that they will fall on the side of the edge which happens to cut off the greatest mass. But as a matter of fact, the particles do not hesitate upon the edge and gently slide into their proper receptacles. They strike the edges carrying all the momentum of their velocity and mass, and this again is greater in the case of large lumps. The cutting edges introduce an element which discriminates against the larger lumps. The impact throws the particles out of their predetermined course, and chance and average cannot be depended upon to return them to their appointed places.

Fortunately nature's distribution of the value throughout an ore makes a passable sample possible in spite of these obstacles; otherwise the development of mechanical sampling would have been more rapid than it has been thus far. Vezin, the inventor of one of the best of the modern samplers, realized the possibility of error introduced by the impact of the particles upon the scoop edges, and would probably have remedied the fault had he had an opportunity to do so. He tried to regulate the speed of the particles by adjusting the inclination of the feed spout, hoping apparently to reduce it to that of the revolving scoop. The varying size and mass of the moving particles rendered this adjustment difficult, if not impossible. Others have ignored this defect and have put the blame for unsatisfactory work on the assayer, on battered edges, on ignorance and carelessness in the sampling room, or upon some other factor not in a position to reply with force.

There is good reason for the popularity of the falling stream principle among sampler manufacturers. It is extremely simple and inexpensive and offers an opportunity to make sampling a mere incident in the handling of the ore. The material passes through one or more spouts anyhow, and what could be more simple and economical than to introduce the sampling device somewhere between the car and the bin? Methods of splitting and slicing the stream suggest themselves automatically; the wonder is that there are not more varieties of stream deflectors on the market.

Economy and simplicity are the only arguments in favour of the falling stream principle in sampling. The greatest difficulty to be met in securing a fair sample of a lot of the ore is a lack of uniformity in the distribution of the value. All the processes of mixing, crushing and repetition of cuts have as their sole object the removal of this obstacle. Segregation is the nightmare of the careful sampler. Yet all of the mechanical samplers now in use employ methods adopted by the ore dresser to produce segregation and destroy uniformity. Motion with free fall and free settling are among the most effective factors known in causing separation of the coarse particles from the fine and the heavy from the light. The modern sampling machine is extremely inexpensive, and sampling at best must be a compromise between what can be and what ought to be, but the last word has not yet been said about mechanical sampling.

The Ideal Sampling Machine.—The difficulties of getting a really accurate sample out of a lot of ore are many. At best the most important factors must be left to chance, and under the most favourable conditions no one can guarantee accuracy. The best process is merely a series of approximations, the degree to which the approximation can be carried being governed by the expense and various other conditions. Sampling, like bookkeeping and assaying, is a non-productive operation. It is considered as a necessary evil and a nuisance, and will never be an operating engineer's pet. Cost is the main thing, and such concessions as are made in the interests of accuracy are given grudgingly and under compulsion. The modern sampling machine does reduce costs and it does give a fair measure of satisfaction; therefore it probably deserves to flourish. Yet it is not unreasonable to ask for a device that shall transgress fewer of the laws of correct sampling, at the same time not materially increasing the expense.

The ideal automatic sampler will remove the cut while the material is at rest, thus introducing no factor favourable to segregation. Since perfect mixing, especially in large lots is impossible, it will employ a multiplicity of cuts, averaging being the best substitute for uniformity. It will permit of crushing and mixing between cuts with the minimum of motion, and will employ no shaking-tray segregation device. It will be as compact, simple and easy to clean as those now in use, and its work will be incidental to the transportation of material. None of these conditions should be beyond the powers of a good mechanical engineer. As soon as the demand for greater accuracy can be heard above the cries for reduction of costs the machine will no doubt be forthcoming."—A. VAN ZWALUWENBURG, *Mines and Methods.—Mexican Mining Journal*, March, 1913, p. 130. (A. R.)

PRECIPITATION TEMPERATURES.—"My attention has been called recently to the effect of temperature of pregnant cyanide solutions upon the success or degree of precipitation. While the effect of temperature of solutions upon their dissolving properties has been thoroughly investigated and recorded, I have been unable to find anything satisfactory concerning temperature in its effect upon precipitation and I trust that this subject is of sufficient merit to find a place under your department of correspondence and discussion.

Of the two specific cases that have come to my notice, one was in a plant precipitating a low grade gold bearing solution by means of zinc dust, using the Merrill press, and the other in a plant precipitating a fairly high grade silver bearing solution by the zinc shaving method. In both cases it was found that not only was the extent of precipitation lessened, but in some instances the tail, or so called barren sample, was higher in value than the head or pregnant sample, indicating redissolution of precipitate. These conditions were noticed when the solution to be precipitated fell below 50° F. in temperature.

In plants where the solution is kept at summer warmth at all times in order to benefit its dissolving action, the matter will be of little importance for the reason that all solution in the mill will have a temperature higher than 50° F., but in plants where no such attempt is made it becomes of considerable interest in that the 'barren' solution must be again precipitated with a resultant increase in zinc consumption for a zinc dust plant and a loss of time and labour for either a zinc dust plant or a zinc shaving plant.

The reason for the decrease in amount of precipitation would seem to be that the chemical reactions which take place are retarded by low temperature, but this will not explain why some of the precipitate is picked up by the solution during its contact with the zinc and precipitate unless the electro-motive order of the elements in cyanide solution is disturbed by change of temperature. An interesting question in this connection is, 'under what conditions can all of the precipitate be redissolved by cyanide solution?' It has been suggested here that even if no zinc were added to that already with the precipitate, the sulphides present in the precipitate would immediately throw down what precipitate might be dissolved."—E. H. SMITH, *Engineering and Mining Journal*, March 1913, p. 579. (H. A. W.)

SELECTION AND OPERATION OF TUBE MILLS.—"Tube mills have been used in cyanidation and general milling work on this continent for the last 10 years. However, a somewhat extended observation of tube mills in operation has disclosed a singular lack of judgment, in many cases, in the selection of tube mills and, in not a few instances, the machines have been operated inappropriately for the work to be done.

The main causes of criticism may be enumerated as follows: (1) Inappropriate dimensions. (2) Undermotored. (3) Defects in pebble size and load and (4) in one case a set of tube mills was seen crushing to cyanide in a manner which should only be used for concentration. Indeed when one considers the abuses to which tube mills have been subjected the merits of those machines stand out in bold relief.

The selection of a tube mill with suitable dimensions is of great importance. Too frequently the mechanical engineer designing the mill has the predominating voice in the choice of metallurgical machinery. The mechanical engineer is too apt to place low-erection costs and high-power efficiency above low-operating expense and high metallurgical efficiency. Thus the spectacle of the metallurgist struggling to obtain high savings and low costs with metallurgically inefficient machines, is unfortunately not an uncommon one. One case within the writer's personal knowledge illustrates the point. The mechanical engineer had facts and figures to prove that the first cost and power consumption of short tube mills was less, for a given tonnage output, than for longer mills. The extra cost of foundations, excavations, motors and belting for the longer mills amounted to a considerable sum. The general manager, naturally desiring to keep the cost of the mill down to the minimum figure, was readily convinced of the commercial economy of the short mill, and privately entertained the idea that the metallurgist was not a man of business ideas. The short mill was chosen, when clearly from all metallurgical principles the longer mills should have been installed. It is a curious commentary on human nature to say that the G.M. later on expected the metallurgist to do the work of real sliming with a mill at least 6 ft. too short. Experienced men need no diagram to see the end of the episode.

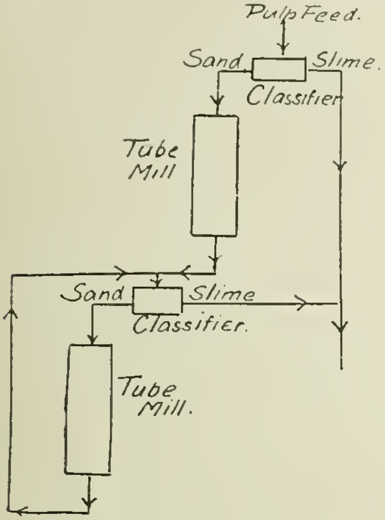


Fig. 1. Diagram illustrating Tube Mill Grinding.

Diameter of the Mill.—When a tube mill of large diameter is selected for the purpose of grinding material already crushed fairly fine, there is a waste of power. For such work a tube of small diameter and small pebbles should be used. For example, a tube grinding tailings from amalgamation should have a diameter of not more than 48 ins. measured inside the liners. For coarse-roll product or stamp-mill of 6 mesh, a tube of not less than 54 ins. internal diameter should be selected. Very good work is done in Western Australia by mills of 43 ins. inside diameter grinding 30-mesh stamp-mill product.

In cataloguing tube mills the manufacturers refer to the internal diameter of the unlined shell. All metallurgical data refer to the internal diameter and length of a newly-lined shell. As wear proceeds the diameter increases, and hence a greater consumption of power and a heavier load of pebbles are required.

Length of Mill.—It is a significant fact that in the all-sliming process mills most of the new plants have long tubes. Short tubes do well in concentrating plants where a fine granular product is required. I think I am right in saying that there are comparatively few tube mills in Mexico less than 22 ft. in length. The all-sliming process requires the sulphides to be reduced to an impalpable powder in order that the extraction by cyanide should be complete in a reasonable length of time. In such processes, also, the pulp sent to the agitators should be so fine that 85% at least should pass through a standard 200-mesh screen. This character of work is done neatly, rapidly and efficiently by a 22-ft. mill grinding a 6-mesh stamp product. Anyone who endeavours to do this work with a 16-ft. mill will have a heart-breaking job.

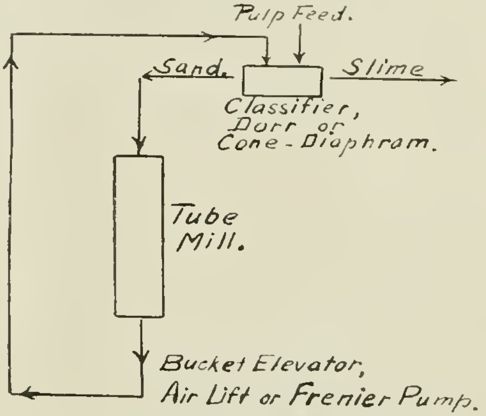

Pulp Feed.
Sand.
Slime
Classifier, Dorr or Cone-Diaphram.
Tube Mill.
Bucket Elevator, Air Lift or Frenier Pump.

Fig. 2. The Ordinary or Closed-circuit in Tube Milling.

Tube mills have been used of late in concentration practice. In Australian practice (Broken Hill) tubes 4ft. × 13ft. are used, grinding a 10-mesh feed so that 10% will remain on a 40-mesh screen. The product is quite granular. The Hardinge conical mill is doing good work in concentration mills in in this country and Mexico. Such mills are all comparatively short.

The length of the tube selected also depends upon the coarseness of the feed. A ¼-in. feed (roll crushed) will require about 2 ft. greater length of tube than a 6-mesh stamp product.

It is suggested that more efficient grinding could be accomplished by placing tube mills in tandem with a classifier between them. Theoretically the suggestion is sound. However, in an important Mexican mill, absolutely no advantage has been developed by the system, although there does not appear to be any disadvantage. It is a 'stand off.' Still this case should not be accepted as a final proof of the fallacy of the theory of secondary grinding, since the primary and secondary tubes in the mill referred to are of the same dimensions. According to the theory the secondary tubes should have smaller diameters and smaller pebbles than the primary tubes which receive the coarse stamp product. (See Fig. 1.)

Tube Mill Circuit.—The arrangement of the tube-mill circuit depends upon the work to be done; for all-sliming the circuit is closed; that is to say the material is continuously passed through the tube mill until discharged by the classifier. In this system the tube mill and classifier acts as a concentrator; the sulphides accumulate in the system until ground so fine as to float off. An ore containing only 2·6 of concentrates gave at the discharge end of a tube mill a product containing 5·3% concentrates. However, the tube discharge gave concentrates of lower value than the classifier discharge, due (1) to the rich, soft sulphides being rapidly reduced to particles sufficiently fine as to float off in the classifier discharge and (2) the hard, tough, low-grade sulphides accumulating in the system until crushed very fine. This condition is extremely favourable to high extractions in the all-sliming process.

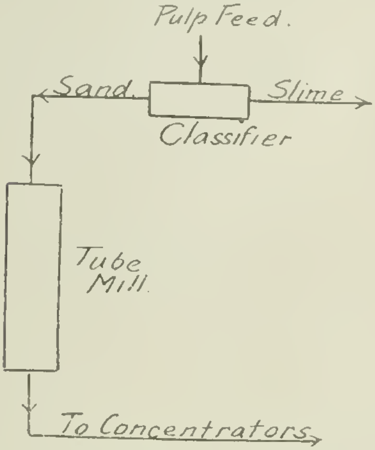


Fig. 3. The Open Circuit Tube Milling.

In concentration work it is desirable to keep the sulphides as coarse as possible, and therefore in such cases the tube mill product is at once removed from the system. This is the 'open' circuit.

For all-sliming cyanide work the open-concentration circuit cannot be used. The requirements of cyanidation and concentration are diametrically opposed. In a case that came under my observation, and in which I made some tests, the use of an open circuit increased the value of the tailings by 10 cts. per ton after cyanidation, as compared with the closed circuit.

Any arrangement whereby the sulphides are kept as long as possible in the tube-mill circuit, acts advantageously on the extraction in all sliming processes. It has been proposed and successfully introduced in Waihi, New Zealand, to give the pulp a rough concentration before agitation, and return the concentrates to the tubes, thereby ensuring a very thorough grinding of the sulphides.

As classifiers for tube mill circuits the Dorr and the cone-diaphragm are now almost universally used. The spitzkasten is becoming obsolete.

Tube Mill Power.—In a number of tube mill plants erected during the last two or three years the mills as erected and operated were furnished with motors of insufficient power:

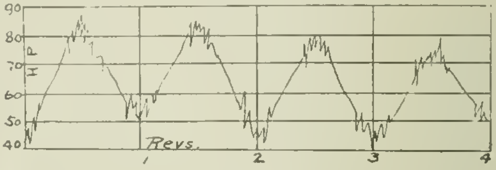
Mill No.			Motors installed.	Power consumed
1	..	...	90 h.p.	114
2	..	...	75 h.p.	96
3	...	...	50 h.p.	64

The power required by the tubes was apparently estimated by the old rule given by Davidson, viz.: Hp. = 0·15 × cubic contents in feet. However, since Davidson formulated the rule, the pebble loads of tube mills have been increased 20% and the pulp feed has been thickened. Moreover, ribbed linings of the El Oro type are now commonly used instead of smooth cast-iron or silex brick. All of which modifications have had the effect of raising the power consumption of tube mills. Investigation in three different plants would indicate that the power consumed is given very closely by the formula.

Hp. = 0·25 × cubic contents in feet.

The mills to be loaded so that 60 of the volume is occupied by pebbles; moisture in feed 38% to 40 ; peripheral velocity of mill 400 ft. per minute.

In calculating the power for mills it is advisable to add 2 ins. to the internal diameter of the tube when new. This will allow for the increased wad of pebbles required by the mill as the linings wear. It is true that most motors are guaranteed to stand 10 overload, but the tube mill power load is hard on motors. The power fluctuates during each revolution and a mill which required 62 h.p. showed a fluctuation, during one revolution of the mill, from 42 to 86 h.p. A power curve during several revolutions showed the following power conditions:



Fluctuation of Power during Revolution of Tube.

This condition of affairs is due, no doubt, to rearrangement in the pebble load during each revolution.

In direct-connected motors, *i.e.*, when the tubes are driven by gearing directly from the motor, the flexible couplings, which take up the shocks between the motor and tubes, are rarely of sufficient strength as supplied by the electric machinery manufacturers. Generally the mill electrician will have to increase the number of links by 20 . Probably this deficiency in strength is due to the very fluctuating load on the motors. Moreover, it takes a great strain to start a tube mill, as the pebbles are cemented on standing by the slime in the charge. Hence, after a short shut-down, nearly the whole mass of pebbles and charge have to be carried over during the first revolution or two. It will take roughly, twice the horsepower to start a tube mill than is required to keep it running. This has a bad effect on the flexible couplings.

In a number of cases when the tubes have been underpowered the motors have been boosted by furnishing them with current of higher voltage. This is readily effected, when power is purchased from high potential transmission lines, by changing the connections in the transformers. This scheme will not injure the motors if the raise in voltage is not too great, since the insulation in the winding will stand a great increase in potential over that given on the machine plate. However, the real remedy is the purchase of motors of ample power, and any such device as boosting the voltage on the motors should be regarded as a temporary expedient only.

Speed of Tube Mills.—Tube mills should not be run too fast, since by so doing there is a loss not only in power but also in capacity. This is due to the fact that power may be expended in simply carrying pebbles over the interior tube mill circle by centrifugal force. Pebbles should be carried high enough to "cascade" freely and no higher. The height to which the pebbles are carried depends upon the peripheral velocity of the interior circle of the mill, as well as upon the viscosity and lubricating quality of the pulp, and to some extent upon the size of the pebbles and the smoothness of the lining.

The speeds of from 360 to 400 ft. per minute will cover most cases, and the mean of 380 ft. may be confidently adopted in a new plant. The best speed at which a tube mill should run in a given case may be determined in a very simple way. The method is to operate the mill at various speeds determining the power consumed at each speed. By dividing the power consumed by the number of revolutions a factor is obtained which is really the power absorbed in each revolution. The higher the factor the most effective the crushing power of the pebbles. The following is an example:

R.P.M	H.P.	Factor.	Peripheral vel. ft. per min.
27	54·6	2·022	378
28½	58	2·035	400
30½	61·6	2·019	430
32	64·5	2·015	450

The speed of 28½ rpm. gives the highest factor, and would therefore be adopted. However, some power might be saved and the efficiency increased by experimenting a little closer around the peripheral velocity of 400 ft. per minute.

Size of Pebbles. It has now been well established that there is economy in using a pebble size appropriate to the size of feed to the tube. Pebbles of 2 in. to 3 in. are adapted to stamp mill amalgamation sands, and 5 in. pebbles for very coarse material such as ¼ in. roll product.

Pebble Load. In early practice it was the rule to rather less than half fill the tube. Now, however, the mills are often filled several inches over the median axis. The present range of practice is to fill from 50% to 60% of the interior volume of the mills. The tendency is decidedly towards the maximum figure given. A cubic foot of pebbles weighs 100 lb., the rule for calculating the pebble load becomes:—

Pebble load = 60 × cubical contents (c. ft.).

Percentage of Moisture in Feed. Much attention has been paid of recent years to the important question of the amount of moisture in the mill feed. Various figures have been given by different authorities. Nearly all agree, however, that the pulp should be thick enough to coat each pebble with a paste which should not be so thich as to unduly impede the movement of the pebbles amongst themselves. It is obvious then that the amount of moisture must depend upon the character of the ore and the conditions in the tube. The following figures show the percentage recommended by various authorities in specific cases and may be used as a general guide:—

	Per cent.
H. W. Fox	39·6
W. Neal	39·0
G. O. Smart	38·5
H. S. Ball	37·7
Average	38·7

This average corresponds to a specific gravity of 1·6 for a pulp in which the solids have an average sp. gr. of 2·65.

Linings.—Silex linings are still largely used, and when all phases of the question are considered, especially power consumption, output and effect of abraded lining on solutions, there does not appear to be any extraordinary advantage in using honeycombed or ribbed iron or steel linings.

However, on the whole there is a decided majority of tube mills in operation lined with El Oro blocks. The Komata ribbed lining is spoken very highly of by authorities whose experience gives their judgment much weight.

General Remarks.—Tube mills, owing to their great weight and manner of drive, are hard on foundations. Care taken in properly building the foundations is amply repaid by the steadiness with which the mills run and the freedom from annoying shut downs due to breaking of foundation bolts, etc, There should be ample room around the mills for handling pebbles and to allow plently of elbow room in making repairs.

Recapitulating the most important points the millmen and metallurgist should be furnished tube mills:—

(1) Of proper diameter: small diameter for fine feed and large for coarse ore.
(2) Of correct length: short for granulating and long for sliming.
(3) With ample power.
(4) With a good type of classifier.
(5) Well erected on good foundations.

It will then be up to the millman to get the best and most economical results from the mill. With such an installation the high costs for power repairs and low metallurgical results that one hears of in connection with tube mill plants should not materialise." A. M. MERTON.—*Mining and Engineering World*, June 14, 1913, p. 1129. (W.R.D.)

CYANIDING SLIMY ORE BY CONTINUOUS DECANTATION.—"The early history of cyaniding is replete with instances of complete failure or great loss when treating ore which tended to form impermeable slime. Leachable sands were treated successfully by percolation, and in some instances it was possible to mix a portion of the slime with the sand and thus avoid a total loss of precious metals contained therein. In other cases, however, it was temporarily more economical to discharge the slime and impound it against the time when some successful method of treatment should be devised. In this connection the revolutionary influence of agitation and vacuum filtration is a matter of common knowledge.

But great as was the effect of vacuum filtration in overcoming the obstacles in slime treatment, and important as it still is, and will be, in the cyanide industry, the more modern system of continuous decantation is proving of almost equal importance in handling slime. The cyanide process is now the prevalent method of gold-ore treatment: and as the simplicity, economy and ease of operation of the continuous decantation system become better known, it is safe to predict its wider use. It has the advantage of low cost of installation and operation, and it can be combined with existing milling plants without great difficulty. In some instances it displaces filtration, other than clarifying: and where filters are used it permits the delivery of a low-grade pulp to the filter, thus minimising the loss of dissolved gold in the filter tailing. It will be apparent, also, that the use of continuous decantation preceding filters will increase the capacity of a given plant of the latter by avoiding the time required for washing with barren solution. Eliminating this step would also decrease the cost of a filter plant. Further, the operation of continuous decantation is simple and flexible, lending itself to a variety of conditions which may be reasonably ascertained in advance.

A case in point is the recent cyanide practice at the remodelled mill of the Ophir Gold Mines, Mill ing and Power Company, Ophir, San Juan district, Colorado, The company is operating a group of mines, locally known as the Suffolk property, which has a record of production extending over three years, and aggregating over \$2,000,000 in value. Considering the situation of the property and the prevailing metallurgical methods it was natural that during the early years of its history, only high-grade ore was mined and milled. Recovery was effected almost wholly by amalgamation with some concentration. The value of the ore treated ranged from \$15 to \$100, and the recovery probably did not exceed 60%. Under the present system of cyaniding, without amalgamation or concentration, ore averaging \$3 gross value per ton can be mined and milled at a profit, with a recovery of over 90%.

The Suffolk ore consists of quartz, decomposed porphyry and andesite, and contains gold to the value of from \$8 to \$12 per ton. Silver is found only in negligible quantities, and practically no other metals are present. Analyses show over 90% insoluble. The ore is soft, a great deal of it being in the form of clean clay resulting from the decomposition of the porphyry. The condition readily explains the failure of former attempts at cyaniding by leaching. It also gives rise to much colloidal slime in the present system of treatment, but no difficulty is experienced in handling it.

Preliminary experiments revealed certain ideal qualities for cyaniding not found in all ores. There are no reducing agents present and the ore is only slightly acid. Chemical consumption of cyanide is only 0·4 lb. per ton, and of lime 1 lb. per ton. Being soft, the ore is readily ground to slime. This is an important characteristic because the gold is very finely divided, and, after crushing, is found for the most part in the fine slime. Despite the fact that the ore contains a great deal of colloidal matter, no difficulty has been experienced in handling it by the present method of continuous decantation.

An interesting feature disclosed in the tests was the fact that the quantity of readily soluble gold varied directly with the gross value of the ore, while the quantity of gold requiring time and agitation for its solution was almost constant regardless of the initial value of ore. Thus it was found that on all grades of ore the gold went into solution almost immediately to a point where the residues had a uniform value of \$2.40 per ton.

These residues yielded their gold on further agitation. Solution goes on quite rapidly, giving residues containing 20 cents gold per ton after thirty-six hours' agitation.

This is the standard now maintained in mill operation, although as a matter of fact 'trace' tailings are frequently obtained.

The present milling system is based on many tests and calculations. In this connection it will be interesting to note that the results obtained in practice checked closely those indicated by the calculations.

The milling process begins by crushing the ore in cyanide solution with twenty 850-lb. stamps. The height of drop is from 6 in. to 7 in. at the rate of 104 drops per minute. With ¼ in. screens on the mortars the twenty stamps have a capacity of 150 tons in 24 hours, and 30% of the discharge will pass a 200-mesh screen.

Considering the nature of the ore, the stamping is not so much a problem of crushing as of disintegrating and washing out of the clay.

The stamp product is separated into sand and slime in a Dorr classifier, the sand being reground in an 8-ft. Hardinge conical mill. The latter carries a 4-ton charge of pebbles and runs at 31 r.p.m. Its discharge flows to an Abbe-Fremer pump which returns the pulp to the classifier. The classified slime is elevated by centrifugal pump to the first Dorr thickener, 33 ft. diameter and 12 ft. deep, running at the rate of one revolution in 12 minutes. Solution of most of the gold occurs so rapidly in the stamp-mortar and tube mill circuit that the overflow of this thickener is clarified and precipitated by zinc dust. The consumption of zinc amounts to 0·15 lb. per ton of solution.

The rich solution is measured before precipitation, by means of an all-iron Worthington direct-displacement water meter. An automatic sample is taken both before and after precipitation. The difference between the assay value of these samples, multiplied by the tonnage registered on the meter, gives the amount of gold precipitated during a given period.

The thickened slime, containing 1½ parts solution to 1 of solids, flows to the agitators. The agitators are of a new type, combining air and mechanical agitation, patented by Mr. J. v. N. Dorr. The mechanism is applicable to flat-bottomed vats of ordinary proportions: those at Ophir are of wood, 16 ft. high and 16 ft. diameter. The driving gear is supported on a framework at the top of vat as shown. The central air-lift pipe is connected with the vertical driving shaft and revolves with it, being suspended in the vat. The open bottom of the central pipe reaches within 8 in. of the bottom of the vat and is directly above the inlet valve for compressed air. Attached to the air-lift pipe near its lower end are the agitator arms to which are riveted angle-irons so placed as to draw pulp from the periphery to the centre of the vat during the revolution of the mechanism. The arms are adapted to be lifted out of the pulp in case operation is discontinued for any length of time. The pulp discharged from the air-lift is directed into inclined radial launders attached to the top of the central pipe and revolving with it. The bottoms of the launders are perforated to distribute the discharge over the surface of the pulp.

In operation, therefore, there is a combination of air and mechanical agitation. The arms, which in this case revolve only once in 54 seconds, continually draw thickened slime from all parts of the vat and deliver it to the air-lift for circulation and agitation. This agitator is giving entire satisfaction. It avoids the inconvenience in mill construction due to the great height of Pachuca vats, and it overcomes the difficulties arising from accumulation of sand, which frequently occurs in cone-bottom agitators.

It is recognised that in many types of agitators the intensity of agitation is determined more by the mechanical necessity of keeping the pulp in suspension rather than by the chemical requirements of the ore under treatment. This frequently results in a waste of power; and where air agitation alone is used, the excess of air may readily be a disadvantage. This agitator maintains the pulp in suspension under every condition, and allows the intensity of agitation to be regulated to suit the needs of the ore.

The agitated pulp is subjected to a system of continuous decantation and counter-current dilution. The final tailing is discharged from the last thickener without filtration. The loss in dissolved gold at this point is but 3 cents per ton, which compares favourably with similar loss from filters, the latter ranging from 4 cents up as high as 20 cents or more per ton. The discharged pulp contains from 48% to 54% solids. In applying this system, a vital thing to be observed is the point at which barren solution is added; and this is governed by the relative value of the pulp in cyanide and dissolved gold, and the proportion of each that may be most economically discharged in the final tailing.

It will be noticed here that in the present case the barren solution is added in the third of the five thickeners. This is based on experiments and calculations designed to give the lowest combined loss of cyanide and gold, and is not necessarily the exact scheme which would be followed with other ores and solutions.

The flow of solution through the entire mill system is in a closed circuit except for that portion of solution discharged as moisture in the final tailing. This is compensated by the addition of hot water from the heating system, so that the solution is kept at an average temperature of 85° F. The quantity of hot water thus introduced is governed by a float in the mill solution storage vat, which operates a throttle valve on the hot water line.

The float is placed in this vat because it is the only one in the entire system in which the level of solution varies. It is also the only vat in the system besides the treatment vats. Sump vats are not needed.

An important feature at Ophir is the convenient arrangement and control of the air-lifts used for transferring pulp from one thickener to another. These lifts are similar to those used at Real de Monte, Pachuca, Mexico The four thickeners are symmetrically arranged and in the central space are grouped the air-lifts and boxes which receive the various pulps and solutions. This arrangement greatly facilitates inspection of the entire system of continuous decantation. The thick underflow discharged through a nozzle and the overflow to be mixed with it, flow into a sump box and the mixture is elevated to the proper vat by an air-lift.

The thickened pulp and diluting solution which are to be transferred to any one of the thickeners are received in one compartment of a box which is 2½ ft. sq. and 3 ft. deep. Attached to the bottom of each compartment and extending downward into a concrete-lined well, is a 9 in. pipe, 23 ft. long, capped at the bottom. This gives a total submergence of 25 ft. below the normal level of slime maintained in the box. Inside the 9 in. pipe and extending downward within 4 in. of its bottom cap, is a 5 in. pipe which rises also 13 ft. above the surface of the slime in the box and discharges the elevated pulp to the feed box of the desired thickener. The inner pipe is supported on legs resting on the cap of the outer pipe. A 1 in. compressed air pipe is introduced into the top of the 5 in. pipe and extends downward within 10 in. of the bottom of the 5 in. pipe.

In each compartment of the box referred to is a float rigidly connected by iron rod with a throttle valve on the air line. As the pulp accumulates in any compartment, the float rises, opens the valve and operates the air-lift. When the level of slime in that compartment has been lowered, the float automatically closes the air valve and the elevation of pulp is temporally discontinued."—H. C. PARMELEE, *Metallurgical Engineering.—Mexican Mining Journal*, March, 1913, p. 145. (A. R.)

MILLING BY COARSE CRUSHING IN CYANIDE SOLUTION.—"The efficient and economical results attainable in the milling of two certain ores by crushing them coarsely, namely, from two to eight mesh, in cyanide solution with stamps, the pulp thus produced being classified, the sand (a large proportion of the pulp) leached in vats and the slime agitated and filtered, were brought out clearly by investigations upon which the writer has been employed professionally during the past year, as to the best method to pursue in milling each of these two ores. Since they are merely well-defined types and as, therefore, the methods found to be suitable for them will no doubt work equally well on other similar ores, with modification, a fuller description of the results of tests upon them, and discussion of these results, may be of interest to your readers.

One of these ores is an oxidised, highly porous gold ore of medium milling grade: the other is an unoxidised quartzose, low grade gold ore, containing about 6% of gold bearing pyrite coarsely disseminated in gangue: neither ore contains sufficient free gold to make plate amalgamation advisable. The high porosity of the one and the coarse dissemination of concentrate in the other are the determining factors in the success attained by the above procedure on ores of such contrasting character. The tests of each were made as a basis for operation on a forty stamp scale.

Results of Tests on the Porous Ore.—A careful inspection of the different kinds of this ore under a strong lens showed that most of it was remarkably porous, even sponge-like or fine vesicular, as was indicated also by the readiness with which the ore took up and retained a moisture content of over 15%. This suggested at once that cyanide solution might penetrate to the interior of quite large ore particles, and hence that it might be possible to mill the ore with an equal extraction when crushed unusually coarse as when crushed to 20 mesh, as was the practice. With a 20 mesh, punched slot screen on the batteries, the stamp duty was about four tons, the last nineteen vat residues had averaged $0·59 in gold and $0·11 in silver, total, $0·70, while about 55% by weight of the feed was leached in the vats as sand, 45% went to the slime agitators and filters.

To determine what could be done by coarser crushing, 100 lb. of ore was taken from the feeders, crushed in the assay office rock breaker to pass 2 mesh, and from this samples of 35 lb. were cut out and crushed to pass 4 and 12 mesh in the same way. About 25 lb. of ore was taken for each test, the slime removed by wet screening on 100 mesh and the oversize, which assayed $2·79 per ton in gold and $0·22 in silver, leached with 2½ lb. solution.

In the test with the ore crushed up to 12 mesh the oversize on 100 mesh was 58% of the ore by weight. After 15 hr. leaching, a residue sample taken from the top layer assayed $0·72 in gold and $0·16 in silver, and after 39 hr., $0·62 in gold and $0·16 in silver, and after 64 hr., $0·62 in gold and $0·14 in silver; total, $0·76, showing 39 hr. to be long enough.

In the test with the ore crushed to 4 mesh, the sand leached was 67% of the ore by weight. The ore leached so freely as to show that if crushed to this size and properly classified, it would leach fast and clean in vats 10 ft. to 15 ft. deep, and that the vats would have a greatly increased capacity over what they have on 20 mesh crushing. After 36 hr. leaching with 2½ lb. solution, an assay sample of residue taken off the top, well washed and screened gave these results:

	Pct. by Wht.	Gold Assay Val.
On 12 mesh ...	67	$0·98
Through 12 mesh	33	$0·98

After 60 hr. leaching, the average residue was $0·72 in gold and $0·18 in silver: after 84 hr., it was $0·52 in gold and $0·18 in silver, total $0·70, the same as the 20 mesh mill vat residue average, and after 110 hr., the same. A sizing test of the final washed residue showed as follows:

Size.	Pct. by Weight.	Gold Value.	Sil. Val.
On 12 mesh	58	$0·62	$0·24
On 40 mesh	28	0·31	0·11
Through 40 mesh	14	0·52	0·08

Comparison of the two tests shows that a longer time of leaching is required with 4 than with 12 mesh crushing, but that they give equally good results finally.

Crushing tests were made next to determine comparative capacity with various screens. A new 20 mesh slot punched screen was placed upon one of the batteries of five 1,000 lb. stamps run at one hundred five-and one half inch drops with a two inch discharge. The time required to mill a weighed amount of ore, 2,500 lb. wet weight, or 2,082 lb. dry weight, the rock used containing more than an average amount of soft, fine ore, was next noted and found to be 75 min., equalling a stamp duty of 3·998 tons per stamp per 24 hr. run. The stamps were then hung up, a 4 mesh screen of heavy steel wire put on, no other change being made in the battery, and another rating immediately made. It required 40½ min. to crush 2,200 lb., dry weight of ore containing more than an average amount of large, hard pieces, equalling a stamp duty of 7·822 tons per stamp per 24 hr. run, which is nearly double that with the other screen.

Battery samples of the pulp made during the above ratings were taken, evaporated to dryness and screened with these results:

Size.	20 mesh Slot Screen	4 mesh Wire Screen.
On 12 mesh	—	15·3
On 20 mesh	0·17	18·4
On 40 mesh	17·0	16·7
On 100 mesh	34·0	25·5
Through 100 mesh	49·0	24·1

This sizing test shows that about 76% of the ore could be leached at 4 mesh crushing, but only about 51% at 20 mesh crushing. Calculated as a proportion of the oversize on 100 mesh, the total amount to be leached, it will be seen that only 44% remained on a 20 mesh screen, of the pulp produced with a 4 mesh screen on the battery, while in the 12 mesh small leaching test, 48% of the sand residue stopped on 20 mesh. Hence, this 4 mesh battery pulp will give the same results as to extraction and time required for leaching as was shown in the small 12 mesh test.

It is clear from a careful consideration and comparison of the above data that the extraction would not be decreased, but that the mill capacity would be nearly doubled by changing the battery, screens from 20 mesh to 4 mesh: that the slime department would not need to be enlarged, and that the only enlargement of the mill required in order to double its capacity would be to add more leachings vats, and even these would not have to be added in proportion to the increase in tonnage on account of the freer leaching.

I tried, also, to make a rating with a 2 mesh wire screen on the same battery, immediately following the test with the 4 mesh, but the amount crushed was so enormously increased that the grade of the launders was not sufficient to carry off the pulp produced: apparently, the stamp duty was as much as 15 tons. Screening of the pulp showed that nearly all of it passed 4 mesh, so that it could be cyanided without loss of extraction, as shown by the small test on ore broken to 4 mesh in the assay office crusher. Hence, even 4 mesh is not the limit of coarseness of battery screens which might be used without impairing the extraction on this ore.

Results on a Hard Ore Containing Coarse Pyrite.—This ore was not at all porous, being mostly medium hard quartz with some wall rock, yielding less than 12% slime, and contained about 6% of fairly coarse pyrite concentrate assaying $35 in gold. After they had been concentrated as closely as possible, the slimes assayed richer than the heads and hence cyanidation is necessary for them. Little concentrate could be obtained on 4 mesh crushing, but it was found that the ore 'opened' suddenly at 8 mesh crushing, at which size many particles of pure pyrite the full size of the mesh were obtained and a recovery made in concentrates of 73·5% of the gold in the ore, while at 12 mesh crushing, a recovery in concentrates of only 70% of the gold could be made. The concentrate was 75% pyrite.

The mill treatment decided upon was to crush in solution, using a 4 to 6 mesh screen on the batteries, to give an 8 mesh pulp, or a still coarser screen with thé use of regrinders run to produce an 8 mesh pulp, size the product at 30 to 40 mesh on belt screens, and concentrate the oversize of these on jigs and the undersize on tables. The fine sand is to be separated from the slime on the tables, supplemented by classifiers if required. All the coarse and fine sand is to flow together to large leaching vats to have the solution washed out and be discharged to waste by sluicing; the slime is to run to Dorr thickeners, thence to a continuous air agitator, if necessary to increase the time of solution contact with the slime, and thence to an Oliver filter, the decanted and precipitated solution being pumped back to battery storage to complete the cycle. This coarseness of crushing is expected to give a stamp duty exceeding 7 tons without regrinders and exceeding 10 tons with their use.

The tests showed that 68% of such a pulp was coarser than 40 mesh, contained 42% of the gold in the heads, and when jigged, 77% of the value contained in this coarse size was recovered in concentrate, which was 75% pyrite and 2·8% by weight of the ore. The sand finer than 40 mesh was only 22·5% of the ore by weight, contained 40% of the gold in the heads, and, by vanning, 88% of the value contained in this fine sand was recovered in concentrate which was 2·9% by weight of the ore and over 80% pyrite: while the flotant slime was 9·5% of the heads, contained 18% of the gold in them, but from it only 29% of its value could be recovered in a sandy concentrate which was 3-10 of 1% of the weight of the ore. The size jigged, coarser than 40 mesh yielded 4·2% of its own weight of concentrate; the size between 40 and 50 mesh, yielded 12·5%; the sand finer than 80 mesh, 16·8%, and the flotant slime yielded only 3·3% of its own weight of concentrate.

These data show the advantages of such a scheme of concentration; the jigs handle with a large machine as a unit, over two-thirds of the ore, and make a close recovery on all liberated concentrate; at the same time, the screens put into the fine sand size of less than a quarter the weight of the ore over half of the concentrate in it, so that attention to close concentration can be focussed on this size, where tables are at their best and do very close work with little trouble; close concentration is not necessary on the slime, where it is bothersome and inefficient, as the cyanide will recover whatever value is left in the slime pulp by the tables.

As to the working of this scheme in the cyanide department: The jig tailings were so poor as hardly to be worth cyaniding but it was, of course, imperative to leach them if crushing were to be done in cyanide solution, and as the chemical consumption was only a ¼ lb. of cyanide, costing six cents per ton of ore, and the sand can be handled cheaply in large vats, it was estimated that the gold recovered from it will pay the cost of treating it. It was desirable to crush in solution to avoid dilution loss, as the slimes require a solution of 2 lb. strength.

It is interesting to note that tests showed the same extraction in bullion, allowing for the loss in residues in cyaniding these mill concentrates, by the above method, as by sliming all the ore to 150 mesh and cyaniding without concentrating, being 88% in both cases.

The coarseness of crushing permissible is a fundamental point to investigate in designing a mill, because the kind of classifiers or screens, the mill grades and the balance of the plant will all depend upon it. In case of uncertainty, and to give the mill flexibility, if it be designed to crush, somewhat too coarse, no great harm is done should a change in the ore necessitate finer crushing, since, on putting on finer screens, reducing the tonnage, the slime plant will handle the larger proportion but no greater tonnage of slime, the pulp will run well in the launders, and only some cheap concentrators or leaching vats will stand idle.

These two examples illustrate that it is by no means always necessary for successful cyanidation to crush the ore even as fine as is required for plate amalgamation: and it can do no harm to put emphasis on this fact now that all sliming plants are so much in fashion. We are all fascinated by the close recoveries made by sliming plants and on ore of high milling grade, especially where the gangue is dense or the ore minerals finely disseminated or slowly soluble in cyanide solution such as the ores of Tonopah and Goldfield, and much of the ore in Mexico, no other type of mill can compare with them. But fine grinding plants are not the best except where their higher recovery much more than wipes out their higher operating cost, plus the greater amount which must be charged against each ton to redeem the capital invested in them with interest.

We must not forget that a low milling cost often increases the tonnage available, and that the more expensive the type of mill, the smaller must be the scale of operations for a given investment. Where there is a large deposit of cheaply mined ore, such coarse crushing plants permit the milling of a large tonnage cheaply, with a comparatively small mill investment and at the same time with a good recovery of values on suitable ore."—ERNEST H. SIMONDS, *Mexican Mining Journal*, March, 1913, p. 135. (A. R.)

DIMENSIONS OF TUBE MILLS.—"The recent date of most of the mills of the Tonopah district affords to this camp the distinction of illustrating very modern ideas in the dressing and chemical treatment of silver ores. Accordingly, a comprehensive review of the methods there in force serves in large measure to reflect the prevailing views and practice of metallurgists.

This result is well accomplished by the cyaniding specialists, Herbert A. Megraw, in a series of articles on 'Silver Cyanidation at Tonopah,' appearing in the *Engineering and Mining Journal*, (Feb. 22 March 8, 1913).

The Tonopah district (says Mr. Megraw) offers to the student of contemporary metallurgy what is probably the most interesting demonstration of efficiency of the cyanide process applied to silver ores. This is not to say that the general methods in use are original, as they are not, but the application of these methods has been attuned to the requirements of the particular ores to be treated and the problems have been attacked and solved with praiseworthy conservative judgment. It is both interesting and noteworthy that in all the Tonopah district there is not one idle reduction plant, nor one which has been designed by a tyro to include exaggerated ideas or 'freaks.'

Five plants are comprehensively, though not exhaustively, considered by the author: The West End, Montana-Tonopah, Tonopah Extension, MacNamara and the Desert mill at Millers, operating upon Tonopah Mining Company ores. While Mr. Megraw summarizes in a very satisfactory way his observations concerning several milling features, his reflections upon the subject of tube mills impress the reviewer more especially. In all the plants recently constructed the length of the tube mills has been reduced, while larger diameters now receive the preference. It is not impossible, in Mr. Megraw's opinion, that the tube mills of the future may be more like drums of large diameter and short length than the long, narrow tubes first used. At Tonopah the result of the change has been a reduction in the amount of power necessary for the same amount of ground product.

The Montana Tonopah plant is one of the original mills in the district to make use of the all-sliming system. It accordingly still has in use two tube mills of the long type, 5 ft. × 22 ft. The West End and Tonopah Extension managements have selected the 18 ft. length, while the MacNamara mill, a recent installation, uses a 16 ft. length.

It appears that this type is meeting with almost universal approval from metallurgists of the present day, the conclusion being that in the former mills there was a great deal of space at the discharge end which did not perform any useful work. The change toward larger diameters and less length has resulted in an increased production of slime at a power consumption, per ton of ore, equal to or less than with the long tubes. There is an added advantage in the smaller floor space occupied.

With reference to the further reduced length of tube at the MacNamara plant, the author says:

> It is stated that with the 16 ft. mill the results are the same, as far as fineness of output is concerned, as if the 22 ft. length were used, and the wear on the lining and the power and pebble consumption are less.

Walter Renton Ingalls, editor of the *Engineering and Mining Journal*, impressed with the showing at the MacNamara plant, added his comment to that of the contributor:

> This reduction in length would be considered a radical departure were it not for the fact that the overall length of the Hardinge mill is not more than 10 ft. Is it possible that metallurgists, after adopting a device from the cement industry, have failed to make the most of it by not adapting it to their needs? If the results obtained at the MacNamara mill prove of general application, it would seem that a simple expedient was overlooked, whereby the efficiency of the tube mill might be increased.

If this development in tube mill practice excites approbation, a similar opinion is not found in Mr. Megraw's commentary concerning the disuse of ribbed lining in the Tonopah mills. He apparently accepts with a great deal of doubt the claims of efficiency for smooth lined tubes which were discarded in Mexico before the El Oro lining came into use. In all the Tonopah installations the author failed to discover a single instance of the use of the El Oro ribbed lining, but everywhere he found the use of smooth cast lining and silex blocks.

The smooth cast lining would seem, theoretically, to be the most inefficient of all possible linings, as it would appear that the pebbles would be more than likely to slide over it and reduce the ore by rubbing it between the lining and the pebbles. The matter of the dropping or the tumbling of pebbles would apparently be reduced to a minimum, and while it is clear that the power requirements would be reduced to the lowest possible point, apparently the efficiency would go down with it. That this is not the case is what several of the operators declare positively. They claim to get better results, both as to character of product, its quantity and cost of production. And it is also stated positively that the pebbles do not wear flat in the smooth lined mills. It is a fact that, if one is careful, he can avoid seeing flat pebbles on the scrap heaps of these mills.

It is an unpleasant reflection that the dubious tone of these observations may diminish one's enthusiasm for the Tonopah claims of short-tube efficiency.

But concerning this general problem of tube length it is worth recalling that in the elaborate investigations of mill feed, pebble load, moisture and speed efficiencies, reported by H. Standish Ball to the Institution of Mining and Metallurgy (*Bulletin*, August 30, 1911), use was made of a laboratory model constructed from an old chlorination barrel, the inside dimensions with liner in being 3 ft. 6 in. long and 2 ft. 10 in. in diameter. Silex bricks were used in that case."—*Mining Science*, April, 1913, p. 211. (A. R.)

DRY v. WET CRUSHING AT KALGOORLIE.—"According to the last annual report of the Hainault Gold Mine, Ltd., owing to change in the character of the ore being mined, treatment by the stamp-mill and cyanide plant is now unsatisfactory and a dry-crushing plant may be necessary if the mine develops satisfactorily. The time seems opportune to review the treatment problem at Kalgoorlie. It is with some diffidence that I do this, although after eight years' general observation one should be in a position to do so easily. There are so many points to study and compare that the discussion may easily be inconclusive. Having watched the treatment of the ores by the dry crushing method most of this time, I may be accused of being biassed in its favour; I have also studied the wet-crushing process, and the general remarks of those in charge have been forgotten.

In general, the ores are of a gray to greenish colour of a schistose structure, and they show an increase in hardness and silica content with depth. They contain pyrite and tellurides of gold and silver, but no minerals harmful to treatment. There is little free gold, it being mainly in the pyrite. The most important constituents have an average of SiO_2, 63%; Al_2O_3, 4%; $CaCO_3$, 12%; $MgCO_3$, 6%; FeO, 7%; and S, 3½%.

First Attempts.—When treatment of the Kalgoorlie sulphide ores was started, the ore was not understood, as nothing like it had been treated in Australia before. In 1899-1901 several large plants were erected, and, while results were low, yet they were the means of spurring metallurgists to further trials. Gradually the present satisfactory methods were evolved. In the dry process, the effect of roasting was not thoroughly understood, and the ore in the leaching vats had to be cut out by picks and bars. Then the analyses revealed the changes in lime and magnesia during roasting. In the wet mills, losses through the gold in tellurides not being dissolved, and trouble through the pyrite not being properly saved, also upset results.

After the early failures in treatment, two schools of treatment grew up. The Associated, Associated Northern, Boulder, Kalgurlie, Perseverance, and South Kalgurlie now treat by dry crushing; the Hainault, Horse-Shoe, Ivanhoe, Lake View and Star, and Oroya Links by wet crushing. Machinery is as effective as can be, labour is well paid and intelligent, supplies of fuel and water are of the first order, economy is pushed into every department, and the plants are kept at work full time.

In the dry plants if the roasting be good, there is little fear in subsequent operations, and there is only one product, and an easy one at that. It soon was found that tellurides were not soluble in ordinary cyanide solutions, but bromo-cyanide proved to be effective. It is used in wet mills where either concentrate, sand, and slime, or concentrate and all slime is produced. This necessitates three or at least two subsequent processes, and close supervision. Bromo is still used at the Horse-Shoe, Ivanhoe, and Lake View. The Oroya Links is not using it much now. Great care is necessary in its use, yet it has helped to materially increase the extraction in the wet mills.

Comparison of Cost.—The dry process is a trifle higher in cost than the wet process, but the residue from the former is lower than from the latter. The comparison may be stated as follows: (1) dry process cost $2.50 plus the residue (average extraction 93% on $8.26 ore, equals 70c.) equals a total of $3.20 per ton; (2) wet process cost $2.24 plus residue (average extraction 88% on $6.84 ore, equals 69c.) equals a total of $3.20 per ton. These figures are the best obtainable, as all cost systems are not similar, and extractions are not always published. It is admitted generally that the dry work is superior to the wet. During the month of May last, the tonnage treated dry divided into the total yield gave the above average figure. In the same month, wet tonnage and output gave the average quoted.

On the one hand there is a residue from the dry plants which will not pay for re-treatment; but from the wet mills a residue of $1 or over can be re-treated by vacuum-filtration at a cost of 50c. per ton; and if the loss in treatment be 20c., the profit would be 30c., which is worth saving. A few of the dumps contain more than $1 per ton, but I speak of results of present practice. The Oroya Links published figures giving the value of its residue lately as 42c. Against the first cost plus residue of $3.20 in the wet mills, must be placed cost plus residue plus re-treatment in the wet process, or $3.20 plus 50c. equals $3.70 per ton. This shows an advantage of 50c. in favour of the dry process.

Dust Losses. Millmen maintain that a deal of gold is lost in dry crushing and roasting. From ball-mills and dust-houses about 0·5% of the ore may be lost, and in roasting in a modern furnace not more than the same amount. Dust losses are difficult and costly to determine, but estimating all dust to average in value with the original ore, 1% of 2,400 tons dry treatment daily of $8.26 ore equals 8c. per ton lost. Experiments have proved that there is no actual volatilization of the gold in roasting telluride ores. The average loss in weight in crushing and roasting ore at Kalgoorlie is 10% : made up of dust, 1% : moisture, 1% : sulphur, 2·5% : and CO_2 from carbonates, 5·5%. In roasting concentrate, the loss is only about 0·05%, and as this product averages some $50 per ton, the loss on 7,000 tons treated monthly would be 3c., or 0·25c. per ton on 2,730 tons treated wet daily.

The only amalgam in a dry plant is in the pans, while in the other process amalgam is collected from boxes, plates, pans, in battery, and the concentrate plant. There is accordingly more supervision necessary in a wet mill, and risk of stealing is greater. The average loss of mercury is low.

Concentration has been carried to a fine point at Kalgoorlie, but none the less very fine mineral and telluride floats and mixes with the pulp going to leaching vats and slime plants. This fact is, to my mind, one of the reasons for difficulties in a wet plant on such ores, in spite of the use of bromo-cyanide.

Classifying in a dry plant consists only in separating the pulp for the pans, and any coarse discharge from the latter is returned for further grinding. In some stamp-mills, classification for tables, re-classification for tube-mills, and further classification for more tables, is necessary, and the system rapidly becomes complicated. It necessitates extensive systems of classifiers, launders, and return pumps.

The average cost of ball-milling is 50c., against say 42c. per ton for stamping. I think the former plant easier to look after than the latter. Both produce products that are equally well adapted to further treatment.

Efficiency of Roasting.—This discussion would not be complete without mention of the general efficiency attained in roasting ore or concentrate. For capacity, fuel consumption, power, labour, and results, furnace work at Kalgoorlie cannot be beaten. As examples of roasting ore, the plants at the Associated, Perseverance, and Kalgurli may be mentioned, and of roasting concentrate at the Horse-Shoe.

One of the troubles at Kalgoorlie is the presence of graphite, which is found in the Associated, Associated Northern, Boulder, and Oroya mines. It does not seem to change during roasting, and causes a premature precipitation of the gold from cyanide solutions. Fortunately, the mineral does not come to the mills very often; but as to this effect, there is no doubt, and it is a nuisance hard to deal with. The Boulder is troubled with it more now than any of the other mines.

Introduction of Vacuum-Filters.—Gradually the filter-press is being discarded in favour of vacuum-filters. In the past an immense sum has been spent on press plants, about 100 presses being erected, 80 of which are treating 100,000 tons monthly. The balance are out of commission, but the benefit derived from this machine at Kalgoorlie has been admittedly large. With a press washing can be carried to a degree that cannot be beaten, but the labour cost is high, and I expect that it will have to give way. The Associated Northern, Boulder, and Oroya Links filter their current mill slime by vacuum systems of their own, while others are talking about introducing the system. The lowest cost of treatment by filter-press is 25c. per ton, while 8c. should cover vacuum-filtration. Such a saving is important with ore yielding as low as $5.50 per ton.

The cheap price made for water, 36c. per 1,000 gal., for sluicing away all mill residue, has simplified this work and done away with the further building of dusty dumps. Sluicing, on the whole, is cheaper than mining dumps, and Kalgoorlie has much to be thankful for in connection with the great water scheme.

On the cost of erecting the two types of plants discussed, there is not much data on account of constant additions, improvements, and alterations. The Associated Northern plant, with capacity of 120 tons of sulphide ore daily, cost originally $180,000. For the same tonnage, a battery of 25 stamps would cost $125,000.

The margin between the dry and wet processes was large enough at one time, although extractions vary from 94% to 86%, respectively, and left no doubt as to which should be chosen. Now the stamp-mill work is improving slowly but surely, and the difference is small. For my part, I would build a dry crushing and roasting plant to treat a similar ore."—M. W. VON BERNEWITZ. *Mining and Scientific Press*, March 15, 1913, p. 409. (A. McA. J.)

MINING.

REINFORCED CONCRETE PROPS AND BEAMS IN MINES.—"Reinforced concrete is very little used underground, and doubtless there are economic reasons for this, but the use is growing, and on many occasions when I have been testing columns and beams reinforced in the usual way, mining engineers who were present at the tests have said that doubtless within a few years the use of reinforced concrete in mines will become more general. The use of reinforced concrete is becoming general for pit head work, such as bunkers and hoppers, and in a few mines in Great Britain it is used on a large scale below ground, some of the principal pits being Baggeridge, near Dudley: Sneyd, near Stoke-on-Trent: and Tunnel Pit, Haunchwood, near Nuneaton.

Plain concrete is not used in mines to the extent which engineers who work above ground would anticipate. Of course, one reason for this is that time is an all-important quantity in mining, and the methods in use at the present, which give the full width of the roads immediately they are timbered, seem preferable to those methods which need temporary support, blocking the narrow roads for considerable periods. To the miner the use of timber is now instinctive, and the many changes in his work necessitated by the adoption of concrete and reinforced concrete seem irksome. The engineer is also to blame in some cases because, without studying the special conditions, he merely aims at substituting one material for another which, owing to years of experience, is being used in the forms to which it is most adapted. For instance, we find in early days before suitable methods were devised for combining timber and iron in bridge structures, that the masonry arch was imitated, and again, the timber arch was imitated in cast iron, and now we find in some cases engineers who use reinforced concrete paying too little attention to the form of the design, and to a large extent imitating structures in steel and wood or even masonry. Without doubt in some cases this may be necessary, but in many cases it could be shown that economy would be obtained by using the material in the special forms to which it is adapted. To a very limited extent, therefore, do I anticipate that beams and posts of reinforced concrete will be used in mining as substitutes for timber, but after a complete study of the possibilities, it ought to be possible to design some really efficient substitute for timber in many cases. Besides, with our present limited capacity in testing machines in this country, it is satisfactory to have for testing purposes numbers of structures of full size as constructed. Few testing machines are capable of testing the columns and beams used in ordinary building construction, and so our experimental knowledge of the actual strength of these members is very limited. On the other hand, the results obtained from experiments on 8 ft. beams and posts will have a direct bearing on the posts and beams which might be used in mining work.

Of course it is only in main roads and permanent work in mines that it is anticipated there will be any extensive use of reinforced concrete work. The strength of the timbers used now in mining was fully discussed in a Paper by Prof. Louis, and he states that the strength of larch props is 3,360 lb. per sq. in., and of Scotch fir 2,688 lb. per sq. in., when the timber in each case is thoroughly seasoned, dry, and also fully grown, while the strength of wet timber is from 40% to 50% less. These experiments were made with ordinary pit props 4 ft. long, and Prof. Louis states that the prop has the same strength whatever its length, under the ordinary conditions of practice. It is evident that frequently in mines the props and beams will be wet, and therefore, according to these figures, the existing strength of a larch bar should not be assumed to be more than 1,780 lb. per sq. in., which will certainly be less than a well made reinforced concrete post four months old. Besides, in a mine dry rot quickly attacks timbers unless they are constantly wet.

The cost of timber for mine work is continually increasing, the price of foreign timber having risen as much as 45% during the last five years. For many reasons, therefore, substitutes have to be found for timber for the main roads where the work is to be of a permanent character, and we find that the use of steel joists is very usual. Joists are used instead of head trees, and at the ends a clip on the flange affords a simple and effective method to keep in their proper position the joists used instead of the side props. If the roof needs support between the joists, timber is generally used, whereas concrete is sometimes used between the joists at the sides. It is evident that reinforced concrete slabs with rock packing, or concrete filling if excessive loads are expected, might be most effectively used in conjunction with joists.

No doubt one of the chief reasons why concrete has not been used in mines is that it takes a considerable time to attain its ultimate strength. It is possible, therefore, that a method of supporting main roads in which can be used posts, beams, and slabs, manufactured and stored for at least six months, will be found economical in some cases.

In arranging a preliminary set of experiments on the strength of beams and posts of reinforced concrete for mine work, the results obtained from beams of (*a*) 8 in. by 12 in., (*b*) triangular 8 in. side cross-section, were compared. All the concrete used was of the same composition, and was made under as similar conditions as possible. The proportions selected were 1 : 2 : 5. The sand was from Leighton Buzzard, and the gravel was well screened between 1 in. and ¼ in. The concrete was mixed wet, that is, with about 8% of water, experiments having been previously made on the effect of varying the amount of water in this concrete. The results are given in Table I.

TABLE I. Influence of Amount of Water in Strength of Concrete.

Per cent. of Water.	Age in Days.	Average Compression Strength Lbs. per sq. in.
6	38	1,247
6	64	1,543
6	121	1,350
8	37	791
8	66	1,253
8	132	1,260
10	35	390
10	64	750
10	130	1,010

These results seem important, since in most reinforced work wet mixtures are required, and till the concrete is over two months old the consequent diminution in strength is very serious. The cement was slow setting and uniform in quality. It was supplied by the makers in barrels, and each barrel tested according to the British Standard Specifications, the strength being:—Lb. per sq. in. at 7 days, 473; lb. per sq. in. at 28 days, 635; and the time of setting 3½ hr. The steel used had a tensile strength of 28-30 tons per sq. in., and a yield point at 20-21 tons per sq. in.: modulus of elasticity 30·8 10^6 lb. per sq. in. All the tests on the beams and posts were carried out at the age of two months. About a hundred cubes of concrete 6 in. by 6 in. by 6 in., made the same time as the beams, were tested at two months, and gave an average compressive strength of 1,600 lb. per sq. in. During the last five years I have carried out many tests on 6 in. cubes of various mixtures, made and filled by workmen engaged on reinforced concrete buildings in the Midlands, and in no single instance has a strength of 2,000 lb. per sq. in. been obtained when the age was under two months. The gravel, which was fairly uniform in size, passed a 1 in. sieve. The voids were 36%.

The gravel was not washed, since preliminary experiments failed to show a noticeable increase in the strength of the concrete by so doing, and it was desired also to have the conditions as like as possible to those usually obtaining in practice. The forms were removed three days after casting, and the concrete was not allowed to dry for six weeks.

Tests of Beams. In testing the 12 in. by 8 in. beams the supports were 7 ft. 6 in. apart; the load was applied at two points 3 ft. 6 in. centre to centre in the method usually adopted. Table II shows the result of the preliminary tests.

TABLE II. Tests of Reinforced Beams.
8 ft. × 1 ft. × 8 in.

No.	No. of Specimens.	Reinforcement. Kind.	Reinforcement. Per Cent.	Breaking Load. Equiv. Uniformly Distrib. Load. Tons per lineal ft.	Method of Failure.
1	3	2 Straight ½ in. ...	0·45	1·0	Tension
2	3	2 Straight ½ in. ... 1 Bent up ½ in. ...	0·67	1·3	Diag. tension
3	3	1 Straight ½ in. ... 2 Bent up ½ in ...	0·67	1·1	Plain & diag. tension
4	3	1 Straight ½ in. top 1 „ ½ in. bottom 4 Bent up ½ in. ..	1·34	1·4	Diag. tension

Tests of Posts. In making the posts for these preliminary experiments, only one rod was used for reinforcement in each case, and the rods were only 7 ft. 8 in. long, thus leaving 2 in of concrete at each end, so that the rod did not come in contact with the crushing plate of the machine. The only reason for reinforcing the posts is to make them portable, as it was considered that the crushing strength of the concrete would be ample for its work, provided that it could be developed. It was anticipated that economic reasons would prevent the use of spirally reinforced concrete columns of such small dimensions. As was expected, the square columns gave better results per square in. than the round columns. Table III. shows the results of tests on the first set of posts made. The results show, however, that there is no difficulty in making, storing, and transporting suitable beams and posts of concrete whose proportions are even as poor in cement as 1 : 2 · 5.

TABLE III. — Compression Tests on Square and Round Posts, Concrete Plain and Reinforced, 8 ft. Long. Age of Concrete (1:2:5) 2 Months.

No.	No. of tests	Section.	Reinforcement.	Crush. Load, lb p.s.in
1	2	8in. 8in.	(Plain)	1,270
2	2	8in. · 8in.	1, ⅜in. ⅜in., 7ft. 8in. long	1,266
3	2	8in. · 8in.	1, 1in. 1in. ,,	1,300
4	2	8in. diam.	(Plain)	1,234
5	2	8in. diam.	1, ¾in. diam., 7ft. 8in. long	1,020
6	2	8in. diam.	1, 1in. diam. ,,	1,068

The cost of these members, when cast in steel moulds and with the simple reinforcement described, can be readily determined, according to local conditions. The posts, being cast vertical, needed hardly any ramming, and from their moderate height presented no difficulties. Both props and beams were easily handled, the beams weighing about 100 lb. per lineal foot.

The considerable increase in strength with age up to 12 months, should be very valuable in the case of mining work. Owing to gradual settlement, the load on the head and side trees in the headings is often found to increase with time. But when great pressures are expected, the only satisfactory method of resisting them is by a complete shell, in which the invert supports the side walls. Here then, are the real opportunities for the engineer in reinforced concrete work. It is necessary to arrange for this work at the beginning, because it would be found very difficult to replace timber work with a reinforced concrete arch, which itself would need to be supported on centres for a considerable time.

At Baggeridge the pit bottom, whose height is 60 ft. and about one-third mile below the surface, is all reinforced work, and also some of the main roads, but though reinforced the concrete is itself very thick, and it will be seen in many cases when reinforced work is underground that little confidence seems to have been placed in the reinforcement. Except on the Continent it is only in a few isolated cases where sections reinforced with steel are made lighter than would be safe with cement only. In arch work designed for compression only, like the original arches on the Monier system, we have a method of construction too little used. I would remind mining engineers of the tests of one of the first Monier arches built in 1890. The test arch was 32·8 ft. span, with a rise of only 3·28 ft., the width being 13 ft. The depth at the crown was 6 in. The reinforcement was iron wire 0·39 in. and 0·28 in. dia. placed 3 in. centre to centre, and 2 in. from the soffit. The following tests were applied :— A locomotive weighing 40 tons was placed at the crown, then it was run off and a locomotive of 53 tons weight was run over half the arch. Having stood these loads successfully, a dead load of 110 tons was piled on the half span, the load being about 1,000 lb. per sq. ft., and the deflection at the centre only 0·12 in. When the load on the half span was increased to 2,000 per sq. ft., the arch failed, but by the abutments yielding. These tests show what a well designed arch of reinforced concrete will stand." —PROF. S. M. DIXON, *Iron and Coal Trades Review*, April 4, 1913, p. 521. (A. R.)

TRANSVAAL TIN DEPOSITS.—"Since the discovery, some six years ago, of casserite in commercial quantities in the Central Bushveld area, numerous articles have been written on these deposits, the latest being those in the *South African Mining Journal* of Dec. 14 and 21, and Jan. 11.

In these articles it has been assumed that these deposits of casserite have originated from 'tin bearing vapours, and gases, which have risen to the surface of the magma' and that these 'tin deposits favour the upper granite horizon.' It is further argued that, as the deposits of casserite have originated from tin bearing vapours and gases, the known occurrences may extend to a considerable depth, 'a depth equal to the lowest working in the Dolcoath mine,' also that as the tendency of tin-bearing vapours and gases is to rise to the surface of the magma in which they have originated and to 'favour the upper granite horizons,' the claims that have been pegged on the lowest horizons are in 'a presumably unfavourable situation.' The views expressed in the said articles are calculated to raise the hopes of the owners of claims on the upper granite horizons and to dampen the enthusiasm of those on the lowest horizons.

The low lying ground forms by far the greatest portion of the Central Bushveld area, and it is fortunate, therefore, from an economic point that the data obtained from actual mining give no evidence whatever to support the theory that these deposits originated from tin-bearing vapours and gases.

About four years ago I first visited the Central Bushveld, but at that time sufficient mining work had not been done to enable anyone to express a definite opinion as to the nature of the tin occurrence, and the probability of its continuance in depth. It appeared to be a secondary enrichment occupying fissures or fractures of limited depth, but none of the fissures had at that time been 'bottomed'; and, therefore, although a careful examination of the mineral, coupled with other observations, warranted the opinion that the occurrence was a secondary enrichment, it was not possible to express a definite statement as to the depth to which these fractures would extend.

I have recently re-visited the region, and find that in the Potgietersrust district in the northeast and also in the Warmbaths district in the southwest, the greater number of fissures have ceased to exist, and that the remaining ones show unmistakeable signs of disappearing. For obvious reasons it is inadvisable to give the names of particular mines.

A careful observation failed to find any evidence that the pipes or fissures were mineralised by 'tin-bearing vapours and gases,' and that the source or 'parent body' is somewhere below the deepest working. There is, however, evidence that the so called 'pipes' or fissures are fractures caused by mechanical stress or strain, and that their cassiterite contents are derived from the erosion or denudation of formerly overlying rocks containing cassiterite. In other words, the rich cassiterite found in these pipes and fissures is a secondary mineral derived from the erosion of rocks that formerly overlay the present surface in considerable thickness.

There is evidence of great erosion in the Central Bushveld area, and careful observation points to the conclusion that, although particular zones of fracture may not have a great extent, either laterally or in depth, yet there are other zones of fracture in the area yet to be discovered, and that in consequence of the immense thickness of the formerly overlying rocks, which contained cassiterite, these yet-to-be discovered zones of fracture will be as rich in cassiterite, as those already found: also that the particular class of rock in which these fractures exist has no bearing on the economic value of the deposits: in other words, these cassiterite bearing fractures may occur in red granite as at Zaaiplaats or in quartzite as at Rooiberg, and that they are as likely to be found on the lowest horizon as on the highest."—G. H. Blenkinsop, *Mining Magazine*, May 1913, p. 370. (A. R.)

Air Blasts in Indian Mines.—"Pausing to consider another aspect of the Kolar field which is becoming of more than local interest, we find that air blasts were experienced with increasing severity during the year.

Of 88 lives lost in the gold mines, 41 were due to accidents caused by air blasts. The Champion Reef was mainly effected, no less than 406 air blasts being reported during the year, one of which killed no less then 21 miners. Largely as a consequence of these phenomena, the death rate underground showed an increase from 3·84 per 1,000 in 1910 to 5·09 per 1,000 in 1911. A seismological observatory has now been erected, and observations taken over a considerable period by Mr. H. M. A. Cooke, of the Balaghat Gold Mining Company. While it is too early yet to express any definite opinions of the results, some definite lessons have been learned. The seismograph distinguishes clearly between air blasts and earthquakes, and indicates that the damage is not in proportion to the amount of earth movement, but of the rapidity of the vibration, as the earthquakes observed have shown much greater movements of the ground than any air blasts but without causing any damage. The results obtained seem to prove that the rocks are continually breaking or snapping locally owing to the state of strain in which they are held being continually altered owing to the underground excavations. It appears that many more air blasts occur than are of sufficient magnitude to be felt. Thus for October, November and December 1911, 5,092 shocks were recorded by the seismograph, but only 47 were noticed by those working in the locality."—*London Mining Journal*, April 5, 1913, p. 326. (A. R.)

Economic Efficiency in Mining.—"In the development of soft ore mines it is usual to raise for the opening up of sub-levels, such raises very often being kept in the ore and allowed to follow the dip of the footwall, the argument being advanced that the ore coming from the raise pays for the labour of putting it up. This theory works all right for the actual work of raising, but if it is intended to drop ore through the raise it should be at least 60 from horizontal to allow a good fall for the ore, otherwise the cost of keeping the raise from blocking up with ore, because it is too flat, and the necessary delay to trammers, etc., will more than offset the cheap cost of putting the raise up; the idea being that for all raises that are to be used for ore chutes, the nearer vertical they are, the cheaper ore can be handled through them, and it will often pay to drift into, and raise through a few feet of footwall, so as to get raises at a convenient angle. A rock raise being safer from caving will have a longer life than one in soft ore, and covering a long period will ultimately be cheaper. Where raises are flat you can usually see one or more men clearing the chute with trammers waiting on them.

A clean ditch, or water drain, alongside of all tramways will facilitate tramming, also mining, for if the miners get their feet wet going to work in the mornings they feel the effects of it, and are not in as good working condition as they would be if their feet were dry and comfortable: while mud and water flowing over the rails makes tramming very much harder, and consequently more costly.

Safety appliances also have a bearing on the ultimate profit of a mine, for personal injury damage suits, and litigation usually resulting therefrom can often be avoided by such expedients as a fence around all raises, open stopes, sumps, etc., and gates around the shaft of each level, or travelling roads: such gates swinging away from shaft to open. The necessary lumber and labour to build such protecting fences and gates would cost very little and should be in place wherever a man is liable to stumble in the dark, or through carelessness.

Electric repeat bells that ring at every station, landing, and in the engine house at the same time, cost very little more than bells of other kinds, and a sleepy cage or skip tender will hear what bells he rings at the time he is ringing them: the sense of hearing guarding against the sense of touch, he immediately detects either a mistake in touch or a faulty circuit; and where these repeat bells are installed the person ringing the signal gets into the habit of counting what he hears, and not what he feels when he presses the button. The landers would also hear what bells were rung and would dump the ore on the correct stockpiles, thus avoiding the mixing of ores.

Wherever chairs are used for skip or cage to rest on, a signal system such as the one used in the Dolcoath tin mine, Camborne, Cornwall, would save money by preventing accidents. When hoisting from more than one level in the same shaft skip tenders sometimes ring down the skip forgetting to draw back the chairs and clear the shaft, the result is usually a damaged skip, broken chairs, and sometimes worse accidents. To prevent this kind of thing the Dolcoath people made an electrical circuit, connecting all chairs with a set of numbers representing the chairs in front of the hoisting engineer; these numbers indicating the chairs and the levels at which the chairs were installed. All numbers being in duplicate in a glass box. Whenever a chair was thrown in the circuit was changed, one number representing the chair burned a red light; the other if the other side of shaft was clear, a white light; if both chairs were in at one level, both lights were red; all other lights burning white. By this means the hoisting engineer always knew what chair or chairs were in, for the red light remained burning as long as the chair was across the shaft. If a skip tender rang a skip to a lower level without drawing the chair, the engineer simply telephoned the level, had the shaft cleared, lowered the skip, etc.; the same thing applied if a chair was thrown in while the skip was below it. The engineer always knew if the shaft was clear or not, and his indicator showed him where his skip and cages were.

Economy in the use of supplies also is a big item. Take the case of the common nails; every mine uses them, usually 60d nails. In more than half the cases a 40d or 5 in. nails can be used with as good results as a 60d, or 6 in nails. In a 100 lb. keg of 40d nails there are approximately 1,800 nails, while in a 100 lb. keg of 60d nails there are about 1,100, being a difference of 700 nails in one keg, or nail for nail, a saving of 39 lb. at 0·021 cts. per pound in favour of the 5 in. nail: this working out to about 82 cts. on a single keg is only a small item, yet it is a small mine that does not use 10 kegs of nails a month; that equals $8·20 per month saved on a single item. And for most practical purposes around a mine a 5 in. nail is as good as a 6 in. Miners are sometimes careless with nails, often laying a few handfulls beside the timbers until the next shift: these either being scattered with the blast or buried with the dirt, and never used.

Possibly the largest single item in the matter of underground supplies is powder. There are a lot of unskilled miners who have a theory that for every foot of hole drilled there should be a stick of powder: they follow this theory irrespective of the percentage of powder used: or the nature of the ground they use it in: or the amount of burden on the hole they are charging: and adding one stick more for luck, or for good measure. Where this class of miner is employed quite a little money can be saved by providing them with powder of a lower strength, as for instance 30% in place of 40%. Supposing two miners drill five 6 ft. holes per shift, or a total of 30 ft., giving these holes one stick per foot, plus one for luck in each hole, they use 35 sticks of powder. 40% powder is approximately 7/10 of a cent per pound dearer than 30%. In a 50 lb. box of powder there are usually from 53 to 56 sticks, or powder weighs nearly 1 lb. per stick. For purposes of arriving at the actual saving, say the 35 sticks used weighed 30 lb., by substituting 30% powder in place of 40%, there would be a saving of 30 times 0·007, or 21 cts. Thus by using a powder of 10% less strength a saving for each miner of 10½ cts. per day could be effected. This only applies to such miners as are unable to use judgment in the amount of powder necessary to obtain the best results. Also by charging their holes to the best advantage the dirt would not be scattered over the drift or stope so much, thus making it easier to clean up, and by apportioning the strength of the powder to the work it must perform, less timber would be blasted out, thus saving time and money in replacing timbers.

One of the most persistent and costly leaks in operation of a mining plant, and one that very often is entirely ignored or overlooked is the loss of lubricating oil through becoming filled with grit or dirt in passing through bearings: less than 40% of the oil becoming worn out, the rest dripping through. An oil filter for purifying gritty or dirty oil, with a daily filtering capacity of 20 gal. only cost about $30, and will save from 60% to 80% of all oil filtered. Figuring on a medium priced engine oil, at say 16 cts. per gal., using 10 gal. per 24 hr., with a 50% recovery, there would be a saving of 5 gal. at 60 cts. or 80 cts. for each 24 hr. the filter was in use. Ten gallons of oil per 24 hr. is a very small amount to figure on, most mines using more than this.

Candles cost approximately a cent each, and miners often leave candles in their boots or clothes in the dry, the heat of the dry (change house) melting the candles, nothing being left but the wick. Some mines allow their candles to run short once or twice a month, and instead of the usual four, the miners are handed two candles each: they somehow manage to put in a good shift's work, and incidentally the mining company is 2 cts. ahead for every man underground in the mine. By this means waste candles are used up, and instead of suffering a loss the company gets the benefit of all candles paid for.

In the use of carbide lamps it is better to fill the holder, or container, only one-third full, than half or two thirds full; being one-third full allows room for expansion as the water drips on the carbide, and unless this is done, when the carbide holder is cleaned for refilling purposes, a small amount of good carbide is thrown away. If this is emptied on the drift bottom where there is moisture, the result is a gas for the miners to breathe, which is very often the case.

In the matter of equipment for drilling purposes, money can often be saved by purchasing small drills, instead of the large 300 lb. machines so commonly used; the small drill will put holes of 5 ft. to 6 ft. in moderately hard material just as quickly as the larger machine; there being no time lost in rigging up, one man being used with a small machine as compared with two on a larger machine; less air being used because a smaller hole is being drilled, and with a very much lower initial cost for the machine. In softer material where hand drills are used, or where auger and jumper drilling is done; a good ratchet mining drill will save money by doing the work faster. Some makers of ratchet drills claim their machines will drill a 6 ft. hole in 45 min., and with only one man to operate, or they will drill 45 ft. per shift: the cost of such a machine being approximately $30. In mines that are furnished with power drills there is often a short crosscut, or a seam of hard pan to be put through where there are no pipe line connections. In such places a ratchet machine would pay for itself, rather than a power drill that would necessitate the laying of air lines, making connections, etc.

Another idea that makes for efficiency is the cooperation or good will of the employees. In any business the good will of the employees is an asset; this is especially true of underground mining, where the men work in isolated places in twos or fours, supervision cannot be constant as could be the case in factories or foundaries: consequently should they be inclined to loaf they can more readily find opportunities to do so: or as is more often the case a competent miner will make a fair showing with very much less effort than an unskilled miner, but will seldom reach his maximum output because he feels that a set wage is all that he will be allowed to make. This method of reasoning is often carried out in actual working and limits the output to the maximum of the second rate miner. To obtain the good will of the miner in this respect it will pay to give the men a genuine bona-fide contract, say if an average cost for the three or six months preceding showed that the actual breaking of rock, or mining of ore, was 20 cts. per ton, the men earning $2·80 per day set a three months' contract at 15 cts. per ton and explain to the men that the more they can earn for themselves the more they earn for the company. The competent miners will gradually let themselves out; probably the first month instead of the usual $2·80, they will make $3·40 or $3·50, and as they gain confidence their rate of pay will increase; in this way the company gets the best the men can do; gets the ore mined cheaper; the payrolls will be larger; but the cost per ton will be lower. Incidentally the second-rate miner will improve in ability because he will know with a greater ability there will come greater earnings; the engineers, landers, topmen, etc., will have to meet the pace set by the men underground, and will be doing more work for the same wage. Better conditions will prevail as soon as the workmen feel that their interest is identical with the interest of their employers."—W. J. CROCKER, *Mining and Engineering World*, April 19, 1913, p. 765. (A. R.)

"LIFE" FACTOR IN MINE VALUATIONS.—"It has been authoritatively remarked that the difference between the boom and slump values of mining shares depends largely upon the rate of interest assumed for valuation purposes. The extremes in mining ventures are probably 6% and 10%, though individual mining conditions would require to be very good and money very cheap to admit the minimum stated, and 7% to 10%, no doubt represent more general limits of variation. Nowadays, with so many investments yielding a safe 4½% or 5%, with an infinitesimal risk of any severe capital depreciation, speculative investors—for in mining the adjective must be applied—in shares of mining companies naturally ask for a higher yield than would have satisfied them a few years back. The untoward experiences of certain of the Rand leading mines, such as the Main Reef West, the East Rand Proprietary, the Wit Deep and the New Kleinfontein, in which capital depreciation has been very discomforting, constitute another factor tending to advance the rate of return demanded by people who put their money into mines. At the present moment, it is not unusual to value Rand shares on the basis of a clear 10%. How does the difference between 7% and 10%, affect the price of a share? The discounted value of a mining share paying 10s. a year for a life of 18 years is just over £5 on the 7% tables but on the 10% tables it is only worth 82s. This difference in the valuation basis assumed may lead to an appreciation of 25% in the price of a share.

Amortisation Over 30 and 40 Years.—In mines having long lives, the amortisation factor is comparatively small; the operation of compound interest is such that the amount required to be set aside each year to replace capital decreases very sharply after 20 years. On the 4½% tables 7½d. a year is required to accumulate a sovereign at the end of 20 years, but if the period be 30 years the amount is less than 4d. Our view is that with durations of over 15 years (amortisation is then about 1s. in the £ per annum), it is fairly safe to relegate amortisation to a secondary position, as it is impossible accurately to gauge other factors of favourable or adverse tendency.

A Short 'Life' and a Merry One! It is not infrequent to hear people justify the high price for a mining share by referring to the long 'life' before the property. But, from a mathematical viewpoint, a short life and a merry one is the thing. Reckoning discount at 8%, a mining share has to have a 'life' of 23 years to possess a present value double that of the share of a mine with seven years' assured duration. The shares of a mine paying 3s. a year and with a life of 13 years, and on a 10% basis worth 21s. 4d., increase the life a couple of years, and the present value is advanced only 1s. 6d., increase it to 20 years, and the price justified is no better than 25s. 6d. In other words, if the life be increased 50% from the base period named, the value of the shares is enhanced only 20%.

Small Increase in Present Value after 25 Years.—As the length of years increases, the enhancement of present value becomes markedly slower. Whether 35 years or 25 years makes little difference on the 10% table, and an extension from 30 to 50 years only justifies the advance in share quotation by half a year's purchase!

Typical Instances.—The table annexed shows the value of a mining share paying 5s. a year, discounting at 10%:—

Life.	Present Value. s.	d.
10 years	30	9
12 years	34	0
15 years	38	0
20 years	42	6
25 years	45	6
30 years	47	0
35 years	48	3
40 years	49	0
45 years	49	4
50 years	49	7

The difference is more marked if, instead of taking a clean 10% discount factor, the valuation is based on a 6% remunerative and 3% accumulative as assumed in some recent valuations by the Government mining engineer, or 7% remunerative, and 4% on the sinking fund investment, as may be preferred by others:—

Life.	Present Value. 7% remunerative. 4% accumulative. s.	d.	Present Value. 6% remunerative. 3% accumulative. s.	d.
10 years . ..	32	6	34	0
15 years	41	6	44	0
20 years ..	48	4	51	6
25 years . . .	53	3	57	3
30 years	56	10	61	9
35 years	59	10	65	4
40 years	63	0	68	3
50 years	65	3	72	6

It will be observed that the share of a mine with a life of 20 years is worth only half as much again as the same dividend lasting for 10 years. After 20 years, five years' dividends of 5s. are worth at present only 3s., or 5s. 9d. per share, according to the valuation selected. The shares of a mine lasting 50 and paying a regular 5s. a year are worth only 12s., or 15s. 3d. (according to the table on which the calculation is made) more than if the life were only 25 years. Five years' dividends after 40 years are worth little more than five months' purchase.

The general conclusion from the foregoing is that after 20 years or so the 'life' factor does not greatly improve the present value of a mining share."—*Australian Mining Standard*, March 27, 1913, p. 258. (A. R.)

PRINCIPLES OF MINE VALUATION.—"Almost all enterprise is based on valuation of some kind. Practically all transactions in mining property, whether in stocks, bonds, or actual ownership, are based on some kind of an appraisal. You cannot buy intelligently nor sell intelligently until you know how to form an estimate of what a given property is worth to you, to the man you are dealing with, and to anyone else who might be brought in.

This generalization has no more application to mining than to any other form of business; but it certainly does not apply to mining less than to other forms of business. I think that no more unfortunate idea is taught to young mining engineers than that they are to consider themselves merely as professional men, like doctors or ministers. I would not speak of this if such an idea had not been cultivated in certain quarters. It is true that some mining engineers are employed to do technical work as referees between conflicting interests. Professional honesty is, of course, an absolute essential, but the honesty of a mining engineer is the honesty of the business and of common sense. It is not in the least desirable for a mining engineer to avoid business transactions; on the contrary, he should go into them as soon as possible and study the subject in every possible way. I do not see how any man can make a success of mining who does not realize that it is a strictly commercial business, that the only result to be sought is commercial success, that the only scientific and technical attainments that are worth anything are those which contribute to commercial success. Attainments of a different character are, of course, worth while; but they are valuable to a mining engineer because he is a man and a citizen, not because he is a mining engineer. They would be equally valuable in every kind of profession. Success in business is measured in money. Technical skill is efficient only if it pays. One can judge of the value of technical excellence only if he is prepared with a knowledge of what constitutes the money value of the results.

Undoubtedly this view of the mining business, or profession, has been gaining ground in our universities and technical schools. Formerly, I think, education was considered to be merely a training in principles. Technical training was a discipline in the science of action (I coin the expression), an exposition of the principles of doing things. Whether it was worth while, in dollars and cents, to do a thing was to be found out later in actual business. Now I think it is recognised that, while making money is of course a matter of actual occupation in business, there are certain general principles underlying the value of things, which are well worth study. This is the study of political economy, which is cultivated in all our universities. The mining engineer sooner or later comes into contact with this subject and is bound to find it interesting and important. I shall try now to advance a few ideas about the valuation of mines, which I think are basic enough to come under the head of general principles.

The first thing to which I wish to draw your attention is the general subject of valuation. What is it that causes a price to be fixed for commodities or properties? In general, there are two methods of fixing prices; one is the empirical method, and the other the constructive. The empirical method of fixing prices is simply to obtain a record of facts—of transactions. A man will assume that a town lot is worth $100 because he finds that people have bought and sold similar lots for $100 apiece. Nearly all commodities of a retail nature are valued in that way, and it is perfectly logical that they should be, because each actual transfer is a bargain between the buyer and the seller, theoretically at least. The man who sells for a certain price does so because he believes he can do it profitably. On the other hand, the man who buys computes in the same way. Consequently wheat, iron, and copper, as commodities, are valued empirically by merely ascertaining what sales are made; and fluctuations of price are established by supply and demand.

The constructive method is that which determines the value of commodities in a more or less scientific manner, builds up logically an estimate of what it properly should be worth. This naturally applies to properties which are dealt in only occasionally, for which there is no precedent of an actual sale of the equivalent thing. You easily see that mines belong to that class of property.

Hardly any two mines are exactly alike; therefore you cannot say logically that because one mine is worth $1,000,000, another mine is also worth $1,000,000; at least you cannot say so until you have gone through the factors which give both properties value, and have found that they are identical. It is to those factors which do give value to mines that I wish to call your attention now. I have always supposed, and I think that nearly all mining engineers agree with me, that there are three or four fundamental factors about a mining property which determine its value. These are:

1. The average cost of securing the product of that mine.
2. The average sale price of the product.
3. The rate of production of the mine.
4. The time for which the output can be maintained.

I do not intend to go into detail as to any of these factors, but to point out some things which I think are worth considering, in the hope that later you will think about them and learn the details for yourselves.

In regard to the average cost: In the case of going concerns, that average cost can be obtained empirically by simple reference to the books; to what has actually been done on the property. That seems and sounds exceedingly simple. As a matter of fact it is not easy. I made the mistake of once thinking that it was. I bought a mine in Joplin, Missouri, to be specific, on a record of its working cost for six months. The reason I did not go further back was that older records were not to be had, and also because during this particular six months the property had reached a substantial basis. I figured that concentrate from that mine could be produced for about $20 per ton, and I also estimated that the property should produce so much ore, and that the price would be a certain figure. It happened that development proved that I was right about the price and quantity of ore on the property, but they also proved that I was wrong on the question of costs. It cost us nearly 75% more to produce concentrate from that ore than the record of the property showed. Those were times when we produced concentrate for $15 per ton, but that was not representative of the whole period of working the mine. I think you will find to-day that a great many of the cost statements given out by different companies for fixing the value of their stocks are only temporary or occasional costs, and will not stand analysis.

Then there is another factor in which people deceive themselves, unconsciously, but in the simplest and most obvious way imaginable. They keep on doing it, always have done it, and I suppose always will do it. That is, by failing to see the relation between expenditures that are for plant and those for keeping it going. In keeping books it is very easy to say that the construction of a building, the remodeling of a mine, or the building of a new mill, is something that will not happen again, that it belongs to the property and not to the operation: to a certain extent that is true, but it is a fact, which I have verified by looking up a great number of mines, that the so-called operating costs are ordinarily not more than 50% to 80% of the total cost. Until a mine is shut down, there is always some expenditure to be undertaken which a bookkeeper may think will not be repeated, that it belongs to the plant, but these things keep occurring so constantly that one who is figuring on costs must count on them.

Another thing I believe to be true is that any estimate of cost, or valuation, or anything relating to the future, is a good deal of a guess. One who thinks he can estimate a cost, whether of a going concern or a concern to be started, to a fraction of a cent, or to any fixed amount, is simply mistaken. All you can do is to put a round figure which seems to be pretty near the facts. On certain mines the fluctuations of price in less than a year corresponded to an extension in the life of those mines from 5 or 6 years to 35 or 40 years. Here again I am dealing in a mere approximation. I suppose these mines produce about 20 lb. of copper per ton: therefore to get 200,000,000 lb. of copper per year, you would have to mine 10,000,000 tons. Therefore, the figures that I actually used meant that I believed there was 130,000,000 tons of copper ore which could be depended on. That is an enormous amount of ore. Figured roughly, that would mean a lode of solid ore five miles long: it was actually obtained by projecting the production of the mines and the development of the mines far below their present position. Now if I had to estimate on 12¼c. copper, I would have had to figure on 40 years' life, or I would have had to believe that there were 270,000,000 more tons there than I thought I was justified in believing. On the other hand, if we put the value at 17¾c., I should have had to figure on only 50,000,000 or 60,000,000 tons of ore. Therefore, I say, as affecting the value of mines, the actual fluctuations of price within one year were equal to an uncertainty in the ore production in those mines of more than 300,000,000 tons. An engineer is not likely to make a mistake like that when a mine is so fully developed as these Michigan properties, many of which are worked to a great depth, are well managed, and have their outlines very well shown. He could, however, easily make just as important a mistake as that in figuring on his average price of copper.

As a matter of fact, I do not think the 14c. price I used was an accurate figure. I believe that copper will average more than 15c. per pound taking into view the development of the business, but 14c. was as far as I thought I could go. I do know that mines have been valued within the last five years on prices estimated all the way from 12c. to 20c. Just imagine what a difference that makes in the estimated value of those mines. I know a very important mine in Mexico, the organizers of which believed that copper would not average less than 20c. per pound. I know of mines, on the other hand, that are being valued on an expectation of 12½c. per pound.

I ran against the same difficulty with another group of mines during that same Michigan study. In the iron mines I assumed a price for iron which happened to be almost the same as the price for the year 1911, which was a dull year in the iron business. In this case, however, the price, instead of going up, as the price of copper did, went down about 50c. per ton the next year. Now the margin of profit for those iron mines during the period that I figured on was just about $1 per ton, so that if you take off 50c. per ton you cut the profits in two. Of course, during all this last year, I have been exposed to a good deal of criticism by the iron miners, because they say I assumed a wrong price. While it may have been wrong for the year 1912, I notice that prices for the year 1913 have been fixed lately and at a figure very close to my average, in fact, almost identical with it if certain reductions in freight rates be considered.

In both the iron and copper calculations, I have been asked a great many questions as to how I arrived at the prices I assumed. Here there are certain evidences which can be used as guides, and are well worth studying. One general fact about the prices of all mining products is that they vary in cycles or in waves, irregularly of course. If, after a period of high prices, you find that prices are definitely going down and reach a new level, it is pretty safe to believe that the lower level of prices will continue several years. On the other hand, after a period of low prices, when once the metal rises markedly, it is pretty safe to assume that it will stay at a higher level for several years. Those fluctuations are things well worth studying. It is a matter on which you cannot at once form opinions.

The valuation of a mine, while in the long run it depends on the average, is influenced a good deal by the present price. If you have a copper mine that is going to last ten years, and you expect 18c. copper for three years out of those ten, it is a matter of considerable consequence to your property whether that 18c. price comes during the first three years or during the last three years of the life of your mine. If it comes in the first three years you will get your profits and may invest them, perhaps in other directions, and the interest on that money for the following years will be worth a great deal to you. I must confess, in regard to the question of profits, that I was wrong when I wrote 'Cost of Mining,' in laying too much stress on average price. I now believe the public is right when it puts a higher value on properties during periods of high prices for metals. That is merely an assertion which I offer you to think about. It is a curious thing, which any one can observe by watching stock quotations, that the prices of mining stocks vary almost with the price of the metal itself. When copper metal goes down, copper stocks go down. I always used to think that this was illogical, but I have come to the conclusion now that the public, which bases its valuations on present commercial conditions, has been more logical, on the whole, than the mining engineer.

The question of the life of a mine is not to be overlooked by any means. In many cases that is the most vital factor, particularly in the case of gold mines, because fluctuation of price does not then enter into the problem. Take, for instance, the Goldfield Consolidated. That property has made a spectacular showing in dividends, but it has always been recognised that its life was uncertain; and, now, in spite of the fact that for two years it has been paying very large dividends, the price of the stock has been so low that the dividends have equally 50% of the market value. Recently the stock has been selling for $1.75 per share: the actual dividends paid during the past year have been $1.60 per share. Now, if the mine were valued in the way some copper mines are valued, paying dividends, say, at the rate of 6% of the selling price, the price to be put on Goldfield stock would be $20 per share. In that mine the whole question centres on the length of its life. I speak of it because this is so contrary to usual experience. I do not know of any other case in which the public has been warned, or has become convinced of a mine's exhaustion so long in advance.

On the other hand, there are some mines with an expectation of long life which cannot show any ore reserves at all. Certain mines at Tintic, Utah, which I should be willing to value on an assumed life of five or ten years, do not show ore reserve for even three months. The reason is that the form of the orebodies is such as to prevent ore reserves from being developed. The ore occurs in an immense mass of limestone, through which mineralizing solutions have passed, and have been diverted by various intersecting fissures. The ore bodies are irregular, sometimes small and sometimes swelling to large size; sometimes they go down vertically 400 or 500 ft., forming a pipe, then abruptly go off horizontally for 400 or 500 ft., and and then up again, then shooting to the right or to the left. If nothing were known about the persistence of these orebodies, one would be justified in being exceedingly sceptical about the future; but the experience of 30 years has shown that these deposits, in spite of their irregularities, are exceedingly persistent. In making his valuation the intelligent engineer will, of course, consider this fact a pronounced asset." JAMES R. FINLAY, *Columbia School of Mines Quarterly; Mexican Mining Journal*, April, 1913, p. 138. (A. R.)

MISCELLANEOUS.

INFLAMMABLE DUSTS.—"The Home Office has issued as a White Paper the report of Dr. R. V. Wheeler (chemist to the Committee investigating the causes of explosions in coal mines) on the results of experiments he has conducted on the inflammability and capacity for transmitting explosions of carbonaceous dusts liable to be generated on premises under the Factory and Workshops Acts. It may be remembered that the experiments were arranged as a consequence of the two serious explosions at a provender mill in Glasgow and an oil cake factory in Liverpool that occurred in November, 1911.

Sixty-six samples of dust, collected by Factory Inspectors from beams, ledges, and other resting-places for dust, which they have come across in the pursuit of their ordinary duties, have been tested, the experiments being directed (1) to discriminating between harmless and dangerous dusts and (2) to ascertaining the temperatures at which inflammation of the dangerous dusts takes place readily. As a result of the tests made, the dusts have been divided into the three following classes:—

Class 1.—Dusts which ignite and propagate flame readily, the source of heat required for ignition being comparatively small, such, for example, as a lighted match. This class includes sugar, dextrine (calcined farina), starch, cocoa, rice refuse, meal and sugar refuse, cork, unextracted soya bean, wood flour, malt, oat husk, grain (flour mill), maize, tea, compound cake, grain (grain storage), rape seed, cornflour, flour (flour mill), chicory, briquette, gramophone record, and extracted soya bean.

Class 2.—Dusts which are readily ignited, but which for the propagation of flame require a source of heat of large size and high temperature (such as an electric arc), or of long duration (such as the flame of a Bunsen burner). They include Copal gum, leather, 'dead' cork, cocoanut oil milling, rice milling, sawdust, castor oil meal, oilcake, offal grinding (bran), grist milling, horn meal, mustard, shoddy and shellac composition.

Class 3.—Dusts which do not appear to be capable of propagating flame under any conditions likely to obtain in a factory; either (*a*) because they do not readily form a cloud in air, or (*b*) because they are contaminated with a large quantity of incombustible matter, or (*c*) because the material of which they are composed does not burn rapidly enough. These include organic ammonia, tobacco, spice milling, drug grinding, cotton seed and soya bean, bone meal, coal (foundry blacking), lamp black, sack cleaning, retort carbon, rape seed (Russian), blacking, grain cleaning, charcoal, foundry blacking, brush carbon, stale coke, plumbago, bone charcoal, and mineral and ivory black.

After describing in detail the apparatus used, the report proceeds to tabulate the various dusts and to indicate the ignition temperatures. It is remarked that the latter are relative, and refer only to the particular set of experimental conditions employed. For the sake of comparison the ignition temperatures of most bituminous coal dusts, as determined in the same apparatus, are given as lying between 1,000° and 1,100° C. So far as possible the different dusts (all of which were dried during one hour at 107° C.) were obtained of the same degree of fineness by passing through a 200 mesh sieve, but in some cases this was not possible owing to the fluffy nature of the substance, while in other cases the sample received contained no such fine dust. The report observes that the ignition temperatures in such cases have no relative value, and the experiments made serve only to show whether or not the substance is dangerously inflammable. We append a few specimen results with the remarks thereupon:

Lamp Black.—No ignition. Could not be sieved.

Rape Seed.—Ignition temperature 1,050° C. (probably lower, but no more dust available). Did not sieve; sample too small.

Sawdust.—Ignition temperature 970° C. 20% of sample passed through sieve.

Blacking.—No ignition. 75% of sample passed through sieve.

Starch.—Ignition temperature 1,035° C. 80% of sample passed through sieve.

Foundry Blacking.—No ignition. The whole of sample passed through sieve.

Briquette Dust.—Ignition temperature 1,090° C. 90% of sample passed through sieve.

Flour.—Ignition temperature 1,060° C. 90% of sample passed through sieve.

Charcoal.—No ignition; a few sparks only. 90% of sample passed through sieve.

Coal (foundry blacking).—No ignition: a few sparks only. 70% of sample passed through sieve.

Plumbago.—No ignition. 95% of sample passed through sieve.

Bone Charcoal.—No ignition. The whole of sample passed through sieve."—*Iron and Coal Trades Review*, April 4, 1913, p. 523. (A. R.)

THEORY AND PRACTICE.—"It is probably within the recollection and experience of many here that those who press, and quite rightly press, the importance of scientific and theoretical study as training for an engineering career, also plead very hard for an adequate recognition of their efforts from employers. They invite their cordial co-operation, and sometimes scold them for a backwardness in affording it. Their aspiration for co-operation can be given the fullest sympathy, but I am none the less inclined to take teachers themselves to task with regard to some of the advice they give, either explicity or tacitly, to their students. Too many young men adopt the idea when they have submitted themselves to a course of technical study, and have perhaps displayed conspicuous ability therein, that they leap from that, and that alone, to a level which at once entitles them to look for and expect a lucrative position in engineering works. This is entirely erroneous, unless they have an actual practical training as an additional asset. I do not complain of the natural ambition which probably helps towards such an idea on the part of the young man, but I do seriously complain of those teachers who foster such an error. If teachers meet occasionally with an apparent lack of appreciation on the part of employers of the theoretical and scientific training which they have given to their students, I think that, if inquiry were made, there may be something to be said for the employer. It will very possibly be found that the employer regards such half-trained men with some suspicion, and possibly even he may have suffered from an engagement of such a young man to an extent which leads him to say that, however much he desires a man thoroughly trained both in theory and practice, he cannot afford to employ a man trained only on the former line; and that if he can only get training on one line, he would prefer the practical to the theoretical man.

For myself, I desire to secure the services of men who are thoroughly trained, and I speak from some experience of their value; but I regret that I also speak from experience when I say that there is too often a neglect of the practical workshop course. I am convinced that such an omission of an essential part of the training is most disadvantageous to the student himself, as well as to the ultimate strength of the profession as a whole. One of the results of this tendency is that, when a vacancy has to be filled, it is by no means easy to find a man who is equipped with good theoretical and scientific training coupled with a thorough workshop training, and ability to apply the one to the other in an efficient manner. A good portion of the blame for this must lie at the door of some of the teachers, and I can only hope that any to whom such charges apply, directly or indirectly, may without delay recognise that if they desire, as of course they do, the co-operation of employers in forwarding the interests of their students, they must see to it that they not only avoid the inculcation of such erroneous ideas into the heads of those under their charge, but also must adopt the line of impressing on them that, for a practical profession, they must be practical men, and, in order to attain that, they must submit themselves to thorough practical as well as thorough theoretical training.

One may ask oneself how it happens that men who spend their energies as whole-heartily as the teachers in engineering colleges and the like do, in the interests of the student entrusted to their care, make the mistake which I am suggesting they do make. There can doubtless be several answers given, but probably the true one is that there is something wanting in the training of many or most of those of whom we complain.

If we look round at the leading lights of the engineering educational firmament who have themselves, before embarking in teaching work, not only passed through a thorough workshop training, but have followed that by a considerable amount of practical work and experience, we shall not, I think, find that they are offenders in the sense of which I am speaking. It would seem that it is rather those who have not themselves undergone a workshop training—or, if they have done that, have taken up teaching before they have accumulated an adequate supply of practical experience—who are most prone to decry the importance of practical training and experience, and thus by their explicit or tacit advice throw impediments in the way of a young man carrying his education through in the most complete manner. Such men may be, and probably are, thoroughly conversant with theory, and most perfectly competent to teach that side of their subject; but if they are themselves lacking in practical training and experience, it must necessarily follow that their powers of imparting knowledge of the application of theory to practice must be stunted and narrowed. If a man is to be able to teach the application of science to practice efficiently, he must himself be not only a scientist, but a practical man as well. In the interests of the teachers themselves, and of the students under them, this surely ought to be recognised.

If it could be arranged that every teacher could, concurrently with his teaching, be engaged on practical work of a nature allied to the subject taught by him, great advantage would undoubtedly accrue to the teacher and to the student. Such a departure would, of course, entail, considerable reform in the present methods of teaching, and the allocation of the teacher's time; but that need not be regarded as impossible, because it has, in fact, been found practicable in other countries.

Of course, in the larger educational centres the teachers do have some opportunity of keeping in touch with practical needs from various investigations and researches they are invited to make in the educational laboratories; but if they could, in addition to this, be actually engaged on practical, or even commercial, work as well as laboratory experiments and investigations, the authority with which they could direct the studies of their students must undoubtedly be enhanced.

I hope I shall not be thought to have spoken too critically, but if an excuse be necessary, I would plead the great importance which the subject has for the whole profession; I will on that account even go further and crave indulgence for addressing also a few cautionary remarks to students and young men.

I have not infrequently met cases of young men who, when questioned as to their practical training, advance the statement that the work they have done in their college workshops and the like is equivalent to a workshop training. I have nothing to say against the existence of such a college workshop as a place where a certain amount of manual dexterity may be acquired as a sort of relaxation from the regular work of the study or the lecture-room. It cannot, however, be regarded as in any way approaching what is really required in the way of practical training, or as capable of giving any approximation to the value of the experience to be gained from work in the shops under real working and commercial conditions.

Supposing that the statement, that the importance of practical work is very great, will not be seriously controverted, I must add that all the advantages cannot be regarded as fully secured even when courses of theoretical study have been combined with workshop training. When the primary courses have been completed, the education of an engineer must still go on by the acquisition of experience, and from this point of view the engineer's education is never finished.

Sir A. Trevor Dawson, in an address he delivered last autumn, seems to me to urge his hearers very strongly on these lines, for in more than one sentence he pointedly refers to the importance of the acquisition of practical experience by young men, even at a temporary shortening of salary. For instance, he said: 'For the moment, it is enough to establish that experience is a dominant consideration. This cannot be too forcibly emphasised, for young men are at times prone to seek advancement without carefully considering whether that which looks like advancement will conduce to the winning of the experience so invaluable in future work.' To this I cordially assent.

In my opinion, before such practical experience can be properly assimilated, a workshop training is essential. Its value is very great all round, for it covers a great deal more than manual dexterity. For instance, it affords a man an insight (not otherwise attainable) into the working of men's minds, men, perhaps, of rather a different class from himself, but who, undoubtedly to start with, are head and shoulders above him in their ability and in their methods of handling and carrying through particular, and it may be very important operations. Moreover, he will, of necessity, from working at the bench and the like, even if he keeps his eyes only slightly open, become acquainted with the point from which various matters are viewed by the operative, and he will undoubtedly find that study is not without its uses in after life. He will recognise the fact that the user of a machine will be most cognisant of the peculiarities of the particular piece of mechanism with which he has to do, and it is useful to a young man to watch and see how such a man will care for and nurse the peculiarities of his machine in order to secure the best results. Of course, in this case I am speaking of a good workman, of whose class we may be thankful to-day we have a fine supply; but the young man will also find it instructive and useful to study the doings of a less careful or even a careless workman, of whom there are also some, as he will thereby learn what to avoid.

Both theory and practice are needed so to distribute the various qualities of labour which are required for manufacturing, as distinct from fitting, operations. Cheapness of production must necessarily follow manufacturing lines, but all classes of labour are required for proper organisation. The high class fitter, though required to as great an extent (even if not greater) as in the past, is less prominently apparent as a direct producer, but he really is one of the main sources of power to enable cheap manufacture to be effected. He might be compared to the properly laid rails over which the traffic of cheap manufacture can run smoothly in the hands of men possessing very much less skill.

The less skilled and therefore cheaper labour is apt to be regarded by the casual observer as the main and actual producer, and the highly skilled preliminary and maintaining force is sometimes too much neglected. I must therefore not be understood to advocate cheap labour as against the more skilled, and therefore more expensive labour, though I admit that present day requirements do involve a change of direction in the employement of the latter; at the same time it is well to make it clear that not only is the highest quality of skill required on the part of the workman of the best class, but even more of the class are needed than formerly.

Not only is this so, but the increased demand which follows cheaper production leads to the employment of a large increase in the number of operatives who, though partially skilled have not attained to the level of the highest class. I base this statement on a large number of statistics which were compiled and published some years ago by the United States Government. It was shown thereby that, no matter what the product investigated might be, a large output at reduced selling price led to an enormously increased demand for that product, with the result that not only was there a large increase in the number of men of the highest skill required, but there came into existence an immense volume of work with good wages for operatives of less skill, for whose services there was no use under the old system. Any methods, therefore, which will have such a result must be beneficial not only to the trade of the country, but to the community at large, and any employer who so directs his works that he is able to pay wages on the highest and widest scale, and yet produce at prices capable of competing on favourable terms in foreign markets, is little short of a true benefactor to his country, and is well deserving of the profits of his enterprise which he is able to secure. I say that to attain such results he must not only have and use brains himself, but he must employ brains of high practical efficiency in all branches of his undertakings.

There will be no dissentient voice against the statement that that undertaking is best equipped which has in its administration not only high quality of business ability, but a firm basis of scientific and practical knowledge to guide its actions in framing the policy and giving it first-hand power to form clear and reliable opinions as to the correctness of the operations carried out by the executive under its control. It seems in fact a truism that the administration must be fully informed of a case before it can effectively lay down a policy to be followed. There are few more fruitful sources of loss in working than to lay down a line of policy on insecure foundations, or on lines which are either totally unpracticable of realisation without excessive cost, or which are so far unscientific as to shake the confidence of the administration. Similarly, the executive must be equally conversant with scientific principles as well as thoroughly master of all practicable technical details. In a word, knowledge and practical experience must be the undoubted possessions of both branches, for there is no part of any engineering establishment which will not be more benefited by the services of men holding this combination of qualities than by men qualified only with practical experience. Conversely, there is, in my opinion, no possibility of an establishment being successfully worked by men having only a scientific and theoretical knowledge without practical training and experience in addition.

Let us now try to follow up another line. Every undertaking which aims at keeping at least abreast of current requirements must be prepared to spend some money in experiments, investigations, research and the like, and there can be no question that, on the one hand, in the inception of such work, in its arrangement and its operation, the presence of the mind scientifically trained is essential, and that for practical work, on the other hand, the scientists employed should also be practical in their methods of approaching and handling the investigation of research, and in the application of the results presented. Some will consider that the branch for carrying out this class of work should be primarly under the direct control of the administration, while others will hold that it is more essential to the executive and should be primarly part thereof. The results of the work are undoubtedly wanted by both; by the administration because its policy may be affected, and by the executive because its economies in working may be materially assisted thereby. On the whole it would probably be best to entrust the control of the branch to the executive, with an arrangement that all important results should reach the administration simultaneously with the executive; among the reasons for this preference might be instanced that the executive will find it advantageous to have a considerable number of small experiments carried out which could hardly be honoured by so high a title as 'research,' but which none the less have great economic value. For such work the executive would be primarily interested.

Any research of a larger or more expensive nature might and probably should only be undertaken with the concurrence of the administration. In fact, in order to prevent the expenses of the branch becoming so great as to be overburdensome on the whole undertaking it might be found reasonable that the administration should keep close financial control of such work by laying down that not more than a definite sum should be expended on any one investigation or research without its specific authority. This is the system in use in many places, and has been found to work with satisfactory results." SIR FREDERICK DONALDSON, Presidential Address, Institute of Mechanical Engineers; *African Engineering*, May 1913, p. 157. (A. R.)

REGISTRATION AND TRAINING OF ENGINEERS.—"The problem of the registration of engineers has been exercising the minds of engineers throughout the world for years past, and no solution, or even partial solution, has been reached. At present the only possibility lies in the formation and development of such associations as the Institute of Civil Engineers, the Institution of Mechanical Engineers, and many others at home, and, in a smaller way, our own Association here: and in the admission to those associations of only and all such engineers as possess real engineering qualifications. In this way membership could only be attained by the qualified man, and such membership would then be the hall-mark of efficiency. At present any form of registration, such as is in force in the legal and medical professions, is out of the question. Every fitter, mechanic or plumber is an 'engineer.' Every dreamer who schemes out a rotary engine, a flying machine, or a perpetual motion freak, is also an 'engineer.' The only solution would seem to be the adoption of some title other than that of 'engineer,' by which all properly qualified men, and only such, might be known. At present there is a movement on foot, here in Sydney, amongst the representative engineers, to formulate some scheme of registration, and with this movement your Council is associated. But the prospect of even partial success is, unfortunately, very remote. It was decided, however, at a recent meeting of your Council, to invite the earnest thought and consideration of members on this great question, so that, with engineers in other parts of the world, we may do our best to forward the introduction of a very necessary reform.

I have referred previously to the inventor, and his claims to be registered as an engineer, and in this connection I read with interest some remarks made by the Hon. Sir C. A. Parsons in his Presidential Address, before the North-East Coast Institute of Engineers and Shipbuilders. In discussing the inventor, he made the following remarks:

'In all development of machinery and shipbuilding, invention bears its part, and it is interesting to recall what Lord Fletcher Moulton once said when he was discussing the merits of all patents. He had seldom or never known of an invention which appeared in the full panoply of perfection, but it was generally to be likened to a young sapling in a great forest of large trees. It struggled for existence amongst them, they deprived it of the sun, their roots robbed it of nutriment, and its chance of growing up and taking its place amongst the great trees of the forest is remote, unless, perhaps some unlikely circumstance may assist it to gather strength. So the invention by itself, has generally little chance, unless it is suitable to the times, and assisted by some friendly hand, in the shape of brains and capital. There is no doubt that, in these days, useful inventions come principally from trained brains, and the number of brains that have passed through schools, advanced schools, colleges and universities, increases yearly, and concurrently every year the forest becomes denser. Consequently, we have to dig deeper, and training and specialising have become more necessary for success in any line. One hundred years ago it was possible for one brain to master many subjects; now the range and complexity is so great that it has become imperative to specialise, if more than an ordinary superficial knowledge is required.'

From these remarks, delivered by so eminent an engineer and inventor as Sir C. A. Parsons, one must realise that the day of the 'accidental invention,' as one might call it, is passing, and that useful inventions must become more and more the fruit of long and continued thought and study, by trained brains. And this leads us up naturally to a consideration of the supply and development of trained brains. In considering this question, certain axioms may be generally accepted.

(1). All training, to be complete, must embrace an apprenticeship in both theoretical and practical work.

(2). All boys should have an opportunity for showing their fitness for advanced training, such fitness to be generally determined by their aptitude and progress in the elementary schools.

(3). It is absurd to attempt advanced training with the average dull boy. As one writer puts it:—

'It would be no more mad to train a dray-horse for the Derby than it is to attempt to cram good, honest dull lads with the 'high-falutin' nonsense which goes to fill the average day in our elementary schools.' (Referring here to English practice.)

And the same writer again says:—

'For the boy whose limitations extend only to the three R's devote the shining hours in perfecting his knowledge in these subjects; but to him who knows no limitations, let the door be open wide. If he be above the average intelligence, let him take the extra subjects, and let him be one of a small class, under the charge of the best teacher on the staff; give him every facility for making himself proficient; give him every encouragement; let the benefits of the secondary school and university be open to him, and don't let the want of money stand in the way of his progress. He is a paying proposition. He will become a valuable asset to the State.'

Sir Trevor Dawson, managing director of Messrs. Vickers, Ltd., even goes further. He asks:—

'Was it too much to expect that, as there was a great national need, the State should extend its educational system so as to enable the best youths to enter upon a combined practical and theoretical training for the greatest of manufacturing industries, without requiring great sacrifices on the part of parents, who might be unable to support their sons throughout the prolonged period of training? There were details to be considered, such as the choice of the State industrial apprentice, the provision of methods of testing their earnestness, application and proficiency from time to time during the course, so that any retrogression might be visited by the withdrawal of the State support, and the assurance that State industrial apprentices would continue to serve the Empire only after their training had been completed. But all such details could easily be arranged when once was accepted the principle of selecting and nurturing the most promising mental talent of the nation, independently of the stratum of society in which it was discovered. The duty of fostering the mechanical science was one which would yield the highest reward to the nation.'

Far-reaching as these propositions admittedly are, it is to be noticed that State endowments are proposed for the training of only such youths as display mental capacity and aptitude for an engineering career, and that their retention of such State aid throughout their period of training is conditional upon the maintenance of interest and progress in their studies.

To my mind there is much to recommend such a proposition. There can be no doubt that a certain percentage of lads are not sufficiently endowed with brains to warrant any attempt to cram them with anything but the most elementary subjects. In such cases it must surely be better to devote the whole of one's efforts to perfecting the lad's knowledge in the most elementary work, so as to fit him to take up a trade requiring only a certain amount of intelligence, beyond which step he could probably never hope to advance. Again, there is another percentage, and unhappily in Australia it is a very large one, of lads who, whilst undoubtedly possessing good average ability and intelligence, and whose brains are capable of being trained to greater knowledge, at the same time are entirely lacking in application, and whose only idea is to get through the day's 'work' somehow, or anyhow, to spend the rest of the twenty-four hours in idleness or amusement. The blame for so high a percentage of this type of lad must largely be laid at the doors of the parents; but, wherever the blame lies, the result is the same. It is practically useless to expend money and energy in an endeavour to assist such lads as will make no effort to assist themselves. To this class, unfortunately, a large proportion of the apprentices one sees in the workshops belongs. They are without ambition, and without any wish or anxiety to rise to higher things, if it involves additional work. Their chief aim is to get through their day's work, to do as little work as possible during the day, and to get paid as much for it as they possibly can. From this source the bulk of our mechanics are drawn, and it is, I think, because of this failure on the lads' part that, to-day, the supply of really competent and skilled tradesmen is so short. There is, however, a fair percentage of lads possessing quite average intelligence, and in many cases considerably more than average intelligence, who are ready and willing to make some effort to advance their knowledge and rise in the world if given a little encouragement and the necessary opportunity, and there can be no doubt that it is the duty of the State, backed by the engineers, to encourage such boys, and to afford them every possible means for the training of their brains, and this conclusion leads us directly to such a proposition as that put forward by Sir Trevor Dawson.

The question is one which is at present greatly agitating the minds of employers and engineers at home, and the consensus of opinion amongst such men at present is to the effect that existing systems in England are hopelessly extravagant when compared with the results attained. Here in Australia the average lad is, I think, more self-reliant and more quick-witted than his British brother, but the obvious lack of control exercised by parents again handicaps our boys, unfitting them for steady and self-sacrificing work in the subsequent training of their brains. The whole future of the engineering industry in Australia must largely depend upon the supply of efficient men, both of those more highly trained, to take leading positions, and those thoroughly skilled in the various trades. Without such a supply, our industries cannot long hope to compete with those of other parts of the world."—G. A. JULIUS (Presidential Address, Engineering Association of New South Wales), *Australian Mining Standard*, March 27, 1913, p. 266. (A. R.)

FIRST AID FOR CYANIDE POISON.—"A patient's life depends on your prompt action, as the poison acts quickly. All cyanides are deadly poisons, and the prussic acid gas liberated from the solutions, if inhaled, causes almost instant death, and when diluted with air, causes dizziness, faintness, and a depressing frontal headache.

Dilute solutions of cyanide are very poisonous when taken internally, and when they come in contact with the skin, produce, in some persons, eruptions of painful red boils.

If the poisoning is the result of inhaling prussic acid gas, proceed as follows:

Dash water in the patient's face; start artificial respiration; make the patient inhale either a small quantity of ammonia, or a small quantity of ether, or a small quantity of chlorine gas which can be quickly made by sprinkling a small portion of chloride of lime (bleaching powder) on a piece of flannel cloth, moistened with acetic, and then holding the cloth to the nostrils of the patient.

If the poisoning is the result of taking cyanide solutions internally, place the patient in a hot bath, and apply cold water to the spine and neck, providing no delays are permitted to intervene in carrying out the previous instructions.

Incite vomiting by tickling the back of the patient's throat with the finger, or with a feather, or by causing him to drink luke warm water, or strong mustard and water; or by the use of a stomach tube and hot water; or by some physical means.

Give ammonia water solution diluted) or nitrate of cobalt solution (diluted); or peroxide of hydrogen solution (diluted), or give carbonate of iron (freshly precipitated), made by mixing equal parts of ferrous sulphate and sodium carbonate and administer it internally to the patient at once

Or follow special directions accompanying antidote in cabinet compartment number 4.

Place special wood cabinets with compartments of suitable size in conspicuous and easily accessable parts of the cyanide plant.

Have the emergency orders for the handling of patients affixed to the inside of cabinet door.

Properly label each bottle, also see that its contents are kept in a fresh and pure condition.

Compartment 1 should contain:

One large bottle of distilled water:

One large metal spoon:

One pint metal cup for mixing the different antidotes.

Compartment 2 should contain for external use:

One bottle of ammonia:

One bottle of ether;

One bottle of acetic acid:

One bottle of chloride of lime (bleaching powder):

One piece of flannel cloth for administering the chlorine gas.

Compartment 3 should contain for internal use:

One bottle ammonia water solution (diluted);

One bottle of cobalt nitrate solution (diluted);

One bottle of ferrous sulphate (fresh);

One bottle of sodium carbonate (fresh).

Compartment 4 should contain for internal use:

One bottle (sealed) containing 30 C. C. of caustic potash;

One bottle (sealed) containing 30 C. C. of 38% solution of ferrous sulphate;

One package (sealed) containing oxide of magnesium (light), to be used as follows:

1st.—Quickly empty the contents of the two sealed bottles and packages into the metal cup, and stir thoroughly with the metal spoon.

2nd.—If patient conscious make him swallow the mixture at once and lie down for a few minutes.

3rd.—If patient unconscious place him on his back and pour the mixture down his throat in small quantities, if necessary pinching his nose in order to make him swallow.

4th.—Incite vomiting by one of the previous methods suggested, and it is advisable to keep a soft rubber stomach tube having a funnel and exhaust bulb in one of the cabinets, if the services of a physician cannot be had promptly.

N.B.—It is advisable to have a physician prepare the several solutions required and prescribe the dose of each to be taken by a patient in an emergency, and to further assist in preventing mistakes, bottles of blue glass can be used as containers for the acids, and bottles of white glass for the alkalies, if it is desired."—W. H. Kritzer, *Salt Lake Mining Review.* —*Mexican Mining Journal*, March, 1913, p. 144. (A. R.)

THE METRIC CARAT AND THE WEIGHING OF DIAMONDS.—"As a part of the universal tendency to unify weights and measures, the recent movement for the establishing of a standard carat in the weighing of diamonds is an interesting example. The metric carat has been adopted by various European nations, although it encounters the usual inertia of English-speaking countries with respect to the metric system of measures. The new carat is 200 mgm. and Spain has the distinction of having been the first to adopt what is now a statutory measure in ten or twelve countries. The *American Jeweler* gives the following account of the origin of the carat:

The Greek weight 'keration,' and also the Roman 'silique' were equivalent to 1-144 oz., or 3⅓ gr. of our present weights; that is only slightly more than the present value of the carat (about 3⅙ gr.).

The carat weight had its origin in the use of certain hard leguminous seeds, fifty seeds of Ceratonia Siliqua taken at random and weighed together, gave an average weight of 197 mgm. per seed, thus approximating very closely to the present value of the carat weight, which is 205·304 mgm.

The above mentioned Ceratonia Siliqua is the carob or locust tree, the fruit of which is the well-known locust-bean, or St. John's bread. The Greek name 'keration,' refers to the horn-like shape of the fruit pods; and carat is an obsolete English name for the seed. The seeds are remarkably constant in weight.

An International carat of 205 mgm. was proposed in 1871 by the Syndical Chamber of Jewellers, etc., in Paris, and the Association of Diamond Workers in Amsterdam adopted on October 17, 1890, a carat weight of 205·128 mgm.

The German N.A.G., in 1905 or 1906, petitioned the German Reichstag, in order to get official permission for the use of the carat weight for diamonds and precious stones, but did not succeed.

The first country to introduce the new carat of 200 mgm. was Spain: Italy shortly afterwards followed, then Bulgaria and Denmark (1910), Norway (1910), Japan (1909), Portugal (1911), Roumania (1911), Switzerland (1909), Sweden (1911).

France adopted the new carat by the Act of June 22, 1909, and it was destined to be used from January, 1911. At the request, however, of the jewellers and dealers in diamonds, and because the manufacturers of weights could not supply the trade quickly enough with the new weights, the reform was postponed until January, 1912, since when the metric carat is the only legal weight for precious stones in France.

Belgium and Holland, two countries equally important for the diamond trade, have not yet officially adopted the reform, but the suggested acts or laws are already being discussed by the authorities, and ere long they will be brought before the legislative chambers, and there is no doubt that they will be adopted, as the trade of both countries have declared themselves in favour of the reform.

Germany has been the last country to introduce the new weight for precious stones, a new act for weights and measures having been operative since April 1, 1912, by which all old weights (including the old carat) were strictly abolished, the metric system enforced, and the denomination of carat allowed for the unit of 200 mgm.

The movement for the general adoption of the metric carat receives little support in England. It may be added, that the English carat is not a legal unit of weight under the weights and measures act of 1878, but the metric would be covered by the weights and measures (metric system) act of 1897.

The National Association of Goldsmiths in annual convention at Scarborough, has unanimously resolved that the metric carat should be the standard weight.

The difference between the old and the new units is only slight, the metric carat being approximately 2½% less than the present carat."—*Mining Science.* April, 1913, p. 212. (A. R.)

MANTLES FOR INCANDESCENT GAS LIGHTING.—"*Knitting.*—The different kinds of web produced on the round knitting machine have the drawback of being less pliable than the ordinary web. Cotton mantles appreciably decrease in brilliancy after use, those of ramie are very constant, but the artificial silk mantles, now coming into use, combine the high and well-maintained illuminating power of ramie fibre with a great elasticity of the ash skeleton. Cuprammonium cellulose, chardonnet, and viscose silks are all used for the manufacture of mantles.

Washing.—Both cotton and ramie mantles require washing before impregnating, to remove the fat and mineral matter: this was done formerly by using a 3% to 5% solution of soda, followed by a 1% to 3% solution of hydrochloric acid. Recently, however, the process has been simplified; the mantles are soaked over night in a 2% solution of nitric acid and the next morning they are centrifuged, treated with a week solution of ammonia, and then washed, first with condensed and afterward with pure distilled water.

Drying.—The mantles are dried in a current of hot air, and the imperfect ones sorted out. From this point scrupulous cleanliness in handling the mantles is imperative.

Impregnation.—The mantles are impregnated by being soaked in a 25% to 33% aqueous solution of thorium and cerium nitrates in the proportion of 99:1, to which very small amounts of beryllium and magnesium nitrates are added to harden and strengthen the ash skeleton. After soaking in this liquid for a time—from one to two minutes in the case of cotton and ramie mantles, and five hours for artificial silk mantles—they are passed through a wringing machine. The quantity of salts left in the mantles can be varied by adjusting the rollers of the wringer. The mantles are next dried at a temperature not exceeding 30° C.; the cotton and ramie mantles, cut in short lengths before impregnation, are stretched over glass cylinders, and the artificial silk mantles, impregnated in lengths, are run over wooden rollers. The upright mantles are then fitted with a tulle head sewn on, or with a patent head, and this is fixed by treatment with a 50% aqueous solution of aluminium and magnesium nitrates. Then the asbestos loop by which the mantles are suspended is sewn on. The inverted mantles are drawn together with impregnated cotton, to form the 'spider,' and are then fastened on to the supporting ring.

Burning off.—If a trade-mark is desired on the mantle, this is stamped on with a solution of didymium nitrate, dissolved in a mixture of alcohol, distilled water, and glycerine, to which a little methyl Blue has been added Then the mantle is shaped on a wooden shaper and is burned off. This is done with a Bunsen flame, beginning at the top of the mantle, and the mantle is hardened by increasing the pressure of the gas for a period of 1½ to 15 minutes. The mantles now consist of a skeleton of ash and are ready for use, but they will not bear transportation. Therefore they are dipped into a collodionizing solution, consisting of 100 parts of 4% collodion solution, 40 parts ether, 6·5 parts camphor, and 3·5 parts castor oil. After rapid drying at a temperature of from 50° C. to 60° C., the mantles are trimmed to the proper length, sorted out into different qualities, and the imperfect ones rejected. Then follows a list of various defects to which mantles are liable, and the causes to which these defects may be attributed."—C. R. BOHM (*J. Gas Lighting*, cxxi, 33, 100, 181), *Journal of the Franklin Institute*, vol. clxxv, May, 1913, No. 5, p. 553-554. (J. G.)

Reviews.

DE RE METALLICA. BY GEORGIUS AGRICOLA, translated from the First Latin Edition of 1556 by Herbert Clark Hoover and Lou Henry Hoover, Vellum; folio; xxxi+640 pp. *Mining Magazine*, London, 1913. Price £1 1s. net.

"The appearance of a translation into English of Agricola's famous treatise on mining is an event of great interest to the profession. Most of us are familiar with the big folio volume with its hundreds of quaint illustrations, though but a few of us, I fancy, have had the courage to attempt to read many pages of the Latin text. This English edition is, however, much more than a translation, as Mr. Hoover has added introductory chapters, and an appendix, these containing among other things a sketch of the life of Agricola and an essay on his intellectual attainments and position in science, and a bibliography and review of his writings, as well as of the writings of ancient and medieval authors on mineralogy, geology, mining and metallurgy. In the text are numerous historical foot notes on alchemy; on the ancient mines of Laurium; on medieval views regarding ore deposits; on the early development of mining law; on medieval mineralogy with names of the known minerals in Latin, German and English; on the mechanical appliances used in mining; on assaying; on the use of the touchstone; on crushing and concentrating of ores; on liquation, amalgamation, parting, and other metallurgical processes; and on the early methods for the manufacture of chemicals. These notes add greatly to the value of the book and indicate most indefatigable research on the part of the author; by themselves they would constitute a fair sized volume.

Georgius Agricola, whose German name was George Bauer, was born in Saxony, March 24th, 1494, and died Nov. 21st, 1555. He was thus a contemporary of Luther 1483-1546, of Erasmus 1467-1536, of Melanchthon 1497-1560, of W. Meurer 1513-1585 and of G. Fabricus 1516-1571. Agricola was one of that company of scholars who contributed to the revival of learning and the re-awakening of the arts and sciences, known as the Renaissance, which begun in Italy and later spread through Germany and Europe.

Agricola was a scholar of high attainments and wide learning. He received the degree of Bachelor of Arts at the University of Leipzig in 1517, where he was afterwards a lecturer between 1522 and 1524. He also studied at the Universities of Bologna and Venice and probably at Padua for nearly three years between 1524 and 1527. Returning to Germany he was made town physican of Joachimsthal in 1527, where he spent about three years and devoted all his spare time to visiting mines and smelters; in reading ancient classics and manuscripts bearing upon mining, and in association with the most learned and skilful among the mining folk. In 1530 he published his first book on mining, *Bermannus*, in the form of a dialogue on mineralogy, mining terms and mining law. After devoting two or three years to travel and study among the mines, he began his great treatise, *De Re Metallica*, which was not published until twenty-five years later. About 1536 he became city physician of Chemnitz and here he lived until his death in 1555. He was a shareholder in 'God's Gift' mine at Alberthau, which was discovered in 1530, and obtained dividends therefrom for many years. He was a recognised authority on mining matters and was frequently called upon for advice. In 1546 he was made burgomaster of Chemnitz and in that capacity attended the Diets of Freiburg, Lepzig, Dresden, etc. He frequently served the Duke of Saxony as an adviser in matters of State and was sent on diplomatic missions to other rulers. In these and similar capacities he continued in public service with short intermissions until the time of his death.

Agricola was a deep thinking religious man, and while liberal in such matters he remained a staunch Catholic and steadily refused to accept the Lutheran Reformation. The character and ability of the man are evident from the fact that he was on terms of intimate and cordial friendship with the great leaders of the Protestant Reformation and commanded the respect and confidence of the Protestant rulers of a State 'seething with militant Protestantism.'

In addition to the two books mentioned Agricola was the author of many treatises on natural science, which are reviewed by Mr. Hoover in the appendix. Among these are *De Natura Fossilium* which is the first attempt made at systematic mineralogy on the basis of solubility, fusibility, odor, taste, etc. Dana, among some four score minerals described by Agricola and his predecessors, credit a score to Agricola himself.

Another book is *De Ortu et Causis Subterraneorum* consisting of five chapters on geological phenomena. He was probably the first to recognise that the topographic features of mountainous regions are primarily due to the erosive action of wind and water.

Agricola projected a complete series of works covering the whole range of subjects relating to minerals, mineralogy, geology, mining, metallurgy, history of metals and their uses, mining laws, etc. While many of these books were completed in manuscript they failed of publication mainly through the lack of interest on the part of his heirs.

Mr. Hoover says in his preface 'We do not present *De Re Metallica* as a work of practical value' . . . Nevertheless we venture to quote a few paragraphs from Book II in which Agricola gives advice as to mine management and upon mines as a business investment. As will be seen Agricola was not altogether ignorant of what we now style 'efficiency engineering' and found it as necessary to warn investors against 'wild-cat mines' as we do to-day.

'It is not sufficient for any one to store away in his mind knowledge of all the departments of the metallic art without regard to the outlays that must be made upon mines or without experience in undertaking and carrying through the practical work.' . . . 'When any one in an endeavour to increase his fortune meets the expenditure of a mine alone it is of great importance that he should attend to his works and personally superintend everything that he has ordered to be done. For this reason he should either have his dwelling at the mine, where he may always be in sight of the workmen, and always take care that none neglect their duties, or else he should live in the neighbourhood so that he may frequently inspect his mining works. Then he may send word by a messenger to the workmen that he is coming more frequently than he really intends to come, and so either by his arrival or by the intimation of it, he so frightens the workmen that none of them perform their duties otherwise than diligently. When he inspects the mine he should praise the diligent workmen and occasionally give them rewards that they and the others may become more zealous in their duties; on the other hand, he should rebuke the idle and discharge some of them from the mines and substitute industrious men in their places. Indeed the owner should frequently remain for days and nights in the mine, which in truth is no habitation for the idle and luxurious; it is important that the owner who is diligent in increasing his wealth, should frequently himself descend into the mine, and devote some time to the study of the nature of the veins and stringers, and should observe and consider all the methods of working, both inside and outside the mine. Nor is this all he ought to do, for sometimes he should undertake actual labour, not thereby demeaning himself, but in order to encourage his workmen by his own diligence, and to teach them their art, for that mine is well conducted in which not only the foreman but also the owner himself gives instruction as to what ought to be done.' . . .

'When several share together the expenditure on a mine it is convenient and useful to elect from among their own number a mine captain and also a foreman. For, since men often look after their own interests but neglect those of others, they cannot in this case take care of their own without at the same time looking after the interests of the others, neither can they neglect the interests of the others without neglecting their own. But if no man amongst them be willing or able to undertake and sustain the burden of these offices, it will be to their common interest to place them in the hands of the most diligent men.' . . .

'When a man owns mines but is ignorant of the art of mining then it is advisable that he should share in common with others the expense, not of one only, but of several mines. When one man alone meets the expenses for a long time of the whole mine, if good fortune bestows on him a vein abundant in metals, or in other products, he becomes very wealthy; if on the contrary, the mine is poor and barren, in time he will lose everything which he has expended on it. But the man who, in common with others, has laid out his money on several mines in a region renowned for its wealth of metals, but rarely spends it in vain, for fortune usually responds to his hopes in part. For when out of twelve veins in which he has a joint interest one yields an abundance of metals, it not only gives back to the owner the money he has spent, but also gives a profit besides; certainly there will be for him rich and profitable mining, if of the whole number, three, or four, or more veins should yield metal.' . . .

'Some owners prefer to buy shares in mines abounding in metals rather than to be troubled themselves to search for the veins; these men employ an easier and less uncertain method of increasing their property. Although their hopes in the shares of one or another mine may be frustrated, the buyers of shares should not abandon the rest of the mines, for all the money expended will be recovered with interest from some other mines.' . . . 'Moreover a prudent owner, before he buys shares, ought to go to the mine and carefully examine the nature of the vein, for it is very important that he should be on his guard lest fraudulent sellers of shares should deceive him.' . . . 'Nor ought miners to be altogether distrustful of fortune, as we see some are, who, as soon as the shares of any mine begin to go up in value sell them, on which account they seldom obtain even moderate wealth.'

Translation of Agricola's book has been attempted by others but we believe this to be the first time that the work has been done by anyone having the necessary scholarship combined with the practical knowledge of the art of mining required for the purpose. Mr. Hoover in his preface outlines some of the many difficulties that had to be mastered, but for these we must refer the reader to the book itself. The work has been a labour of love which has occupied the spare time of a busy professional man for a period of about five years. The printing and distribution of the book is a princely gift to the profession, as the translators have expended on the preparation and publication several times the nominal price at which the edition will be sold. We owe to Mr. Hoover and to his talented wife our hearty thanks for their unselfish devotion to the task which they have done so well and we feel sure that copies of Hoover's Agricola will have an honoured place in the libraries of many generations of mining engineers in the centuries to come."—*School of Mines Quarterly, Columbia University*, Vol. xxxiv., No. 3, April 1913, pp. 280-284. (J. G.)

A TEXT-BOOK OF MINING GEOLOGY. By JAMES PARK. Third Ed., revised and enlarged. Cloth, octavo, 320 pages with many illustrations. London: Charles Griffin & Co. Price 6s. net.

"This is one of the most useful of technical books. Published first in 1906, then in 1907, and last in 1911, it has been kept up-to-date with a painstaking care characteristic of the author in all his work. An introductory chapter deals with the elements of geology. The second is devoted to a classification of ore deposits, with a great variety of examples, with simple illustrations. Veins are given a chapter to themselves. Under this heading are discussed the origin, structure, and mineralization of the dominant types of ore deposit. Secondary enrichment and persistence of ore are explained simply. A large quantity of pertinent information is condensed within a few pages, the selection of facts being made with admirable good sense. In the next chapter the student is given a clean analysis of faults and faulting, illustrated by a select bibliography, of the greatest convenience to an inquiring reader. In Chapter VII. we find the outlines of economic mineralogy, informing the student concerning the principal ores of the metals, with sufficient reference to their geographic distribution. The commercial non-metallic minerals, such as oil, mica, gypsum, and graphite are included. The last two chapters deal with the sampling and valuation of mines, giving the simple calculations required for ordinary cases of dislocation. Chapters V. and VI. are devoted to theories of ore deposition. It is a difficult subject for a small text-book, but Mr. Park has used discretion, so that the reader is only given the essentials." — T. A. RICKARD, *Mining Magazine*, May 1913, p. 381. (A. R.)

Abstracts of Patent Applications.

(C.) 333/12. Louis David Mills. Improvements in precipitating apparatus. 26.6.12.

This application describes a combination precipitation apparatus comprising a liquid receptacle or a tank, a submerged filter leaf, means for circulating and distributing the mixture of solution and precipitant and means for drawing the solution through the cake of precipitant formed uniformly on the filter leaf.

(C.) 347/12. Guy Aubrey Chalkley. Improvements in supporting rock drills. 2.7.12.

This is a device for keeping in alignment the drill used in a pneumatic hammer drill.

It consists of a supporting bar carried on the arm of an ordinary rock drill column, and having at its outer end a conical projection to fit into the collar of the hole to be drilled, and having an opening through which the drill steel passes and in which it works.

To this supporting bar is fitted a tubular bar, arranged to carry the hammer drill on two projecting brackets, and capable of sliding along the supporting bar under pressure from an air-feed cylinder, operating at its outer end, and secured to the supporting bar.

(C.) 396/12. Theodore Reunert. Improvements in mortar boxes. 25.7.12.

This application is for a mortar box constructed of a metal—preferably cast—base. The sides and ends are pressed to shapes the same as the pressed steel plates for the sides and ends, and ordinary cast mortar box. The sides are attached to the base in different ways, *i.e.*, by bolting, and by strips being cast into the base and then riveted to the sides.

(C.) 408/12. William Daly. Improvements in stamp stem guides. 26.7.12.

This application relates to a stamp guide consisting of a hollow open-ended cylinder having circular ribs on its extended periphery—these ribs providing grooves and U bolts taking into these grooves so that the guide may be readily turned.

(C.) 410/12. William Daly. Improvements in stamp tappets. 26.7.12.

This application relates to a stamp tappet in which the ordinary form of gib is replaced by a gib having a semi-circular recess to grip the stem, and the outside egdes inclined so that on driving up the cotter the gib is forced in two directions. There is a further claim for a drift holder to be passed through a hole in the tappet.

(C.) 437/12. Samuel Thomas Tregaskis. Improvements in furnaces for heating rock drills. 14.8.12.

This application relates to a drill heating furnace consisting practically of a coal gas producer and a flue into which the gas from the producer is blown and burnt and in which the drills are heated.

(C.) 438/12. Oliver Harrison (1), John Robert Pratt (2). Improvement in means for controlling the underflow from classifiers and the like. 14.8.12.

This application relates to means for automatically controlling the underflow from classifiers in order to obtain a product of required degree of moisture.

The device shown consists of a system of levers and compensating weights held in equilibrium by the weight of pulp outflowing and which on being disturbed by a sudden rush of thin pulp causes the discharge orifice to be closed until readjusted by a water filled tippler box.

(C.) 211/13. Alfred James. Improvement in apparatus for agitating or collecting ore pulp. 10.5.13.

This application relates to agitation of pulp and describes an apparatus for raising and lowering the agitator arms, etc. by means of a clutch on the driving shaft which operates gearing for that purpose.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 265/13. Thomas Holmes (1), Arthur John Walton (2). Improvements in rock drilling and like percussive tools with detachable bits. 9.6.13.

(P.) 266/13. John Matthew. Improvements in electrical water heater. 10.6.13.

(P.) 267/13. Richard Casson Power. An improved detachable bit. 10.6.13.

(P.) 268/13. Bruce W. Taylor. Improvements in filtering apparatus. 10.6.13.

(C.) 269/13. Wilhelm Goert Boonzaier. Scrapers, reamers, and burnishers for smokers, pipes, tubes, cylinders and the like. 13.6.13.

(C.) 270/13. Charles Washington Merrill. Process for precipitating and recovering valuable metal bearing materials from solutions and precipitants for use in connection therewith. 13.6.13.

(C.) 273/13. Samuel Campbell Irving. Improvements in or relating to roofing sheets. 13.6.13.

(P.) 274/13. John Oliver. Improvements in tube mill liners. 14.6.13.

(P.) 276/13. Arthur Miles. Improvements in rollers for haulage ropes and the like. 14.6.13.

(P.) 277/13. Walter Bolter. Rock drill head coupling. 16.6.13.

(P.) 278/13. Louis Bulbring. Improvements in the manufacture of certain tools and implements, namely forged shovels, spades, forks and the like. 17.6.13.

(P.) 279/13. Reuben Jones. Improvements in and relating to rock-drills and detachable bits or cutters therefore. 18/6/13

(P.) 281/13. Henry Marius Schade van Westrum. Improvements in methods of making and maintaining roads or like surfaces. 19 6.13.

(P.) 282/13. Sydney Joseph Crowther Wilkinson. Improvements in detachable or loose bits for drilling purposes. 19.6.13

(C.) 284/13. Nicolas Henrie Marie Dekker. Improvements in or relating to electrolytes for use in electro-metallurgy. 19 6.13.

(P.) 285/13. Robert Robson. Regulating feed for sand cones. 20.6.13.

(C.) 286/13. Oliver Morgan Babcock. Improvements in or relating to boxes and machines for making them. 26.8.13.

(P.) 287/13. John Shanks. An improved apparatus or appliance for listing drain pipes, water pipes, gas pipes and the like. 20.6.13.

(C.) 288/13. Albert August Jahnke (1), National Wireless Telephone and Telegraph Company (2). Apparatus for the production of high frequency oscillating currents. 20.6.13.

(P.) 289/13. Davis Ord-Hume, jr. Improvements in gates. 21.6.13.

(P.) 290/13. James Matley (1). Constantine Russell (2). Improvements in means for completing an electric circuit when a predetermined temperature is reached, applicable for actuating fire alarms and for other purposes. 21.6 13.

(P.) 291/13. Charles Weir. A new and improved method and means for spraying cattle and the like. 21.6 13.

(P.) 292/13. Frank Simon (1), John William Scott (2). Improvements in mining columns. 23.6.13.

(P.) 293/13. Frank Simon (1), John William Scott (2). Improvements in feeding means of percussive rock drills, coal cutters and the like. 23.6.13.

(P.) 294/13. Walter Bolter. Converted drill sharpener. 24.6.13.

(P.) 295/13. George Heathcote Rotherham. A renewable shoe shank for stamp mills and the like. 25.6.13.

(P.) 297/13. Harry Brown. Improvements in ore separating cones and the like. 25.6.13.

(P.) 298/13. Alfred Whitby (1), Josiah Whitby (2). Improved pole for mine packs. 26.6.13.

(P.) 299/13. Arthur John Phillip. Improvements in or relating to shells or cradles of rock-drilling machines. 26.6.13.

(P.) 300/13. Wm. Lippiatt. Improvements in catches or fastening devices applicable for sliding doors and the like. 26.6.13.

(P.) 301/13. Guiseppe Antonio (1), Pietro Provay (2). Improvements in tipplers or means for emptying railway trucks and the like. 26.6.13.

(C.) 302/13. John Godfrey Parry Thomas (1), The Thomas Transmission, Ltd. (2). Improvements in and connected with electric power transmission. 26.6.13.

(C.) 303/13. John Francis Staley (1), Cyril Frank Jackson (2). Improvements in safety mechanism for the winding gear of coal pits and the like. 26 6.13.

(P.) 304/13. George Mellis. Improvements in or relating to nut-locks. 26.6.13.

(C.) 305/13. Harry Palent. An improvement in the manufacture of engine packings from leather. 27.6.13.

(P.) 308/13. Spiro Matelas. How to manufacture rubber from indigenous roots and the like. 28 6.13.

(C.) 310/13. Wm. Fennell. Improvements in or in connection with tube mills. 30.6.13.

(C.) 311/13. Thomas Unsworth. An improved pneumatic wheel. 2.7.13.

(P.) 312/13. Carl Dresdner. The advertising clock. 2.7.13.

(P.) 313/13. Charles Wm. Twiss. Improved process for cleaning diamondiferous soils and gravels. 2.7.13.

(P.) 314/13. Albert Carmel. An improved specific remedy for the cure of piles. 2.7.13.

(P.) 315/13. Albert Carmel. An improved specific remedy for hysteria, convulsions, worms, and insomnia. 3.7.13.

(C.) 316/13. Samuel Redfern. Improvements in or relating to packing for steam engines and the like. 3.7.13.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

ANCARINI, H., *l/o* Paris; Mennacy, Seine-et-Oise, France.

BELL, H. C. F., *l/o* Germiston; Zaaiplaats T. M. Co., Sterkwater, Transvaal.

FILET, G., *l/o* Sumatra; Mijnbouw Maatchappij, "Salida," *via* Padang, D. E. Indies.

GILCHRIST, D., *l/o* Randfontein; Brakpan Mines, Ltd., P. O. Box 3, Brakpan.

GULLACHSEN, B. C., *l/o* Randfontein; Manor House Hotel, Jeppes.

HORSFIELD, J. E., *l/o* Cam and Motor Mine; Golden Kopjes Mine, Lomagundi, Rhodesia.

HOWARD, T. C. L., *l/o* Knights; Main Club, East Rand Propriety Mines, Ltd., East Rand.

JOHNS, J. HARRY, *to* "Thorsden," Guildford Road, Woking, Surrey.

KIERNANDER, M. C., *l/o* Cleveland; Bantjes Cons. Mines, Ltd., P. O. Box 2, Florida.

KRAUSE, H. L., *l/o* Benoni; New Rietfontein Estates, P. O. Box 15, Reitfontein.

LAMONT, K. B., *l/o* Langlaagte; Van Ryn Deep, Ltd., P. O. Kleinfontein.

LESLIE, H. M., *to* 10, St. Andrew's Drive, Pollokshields, Glasgow.

LOVELL, W. J., *to* P. O. Box 30, East Rand.

MILLER, J., *l/o* Long Tunnel Mine; "Shrewsbury," Esplanade, Middle Brighton, Victoria, Australia.

NORMAND, J. N., *l/o* Gwelo; Queens Mine, Bulawayo, Rhodesia.

ORDMAN, D., *to* 23, Upper Meyer Street, New Doornfontein.

POOLER, F. J., *l/o* Johannesburg; 123, Windmill Lane, Smethwick, Staffs, England.

ROBERTSON, G. A., *to* May Cons. G. M. Co., P. O. Box 193, Germiston.

STRANGE, E H., *l/o* Colorado; P. O. Box 732, Kellogg, Idaho, U.S.A.

VEASEY, J. H., *l/o* Rhodesia; P. O. Germiston.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. XIV. AUGUST, 1913. No. 2.

Proceedings
AT
Ordinary General Meeting, August 16th, 1913.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, August 16th, 1913, Mr. Alex. Richardson (President) in the chair. There were also present:

39 Members: Prof. G. H. Stanley, Messrs. J. E. Thomas, H. A. White, J. Littlejohn, T. Donaldson, S. Salkinson, C. Toombs, F. Wartenweiler, F. W. Watson, John Watson, R. G. Bevington, A. F. Crosse, W. R. Dowling, A. McA. Johnston, J. R. Williams, John Yates (Members of Council), A. S. Allam, E. H. Croghan, J. M. Dixon, R. Dures, B. J. Hastings, J. M. L. Henry, G. A. Lawson, W. A. Ledingham, H. H. Morrell, P. T. Morrisby, J. M. Neill, S. Newton, G. Nicol, F. D. Phillips, J. C. Phillips, C. D. Richardson, D. F. Scott, A. L. Spoor, J. A. Taylor, A. Thomas, W. E. Thorpe, J. H. Veasey, and H. Wood.

13 Associates and Students: Messrs. N. S. Arnot, E. G. Baskett, G. J. V. Clarence, J. Cronin, O. A. Gerber, J. Ingram, R. W. Irwin, L. T. Leyson, S. E. Willows-Munro, R. Ramsay, P. Scatterty, R. R. Vail and W. Waters.

13 Visitors, and Fred. Rowland (Secretary).

MINUTES.

The Minutes of the Annual General meeting were confirmed.

NEW MEMBERS.

Prof. J. Yates and Mr. J. R. Williams were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected, as follows:—

CUMMING, JOSEPH, Simmer Deep, Ltd., P. O. Box 178, Germiston. Amalgamator.

KELLY, JOHN VINCENT, El Favor Mining Co., Hostotipaquillo, Jalisco, Mexico. Mining Engineer.

The Secretary announced that since the last meeting of the Society the following Associates and Students had been admitted by the Council:

As Associates—

AYMARD, MALCOLM, 27 Primrose Terrace, Berea, Johannesburg. Metallurgist. *(Transfer from Students Roll.)*

CREED, CECIL THOMAS, New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni. Assistant Assayer.

WHITE, STANLEY BROWN, Bullfinch Proprietary G. M. Co., Bullfinch, W. Australia. Assayer.

As Students—

KINAHAN, BRIAN ARTHUR DOUGLAS, Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Post Graduate Student.

LONG, FRANK DOUGLAS, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

GENERAL BUSINESS.

INTERNATIONAL ENGINEERING CONGRESS.

The Secretary: I have to bring to the notice of members that a most cordial invitation has been received from a number of Engineering Societies in America to attend and participate in the proceedings of an International Engineering Congress to be held in San Francisco in Sept., 1915. The following details are published for information of members:—

"In connection with the Panama-Pacific International Exposition which will be held in San Francisco in 1915, there will be an International Engineering Congress, in which engineers throughout the world will be invited to participate.

The congress is to be conducted under the auspices of the following five National Engineering Societies: The American Society of Civil Engineers, American Institute of Mining Engineers, The American Society of Mechanical Engineers, American Institute of Electrical Engineers, and The Society of Naval Architects and Marine Engineers.

These societies, acting in co-operation, have appointed a permanent Committee of Management, consisting of the Presidents and Secretaries of each of these Societies, and eighteen members resident in San Francisco.

The Committee has effected a permanent organisation, with Prof. Wm. F. Durand as Chairman, and W. A. Cattell as Secretary-Treasurer, and has established executive offices in the Foxcroft Building, 68 Post Street, San Francisco.

The ten members of the Committee, consisting of the Presidents and Secretaries of the five national societies will constitute a Committee on participation, through whom all invitations to participate in the Congress will be issued to governments, engineering societies, and individuals.

The Honorary Officers of the Congress will consist of a President and a number of Vice-Presidents selected from among the most distinguished engineers of this and foreign countries.

The papers presented at the Congress will naturally be divided into groups or sections. During the Congress each section will hold independent sessions, which will be presided over by a chairman eminent in the branches of engineering covered by his section.

The scope of the Congress has not as yet been definitely determined, but it is hoped to make it widely representative of the best engineering practice throughout the world, and it is intended that the papers, discussions and proceedings shall constitute an adequate review of the progress made during the past decade and an authoritative presentation of the latest developments and most approved practices in the various branches of engineering work.

The papers, which will be collected and published by the Congress, should form an invaluable engineering library, and it is intended that this publication shall be in such form and at such cost as to become available to the greatest possible number.

The various committees are now actively at work, and it is hoped that further and more definite announcements as to the membership fees, schedules of papers, etc., can be made in the very near future."

IRIDIUM PRIZE PAPERS.

Mr. J. R. Williams (*Past-President*): I would like to ask whether the two Iridium prize papers appearing in the last *Journal* are open for discussion. My reason for asking the question is that I know that in connection with papers of this nature full benefit from them can only be derived by a thorough discussion. I think I am expressing the wish of a number of members when I say that, unless there are good reasons to the contrary, the papers should be open for discussion at future meetings.

The President: In reply to Mr. Williams, the Council went fully into the matter of discussing these Iridium papers, and the conclusion we came to was, that it was scarcely fair, under the circumstances, to throw them open to discussion. It is quite open to any member, however, to bring forward a contribution on the subject as an original paper of his own, and thus re-open the whole question.

I have much pleasure in presenting the prizes to the joint winners in this competition. I am sorry to say that Mr. Gray is unable to attend, but Mr. Toombs is here, and I am sure he will, while receiving his own prize, be pleased to represent Mr. Gray on this occasion.

Mr. C. Toombs (*Member of Council*) briefly expressed his thanks.

MR. PERCY WHEELER.

The Secretary: Enquiries have been received through one of the British Legations on the Continent, for a Mr. Percy Wheeler, a mining engineer, who is reported to have been on the Witwatersrand for about ten years. Private enquiries have proved fruitless and the name cannot be found in any directory. Should any member be cognisant of his whereabouts I shall be glad if he will communicate with me.

JOHANNESBURG TRADES SCHOOL.

A letter has been received from the Secretary of the Johannesburg Trades School, drawing attention to a pamphlet showing the work done by that school. I shall be pleased to send a copy to any member interested.

SOUTH AFRICAN RED CROSS SOCIETY.

I also wish to draw the attention of members to the fact that the South African Red Cross Society is now prepared to give lectures, hold examinations, and give certificates to successful candidates in ambulance work. The Society, being newly organised, is in need of funds; and if any one present would like to become a member of the Red Cross Society the necessary forms can be obtained on application. The subscription is 10s. 6d. per annum. The syllabus and circulars giving the aims and objects of the organisation will be sent to anyone requiring them.

The President: I may mention that the Red Cross Society is taking over the organisation of the competitions for the Society's Ambulance Shields, which, as may be remembered, were so very successful last year. It is expected that the competitions will be held about the end of October of this year; and, I think, under the auspices of such an expert body as the Red Cross

Society, they will probably cover a wider field; and I have little doubt that, as the mining public becomes more familiar with these very useful competitions, their success will continue to increase.

Mr. A. F. Crosse (*Past-President*): I do not know whether it is the intention of the Council to hold an Exhibition next year, but, if so, it is desirable that preparations should be made many months beforehand, in order to permit of oversea exhibitors being well represented.

The President: I have much pleasure in informing Mr. Crosse that an Exhibition Committee has been appointed, and that it intends, in the course of a week or so, to begin work in connection with the organisation of the Exhibition.

The President: I will ask Prof. Stanley to take the Chair while I read my Inaugural Address.

INAUGURAL ADDRESS.

By Alex. Richardson, M.Inst.M.M. (President).

When you elected me President of this Society I was at once deeply sensible of the honour conferred upon me; and I soon became conscious that the occupation of the chair was coincident with the performance of certain duties, the most pressing of which was the presentation of an inaugural address. There being no restrictions as to the subject-matter and form of such an introductory lecture, other than the elastic ones dictated by a sense of propriety, I experienced some difficulty in deciding upon a theme. Many subjects presented themselves to my mind; but on consideration I found that most of them were either too technical or controversial to be satisfactorily dealt with in a paper not open to free discussion, or of too general a nature to allow me to do justice to them without the risk of wearying you. I therefore decided to utilize past experiences of my own, in this and other districts of South Africa, and to base thereon a brief review of the training and work of the mine surveyor.

Mine surveying is to-day recognized throughout the Union as a distinct profession. This recognition is partly due to the increased responsibility attaching to the work through the magnitude and systematic organisation of modern mining operations, but mainly owing to its being necessary for mine plans to be signed by a certificated mine surveyor before the Government will officially accept them. Now every profession has its own characteristic scope, and calls for special skill in certain directions; and it is therefore desirable that anyone who aims at becoming a surveyor should devote particular attention to those lines of study which have a direct bearing on his work. With the object of assisting students of surveying, or those just beginning as assistants on the mines, to form an idea of the knowledge that will be required of them, I have drawn up a concise syllabus embracing these subjects which experience has shown me require closer attention than others.

Mathematics:—*Arithmetic.*—Addition, subtraction, multiplication, and division; their tabulation, short cuts, abridged methods, and checks. Greatest common measure. Least common multiple. Fractions. Decimals: rough tests for checking the position of the decimal point. Alligation. Square and cube measure. Ratio, proportion, and rule of three. Application of the terms average and per cent. Square root. Calculation of numerical values from simple formulæ. Weights and measures.

Algebra.—Addition, subtraction, multiplication, and division. Brackets. Fractions. Factorisations. Indices. Transformation and simplication of formulæ. Simple, simultaneous, and quadratic equations.

Geometry and Mensuration.—The theorems, problems, and corollaries of Euclid, books 1 to 4, 6 and 11, but not with strict adherence to the Euclidian order and method. Scales. Drawing to scale. Problems relating to lines, circles, and plane figures. Location by rectangular coordinates. Projection of points and lines. Planes. Projection of plane figures on inclined planes. Line and plane. Areas of plane figures with rectilinear and curvilinear boundaries. Areas of surface of solids. Volumes of regular solids. Volumes of excavations, embankments, and irregular solids. Density and relative density. Capacities and weights of structures. Use of planimeter. Measurement of quantities from plans. Measuring quantities and extending in schedule form.

Trigonometry.—Measurement of angles. Trigonometrical ratios. Application of algebraical signs. Trigonometrical ratios of two angles. Formulæ for the division of angles. Relation between the sides of a triangle and the trigonometrical functions of the angles. Transformation and reduction of trigonometrical expressions. Logarithms. Mathematical tables. Properties of triangles. Solution of triangles. Measurement of heights and distances.

Geology and Mineralogy.—The external and internal forces effecting changes in the earth's crust, and the processes of disintegration, transportation, and deposition. Laws of stratification.

Subsequent features: jointing, inclination, folding, and faulting. Sedimentary, metamorphic and igneous rocks: their formation, characters, and modes of occurrence. The simpler forms of ore deposits. Common rock-forming minerals. Methods of determining the strike and dip of strata, faults, dikes, and veins. Dip and strike problems. Faults and fault problems. Methods of physical and chemical examination by which minerals and rocks can best be identified in the field and in the laboratory. Geological sketch mapping and section drawing. Preparation of geological reports.

MINE VALUATION.—Careful and systematic methods of sampling, booking, and preparing the sample for assay. Correct sampling procedure to adopt in difficult or unusual cases. Preparation of well arranged written or graphic records for easy reference. Right and wrong ways of averaging, and a clear understanding of the principles involved. Calculation of stoping and milling widths, tonnages, and values, of the ore reserves of a mine. Incidence of stoping width and sorting on the yield, working cost, and profit.

SURVEYING.—Use of the steel tape in ordinary and precise work. The transit theodolite, its types, construction, adjustment, care, and use. Field books and methods of booking. Sources of error. Prevention and elimination of error, instrumental or otherwise. Degree of accuracy required in different classes of work, and the simplest ways of obtaining it. Co-ordinates, their calculation and application. Problems and exercises in the use of co-ordinates. Traversing with the theodolite. Minor triangulation. Connecting surface and underground. Dumpy and wye levels, their construction, adjustment, care, and use. Levelling. Tacheometric surveying and levelling. Setting out above and below ground. Contouring. Sketching. Overcoming obstacles inseparable from underground surveying. Development and stope measurement. Concise methods of South African survey practice. Inclusive checks for the detection of error in instrumental, booking, calculating, and plotting work. Lettering, and general draughtsmanship. Enlarging and reducing plans, tracing, and blue printing. Supreme importance of accuracy in all departments of survey work.

MISCELLANEOUS.—Manipulation of calculating machines. Methods of statistical presentation. Sufficient knowledge of structural design for the preparation of plans, specifications, and quantities relating to the simpler forms of foundations, dams, buildings, etc. Use of technical books and periodicals. Collection of significant data, and the compilation of reports. Ordinary mining methods and mining routine.

You will notice that, in the foregoing syllabus, I have made mention of several elementary divisions to which reference might seem to be more fittingly made in the prospectus of a preparatory school. I have included them because I wish to draw attention to the desirability of acquiring a more specialized knowledge of such simple processes than the average school or college course enables one to obtain. It is not common, for instance, to find, outside of financial circles, men who can cast long columns of figures with accuracy and dispatch, or who are acquainted with the many time-saving arithmetical devices of the expert computer; and the mine surveyor will find that familiarity with such matters will ease his work considerably. Several things, such as the determination of azimuth from the sun or stars, the microscopical identification of rocks from thin sections, etc., I have omitted from my list because they come outside his ordinary work; but at the same time, he will be all the better equipped for any emergency if he knows how to carry them out.

Having now dealt with the training of the mine surveyor, I shall, following the practice of historians, skip a few uneventful years, and take up his work at a point where, after obtaining his certificate, he has been appointed to a producing mine of moderate size. On entering on his duties, his first care will be to ascertain what plans are in existence, and if they have been prepared in accordance with the government regulations, and that sufficient essential data exist to enable him to carry on and bring them up to date. Should the plans be much in arrear, he will proceed with the underground survey first; but before doing so he will assure himself that the instruments with which he is to work are in proper order and adjustment. In carrying out this extension of the underground survey, he will on each level begin several stations in from the last available one, and so, by overlapping, check the work of his predecessor at several points. While this work is proceeding, he will attend to any matters of urgency that crop up, such as giving lines for developers to work to, measuring up contract work, and so forth. The surface plan will then engage his attention, and, in addition to surveying in recent additional works, he may have occasion to check the position of some of the beacons by means of minor triangulation. At all times and seasons, his services will be in danger of being requisitioned by the engineer, to mark out sites, give centre lines and levels for construction work, measure up contract masonry and earthwork, and, generally, to attend to various matters appertaining to the office of clerk of works. He will also peg out and give levels for some tennis courts, and find out how few laps to the mile the sports ground will conveniently accommodate. The plotting of additional survey and geological data will always be in progress, as will also be the recording of development values and general mine sampling results. Special plans will have to be prepared from time to time for the Mines Department, consulting engineer, Municipality, insurance company, and others. He will also be expected to make a thorough study of the faults and dikes in the mine, in order that he may assist the manager in correlating them, determining their course, and in forecasting the conditions likely to obtain in the lower levels, so that development may be planned out without incurring the risk of disagreeable surprises. The measuring of contract stope and development faces will necessitate the expenditure of much pedestrian energy, and require close application at the end of the month; and in the pursuance of this arduous duty something will be learnt of the value of tact and patience as neutralizers of acerbity. Once every year, at least, the ore reserves of the mine will have to be recalculated, and this work will call for considerable judgment in determining which blocks may be legitimately classed as developed, the stoping widths likely to be obtained in breaking the ground, the best way of arriving at the value of those blocks whose peripheries have been irregularly or incompletely sampled, and in estimating the value of many other factors not amenable to rule. He will, if ambitious, keep the mine manager's certificate and the mine manager's position in view, and endeavour, by taking an active interest, if possible participation, in every phase of underground work, with a studious regard for the surface departments, to qualify himself for taking control of the mine should it ever become his privilege to do so.

This closes my address; and I hope that in it I have been fortunate enough to touch on some points of interest to you.

Prof. G. H. Stanley (*Vice-President*): I have much pleasure in moving a vote of thanks to our President for a very interesting address. As a some-time teacher of surveying myself, I do not think that he has missed much of the ground that a student who has taken up surveying for his livelihood has to cover. Of course, those students who take up a co-ordinated course will find these things dealt with. But I think that this address of Mr. Richardson's gives a very useful lead and will be a most useful guidance to the much larger number who have to work up by themselves, in selecting what studies they shall follow. They have, of course, in this very long list a very large number of subjects in which they have to become more or less proficient, and some are of more immediate importance than others, but, as I said before, there is scarcely anything which can be left out. Passing on to the latter portion of the address the President tells us what the mine surveyor has to do. Where a mine surveyor takes up this employment I think his time should be so occupied and taken up with work that he might be congratulated on the pleasing varity of his duties and should certainly never feel bored.

Mr. J. R. Williams (*Past-President*): I have much pleasure in seconding the vote of thanks to the President for his very able address. I may say, somewhere a little over 20 years ago, that Mr. Richardson and myself worked together, and I remember making a remark to the manager of the old Langlaagte Estate that he was a pretty careful youngster, and I think if the address which has been given us to-night is carefully followed, it will be found that his taste for accuracy has in no way diminished.

The President: I have to thank Prof. Stanley and Mr. Williams for their very kind remarks, and the members present for the cordial interest they have shown in my paper.

MINING COPPER ORES AT MESSINA.

By J. Allan Woodburn, M.Inst.M.M.
(Associate).

Near the Limpopo, in the extreme north of the Transvaal, there are extensive ancient copper workings, and evidence still exists of smelting having been carried on for some distance around these ancient copper workings. Small pieces of good copper ore, and slags, are to be found, and a small copper casting was recently discovered while prospecting about half a mile from Messina. These ancient workings extend over a considerable area, and go down to a depth of 70 or 80 ft. from the surface. The method of mining appears to have been that of opencast quarrying where the lode was wide, and the waste rock seems to have been thrown behind as the face advanced. The ore appears to have been cobbed and roughly concentrated on the surface, the concentrate being taken to the smelters, and the balance left either on the surface or thrown back into the open workings. The richest shoot so far opened up, which was worked by the ancients to a depth of 80 ft. from the surface, seems to have been systematically filled in with loose debris right up to the surface, so that very little evidence existed of this rich Bonanza lens.

There are several lines of ancient workings, as shown on Plan (A), and so far three distinct lodes have been developed and are at present being worked. These are called North, Middle, and South lodes. The general trend of these lodes is N E. and S W.; towards the west, however, the three lodes appear to approach each other, diverging towards the east.

The formation for fully a mile around Messina is granite, with hornblende-gabbro dikes running in various directions. On the line of reefs various rocks are found, but principally decomposed or altered granites. Both red and grey granites are exposed in the cross-cuts between the reefs, frequently of a gneissic structure, and soft decomposed felsites, like china clay, coloured red yellow, or green, are associated with the copper lodes at some places, and at others pure white quartz occupies the place of the felsites. Glance, bornite, and chalcopyrite are found in the lodes, occasionally in large irregular blocks, sometimes cemented in the cracks of the quartz, or disseminated through the mass.

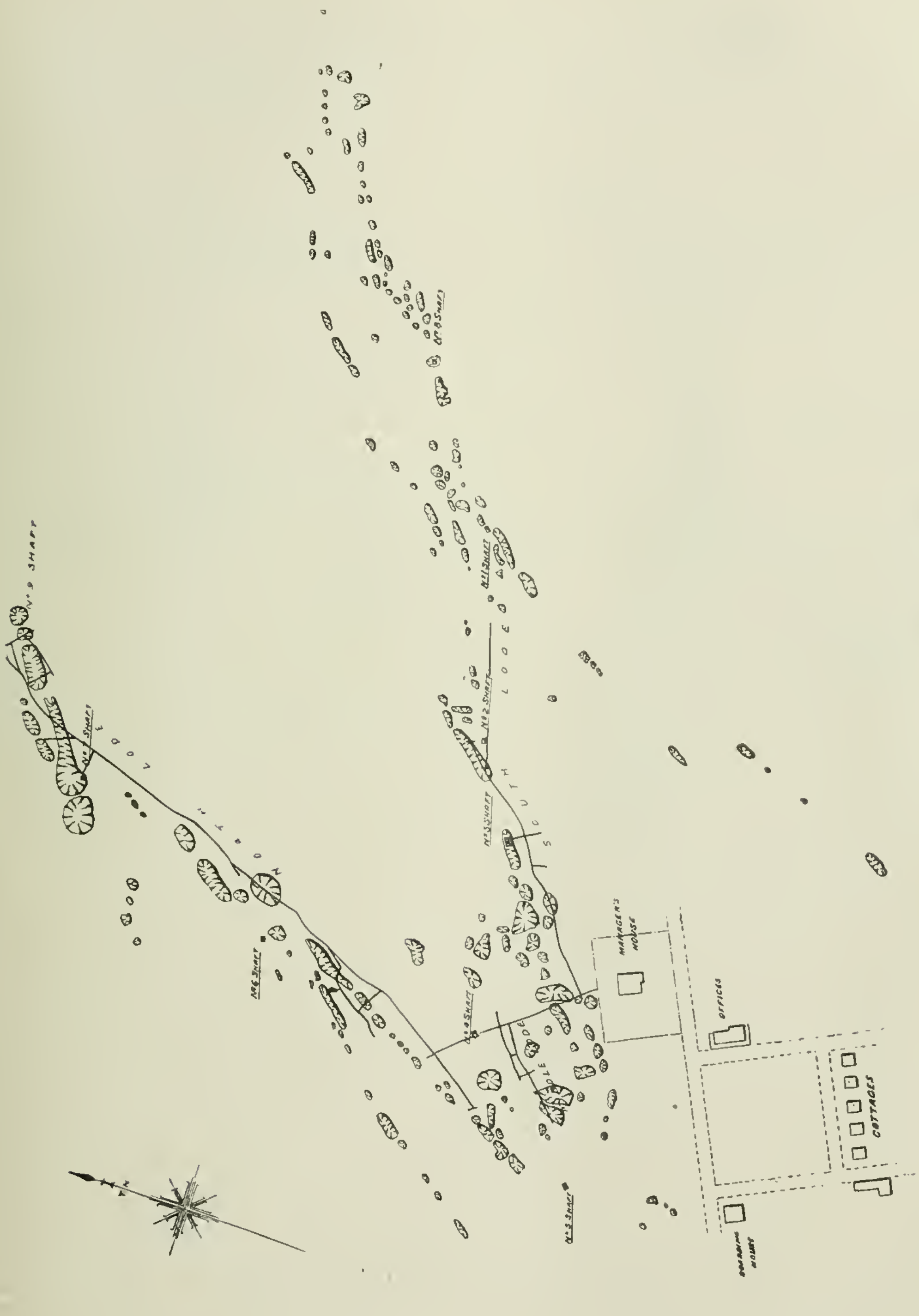
In the early development work at Messina, seven shafts were sunk on the line of the ancient workings, Nos. 1, 2, and 3 on the South lode, No. 4 on the Middle lode, and Nos. 5, 6, and 7 on the line of North lode. These shafts were all sunk to the first level, and a good deal of driving and cross-cutting done without exposing any ore of much value. No. 3 shaft was then sunk to the 2nd level, and connected with Nos. 2 and 4 shafts, and good glance ore was obtained between Nos. 2 and 3 shafts somewhat to the north of the original drive at the 1st level. Near No. 4 shaft a little glance, about 8 in. thick, was cut in the cross-cut on the line of Middle lode, and on the line of North lode some good ore 2 to 3 ft. thick was exposed. This was driven on to the east, and about 400 ft. from the cross-cut a wide mass of good pyritic ore and bornite, associated with felsite, was encountered. This mass was proved to be about 90 ft. in length, and about 25 to 40 ft. in width. At the 1st level it was also found; and while stoping in this ore above the 1st level the ancient workings were broken into, and the ground ran from the surface and filled up the stope. Work was then stopped above the 1st level, and winzes were sunk and connected with the 2nd level, proving the ore to be continuous between these levels and of fairly uniform width and value. This mass was called the "Bonanza" and forms part of the North lode. The general dip of the lodes is to the south, and the angle of dip is from 70° to 90°.

Dealing first with the North lode, which has been exposed for 1,500 ft. along the strike at the second level and proved for a depth of 900 ft., the principal feature is that it is found in lenticular masses running more or less in a uniform direction along the strike, but the connection between each mass is not easily observable. At one point the lode is from 3 to 4 ft. wide, and the walls are solid granite; then further on the lode widens out to 8 or 10 ft., and in places to about 20 ft., where there are two or three portions with granite horses between. In the Bonanza section the lode differs in character, the whole mass being mineralised throughout, and formed of irregular blocks of bornite and chalcopyrite cemented together with soft decomposed felsite, with occasional masses of quartz and altered granite. Plan (B) shows the general width and dip of this Bonanza section from surface to the 8th level, also the relative positions of Middle and South lodes.

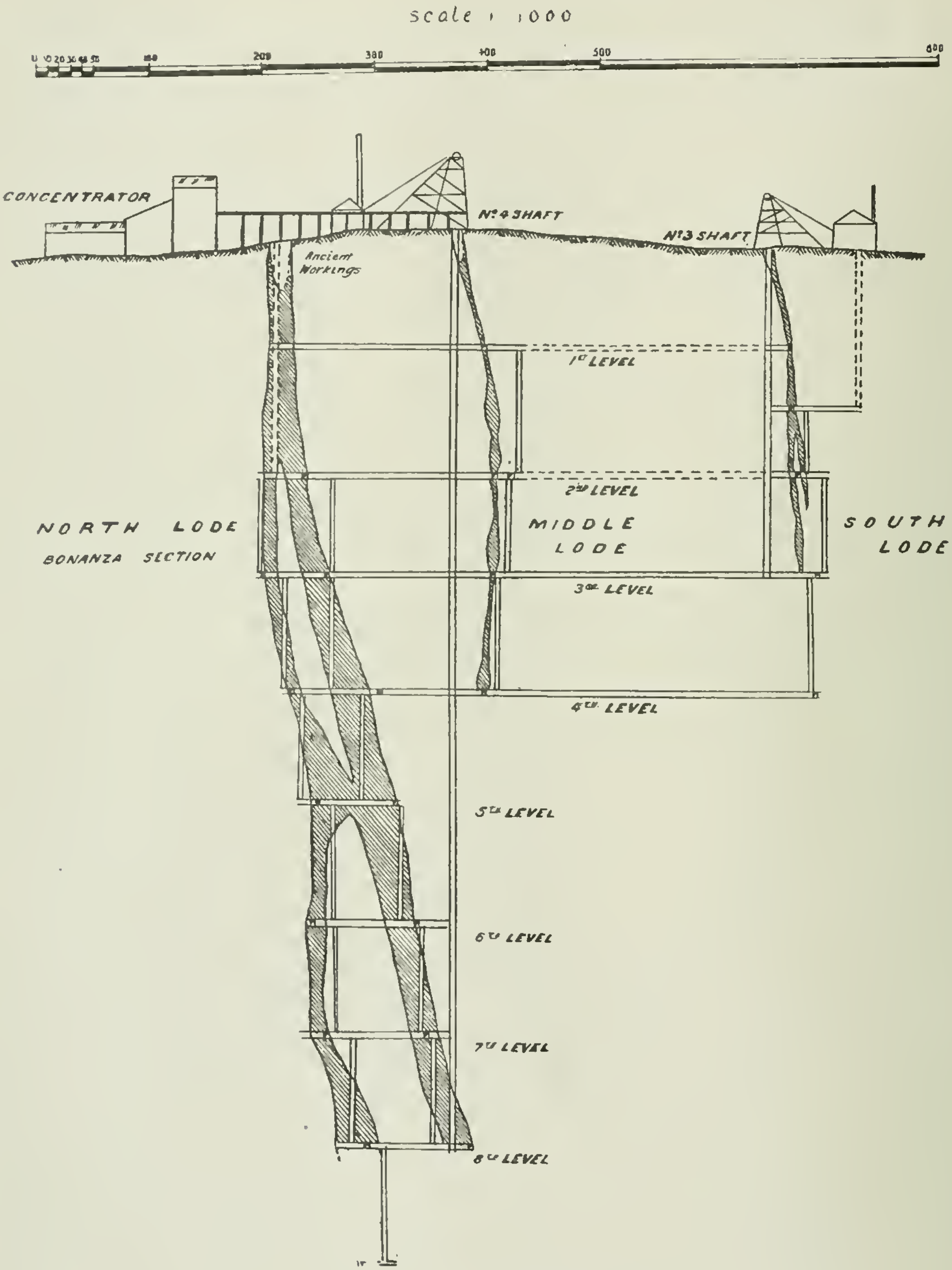
Smaller sections somewhat similar to the Bonanza section occur further to the east, near Nos. 7 and 9 shafts.

In depth the Bonanza section retains its characteristics, so far as developed, but east and west of this section the ore is generally associated with harder rocks; more quartz occurs, and the value of the ore is higher, more glance and bornite and less chalcopyrite being found. At the 3rd level, and for a few feet below it, there is strong evidence of leaching having taken place in the Bonanza section, and there is a hard mass of what looks like fused ore with bornite associated with it. Immediately above this mass, there is solid chalcopyrite 12 to 15 ft. thick and about 50 ft. in length, the average value of which is 33% copper. At this 3rd level the Bonanza section is split up into two portions, called the footwall and hanging-wall portions, and this continues to below the 4th level; but at the 5th level the space between the footwall and hanging-wall, 80 ft. wide, as proved by a cross-cut, is mineralised throughout in the centre of the mass, though east and west the two portions extend separately as shown on Plan (C). From the 5th to the 9th level the two portions seem to become farther apart, both in dip and in strike.

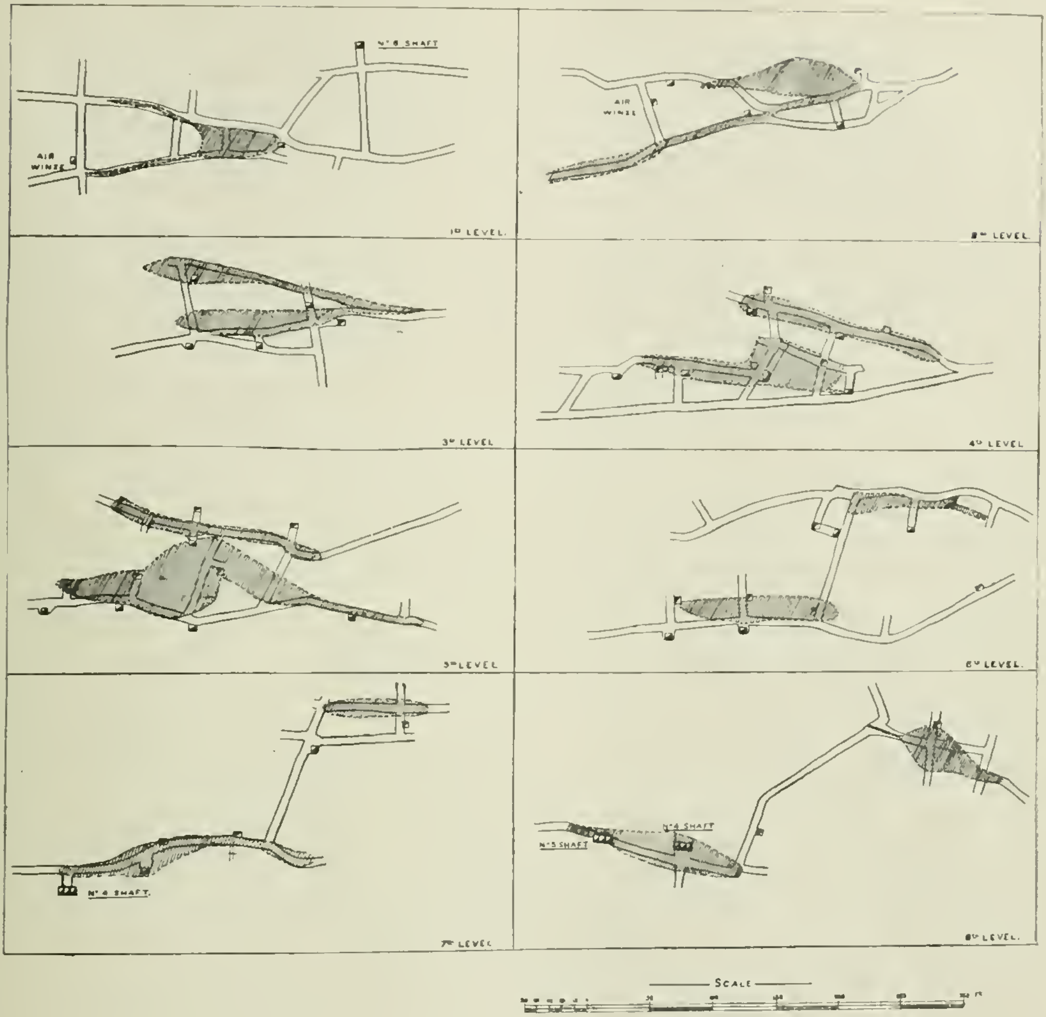
The Middle lode, which was found near No. 4 shaft strikes towards the North lode on the west. It is more erratic than the North lode, carries principally high grade bornite and glance, and swells in places to fully 20 ft. in width, with granite between the masses of ore. At other places it is worked about 2 ft. wide. There is a dike dipping about 40°, running at right angles across the lodes, which cuts through the Middle and North lodes at the 1st level near No. 4 shaft, at the 2nd level about 120 ft. to the west, and still farther to the west at the 3rd and 4th levels. The Middle lode is found on both sides of this dyke, and good values seem to follow the dip of the dyke, though generally 20 to 30 ft. away from it. Both the walls and the reef of the



PLAN A.—Ancient Workings.



PLAN B.—Section through Lodes.

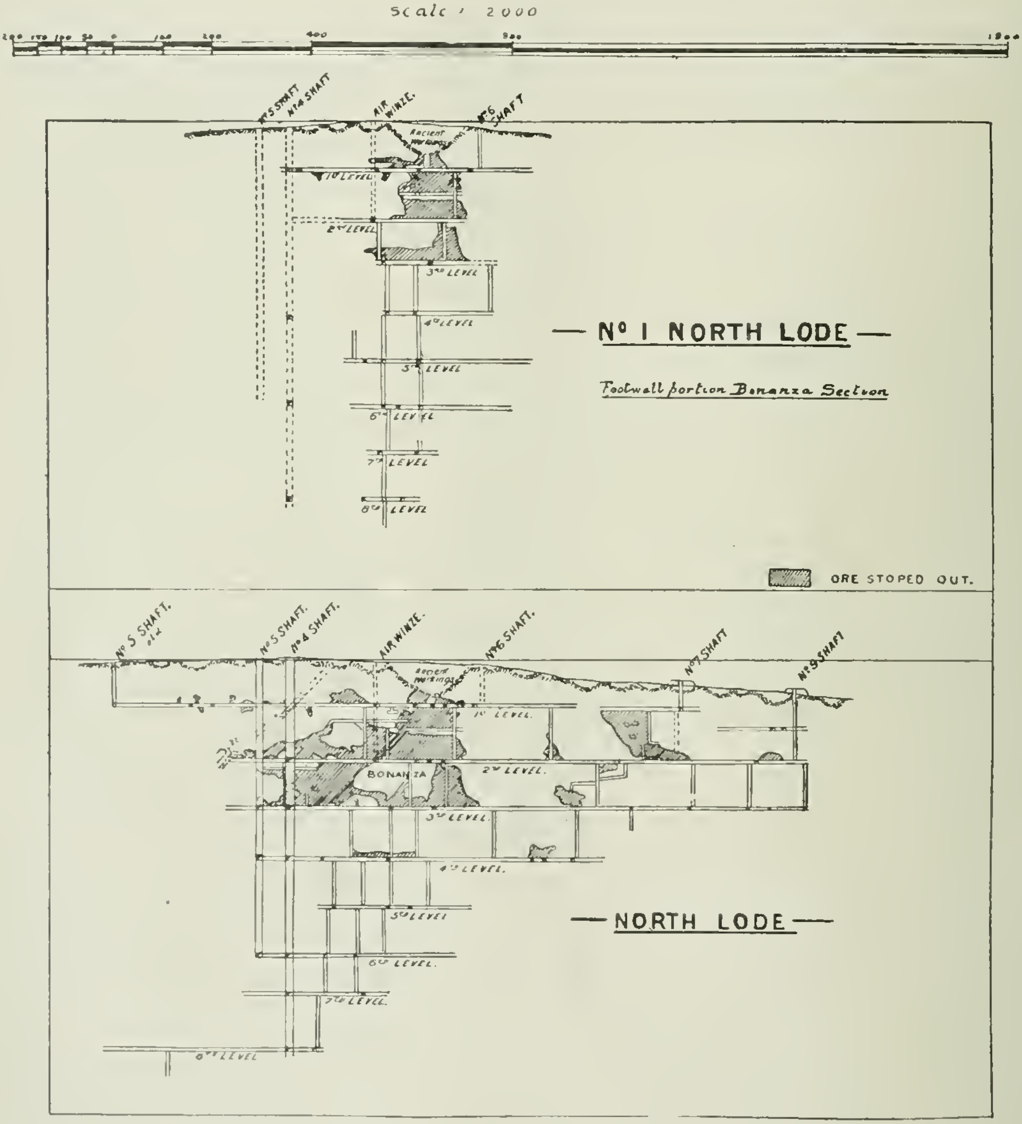


PLAN C.—Sectional Plan of Lode on Each Level of Bonanza Section.

Middle lode are hard and strong, and in this respect quite unlike the Bonanza section.

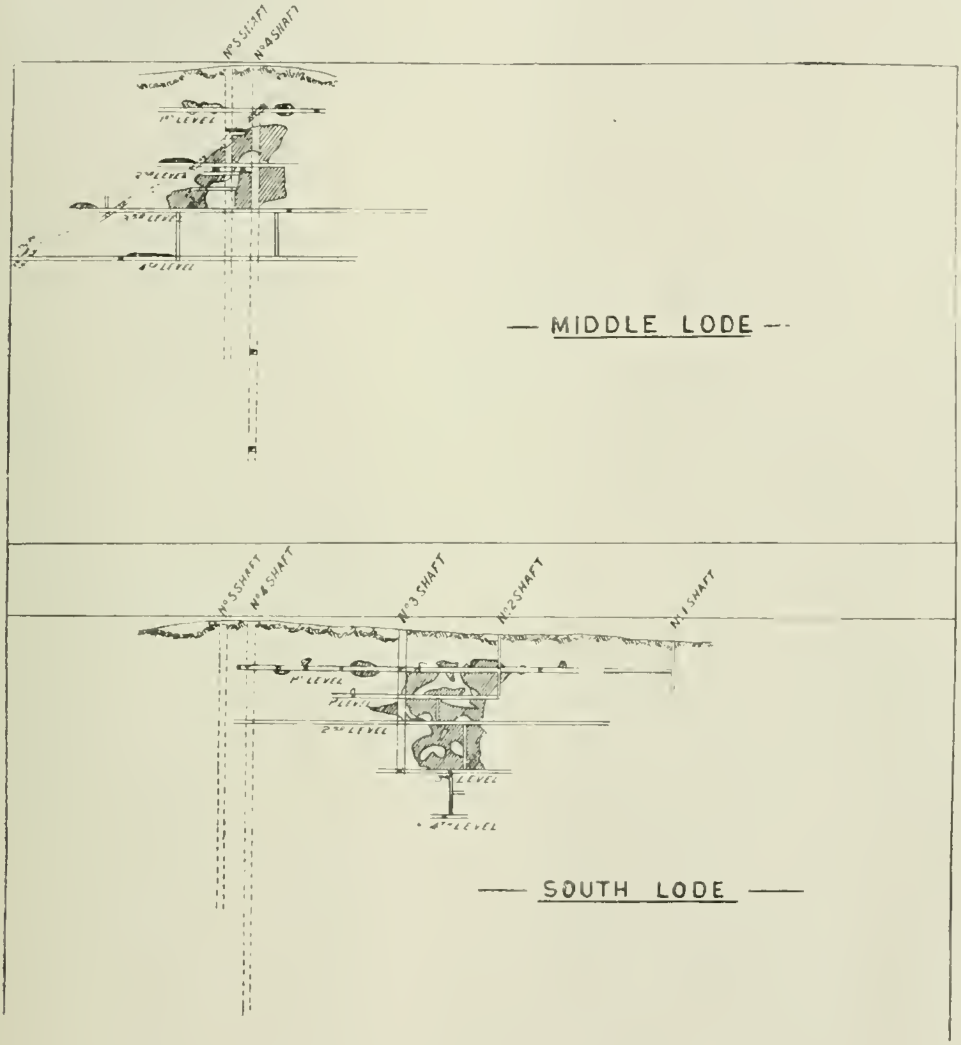
The South lode, found between Nos. 1, 2, and 3 shafts, runs roughly parallel with the Middle lode, though so far it has not been found immediately south of No. 4 shaft. This lode from the surface to the 4th level, exposed only glance ores of high value associated with quartz in hard granite walls. At places it consists of two parallel portions, each about 3 ft. thick, with 3 to 6 ft. of granite between. At other places there are small stringers of quartz and ore connecting the two parallel portions, and at these places the stopes have widened out to 12 or 15 ft. The reef is hard and the walls are good and strong. Plan (D) shows a longitudinal section of these lodes.

Development.—In a mine like the Messina, development is largely of the nature of prospecting, especially in the upper levels, and values are followed wherever found until they disappear; then the drive is continued on the general line of strike, or toward another lenticular mass intersected at a higher level. Before deviating from the line of strike, care is taken to follow any cracks or fissures showing red or green stainings, which are good indications of the proximity of the lode. When this is done and nothing of value is disclosed, cross-cuts are put in at intervals, or winzes are sunk from the level above at points where good ore exists, and should the values leave the winze, cross-cuts are put in to follow up the ore. As each level gives additional information regarding the general trend of the shoots, or masses, it becomes easier to drive the lower levels on more definite lines. When the lode is hard and not too wide, levels are driven in the lode; but in the Bonanza section the 4th level has been driven off the lode in the hanging-wall, and this section has been opened up by cross-cuts through the reef as shown on the plan of this level (E).



Plan D.—Longitudinal Sections of Lodes.

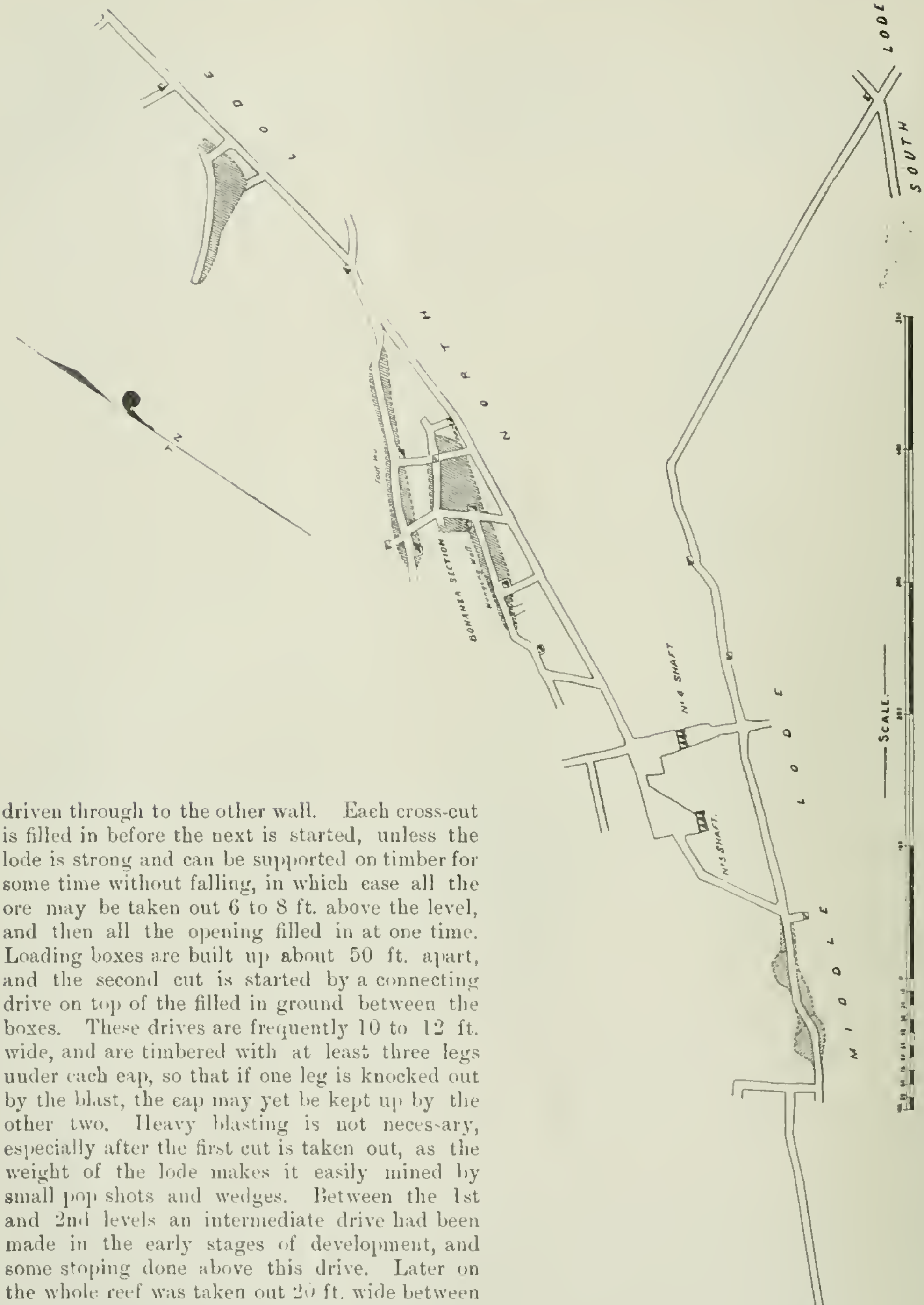
For stoping, and permanent work, this is the best method; for at the 2nd and 3rd levels, where the main drive was in the reef, it was found necessary to put through additional drives in the hanging-wall to pass this section, as the timbering required frequent repairs, and the roadway became narrow and low and unsuitable for tramming. In the South and Middle lodes, and at other portions of the North lode, the drives have been carried in the lode, though the main drives at the 4th, 6th, and 8th levels are kept as straight as possible for the sake of tramming. It is a constant question, whether to keep a drive straight or allow it to follow the bends of the lode, and much depends on the knowledge of the section under development which method is adopted, though every effort is made to prevent the drive from becoming too tortuous. Although, as has been said, development in the upper levels naturally assists in directing development in the lower levels, it frequently happens that changes take place so suddenly in both strike and dip that only by stoping out a portion of the lode can the necessary knowledge be obtained by which to direct development at the lower levels.



Plan D.—Longitudinal Sections of Lodes.

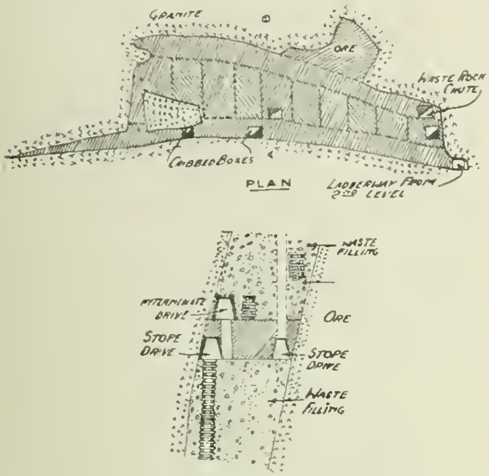
Methods of Stoping.—In the narrow portions of the lodes either underhand or overhand stoping can be carried on, and timber stulls put in for the boys to work on. If underhand stoping is done it is started from the top of a winze, and the ore shot down to the level below. Where a lode widens to 15 or 20 ft. underhand stoping, even in hard ground, may become dangerous, as portions of the wall tend to flake off, especially where small stringers pass into the sides and form false hanging. With overhand stoping, timber stulls can be relied on up to 6 or 8 ft. in width, but should the lode widen to 10 or 20 ft., then the whole stope is cleaned out and filled in with waste right up to the face, which enables the wide lode to be taken out successfully, and sent down through cribbed ore passes to the level below.

In the wide Bonanza section, where the lode is comparatively soft and liable to come down in heavy masses if left unsupported, a definite method of filling in is adopted. Winzes are sunk near each end, down which waste rock can be sent from the level above. A drive is carried along one wall of the lode, and cross-cuts are



Plan E—Plan of Fourth Level, Bonanza Section.

driven through to the other wall. Each cross-cut is filled in before the next is started, unless the lode is strong and can be supported on timber for some time without falling, in which case all the ore may be taken out 6 to 8 ft. above the level, and then all the opening filled in at one time. Loading boxes are built up about 50 ft. apart, and the second cut is started by a connecting drive on top of the filled in ground between the boxes. These drives are frequently 10 to 12 ft. wide, and are timbered with at least three legs under each cap, so that if one leg is knocked out by the blast, the cap may yet be kept up by the other two. Heavy blasting is not necessary, especially after the first cut is taken out, as the weight of the lode makes it easily mined by small pop shots and wedges. Between the 1st and 2nd levels an intermediate drive had been made in the early stages of development, and some stoping done above this drive. Later on the whole reef was taken out 20 ft. wide between this intermediate drive and the 1st level, and filled in with waste rock, and in places with timber chocks and cribs. Stoping was also started at the 2nd level, where the reef was fully 30 ft. wide, and carried up to this intermediate drive. The last cut between the filled in ground below and the filling above, was all taken out by the aid of careful timbering, as shown on Plan (F).



SECTION A-B.

Plan F.
Bonanza Section.—Method of Stoping 40 ft. above 2nd Level.

Stoping was stopped here for a time during the wet season as a good deal of water was coming down through the old workings making the soft felsite treacherous, but after the water had drained off the final cut was taken out without much blasting and with little trouble. The filled in ground had cemented together, and did not tend to run so long as no great area was left untimbered.

In the earlier stages of stoping, when only glance and bornite ores were sought for, the two ends of the Bonanza section, carrying richer ores than the central portion, were mined. Above the 3rd level a stope was carried up about 5 ft. wide for nearly 40 ft., supported by timber stulls. This stope was more like an inclined rise about 15 to 20 ft. long by 5 ft. wide, and a fair amount of water came down The lode on one side began to break away, carrying the timber stulls with it, and before the stope could be filled in, the lode broke away like a chimney, right up to the 2nd level, where through a hole in this level at the top of the chimney, waste rock was thrown down and the whole space filled up. This circumstance is mentioned mainly to show the nature of the ore body in this Bonanza section, where care must be taken to see that not more than 6 to 7 ft. of an overhead cut is taken out at one time. Even where the reef is dry and comparatively strong it frequently happens that after the cut is taken out and timbered, 3 to 4 ft of the lode break away and rests on the top of the timber; so that very little blasting is required on the next cut, only wedging, breaking, and shovelling being necessary to get the ore to the nearest box. The boxholes, or ore passes are generally built up with a partition, so that one portion can be used as a ladderway to get to the stope face, and the other used as an ore pass. Should the ore in the box choke, it can be got at from the ladderway through the partition. It is very important that two travelling ways be always kept open to each stope in this Bonanza section, so that should a fall of reef take place at any point of the stope, there will always be a way out, otherwise what may be quite a normal occurrence becomes a very unpleasant experience if workmen should be imprisoned inside a fall until it is cleared away. Many miners do not fully appreciate the necessity for this precaution, but generally a few months' work in stopes of this nature impresses upon them the advantage of the double entrance to each stope.

Waste rock for filling the stopes is obtained from the surface, and also from development drives on the upper levels. One small shaft, called the air winze, was down to below the 1st level in the footwall at the west end of the Bonanza section. This was continued to the 3rd level, and a winze sunk to the 4th level was practically underneath this shaft, so that waste rock from the surface dump can be sent down directly to the 4th level, or taken off at any upper level and distributed to any stope. From 4th to 8th level waste rock from development will be used, and rock from surface can also be lowered below this level where necessary. This air winze is only timbered for about 30 ft. from surface, and is in solid granite below this point, so that few repairs are necessary. The top of this shaft at surface is alongside the waste tram road to dump, is very convenient as a chute for waste rock

No. 3 shaft, which is down to 3rd level, had been timbered with round bush timber, and, when the question of further sinking was considered, it was thought advisable to enlarge No. 4 old shaft (a two compartment shaft down to 2nd level) and sink it to 800 ft. The North and Middle lodes, including Bonanza section, so far account for $\frac{3}{4}$ths of the developed ore, and No. 4 shaft was more convenient for these lodes, and was also at the highest level on the surface, and thus most adaptable for delivering ore to any new reduction plant. This No. 4 shaft was only intended to replace No. 3. It is a three compartment shaft, each compartment being 4 ft. × 5 ft. 6 in. in the clear, and timbered with 8 in. × 8 in. pitch pine for two sets above and below each level, and 6 in. × 6 in. at other sets. The guides are 6 in. × 4 in. pitch pine; and cages are run in the shaft, capable of carrying 20 cu. ft. trucks.

At the surface and at the 4th and 6th levels landing dogs are fixed in the shaft for the cage to rest on, and the skipman is responsible for seeing that these dogs are open before he takes the cage to a lower level. When the development in the mine improved, and the prospect of a larger tonnage was assured, the question of a second shaft arose, and it was decided to sink a new No. 5 shaft 50 ft. to the west of No. 4, and of practically the same dimensions, each compartment being 4 ft. 6 in. × 5 ft. 6 in. in the clear.

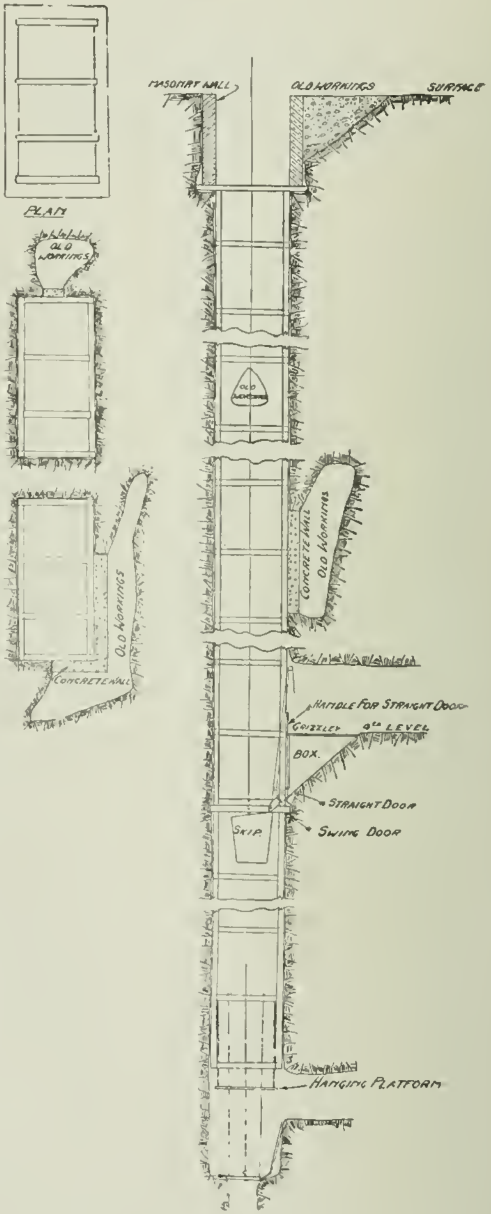
The reasons for adopting this method of practically sinking a twin shaft, instead of deepening No. 3 shaft, were the following:—

1st.—No. 3 shaft is 400 ft. from No. 4, and 20 ft. lower on the surface. Separate boilers would have to be provided, and a special surface haulage to take ore to No. 4.

2nd.—Each shaft would require to be sunk deeper as development advances, and ore could not easily be taken to either shaft alternately, should this be necessary, and, unless each level were connected between the shafts, they would practically form two separate mines.

3rd.—By sinking the twin shafts, all ore and water can be brought to either shaft, and, as all the lodes have to be connected in any case, the amount of actual driving remains the same whether the shafts are close together or far apart. After deciding to sink this No. 5 shaft, as the 1st, 3rd, 4th, 6th, and 8th levels were practically underneath, a point was fixed at each of these levels, and winzes and rises were made so that from the 1st level right down to the 8th there was a winze about 4 ft. 6 in. in diameter right underneath the new shaft. Plan (G) gives details of the methods of sinking.

The shaft was sunk by windlasses from surface to 1st level, and was built up with good masonry walls from rockhead, about 10 ft. below the surface, where the first bearer set was placed. At two points before the 1st level was reached, ancient workings were broken into, and a concrete wall was built up at the openings into these workings, and 1 in. pipes were let into the walls to drain any water which might find its way down. At the stope face in the lower old workings, 60 ft. from surface, tool marks were noticeable, and some stone hammers were found. There were also found several lengths of bush timber with the bark still on, which is not usual in modern mining practice. After the 1st level was reached the windlasses were removed, and a small 10 in. × 6 in. Lidgerwood hoist, with 2 ft. drum, which was lying on the property, was fixed up to lower the timber. The miners blasted the shaft to its required dimensions, and all the rock fell down the winze to the 3rd level, where it was placed in trucks and trammed to No. 4 shaft. A platform about 5 ft. square was hung from the bottom set of timbers and rested on the top of the winze, so that boys during drilling could easily pass from one side to the other.



Plan G.—Method of Sinking No. 5 Shaft.

When blasting, the platform was hung by a central hook to the rope from the hoist and pulled up 20 ft. above the bottom, so as to be safe from damage; the bottom end of the chain ladder was attached to the platform, and it was pulled up at the same time. The platform was lowered before the next shift started, and very little cleaning up was necessary as most of the rock fell down the winze. All the rock was cleaned up at the lower level at each shift, so there was always an opening right up to the shaft, which enormously helped the ventilation. By this method the shaft was sunk at an average cost of £7 per foot, including timbering. The cost of sinking the original winzes and rises was £3 10s. per foot, so that the whole shaft was sunk for a total cost of £10 10s. per foot. This, however, did not include the cost of hauling the rock up No. 4 shaft, which would not be more than £1 per foot, so that the 800 ft. were sunk from surface for less than £12 per foot, including the masonry wall and cement work above the 1st level. The small Lidgerwood hoist could not hold all the rope necessary to reach the 8th level, so after the 4th level was passed, the timber was taken down No. 4 shaft and hung on at each level. As there were stations every 200 ft., the sinkers could easily use the ladderways, and hoisting of men was not necessary. There was no water to trouble the sinkers, and at every blast there was a fine cut in the centre of the shaft provided by the winze, so that benches broke readily towards the centre, and did not damage the timber, which was kept within 30 ft. of the bottom. The shaft was completed before the new headgear and winding engine were erected, so that as soon as these are erected the shaft will be ready to produce ore.

Three ton skips are to be run in this shaft, and at 4th, 6th and 8th levels main stations have been laid off, and in order to save the cost of a large storage bin at each station; the method has been adopted of making small bins to hold the capacity of the skip, and put three truck loads into each bin, and discharge the whole contents into the skip each trip. On plan G is a sketch of the bin showing method of discharge. As the swing door has wings to prevent side spillage, and rests when discharging on the edge of the skip, the trouble of rock falling down the shaft should be reduced to a minimum, and the straight door will close easily when the bin is empty.

Storage in the mine is provided by the numerous boxes or ore passes between the levels, any one of which can store from 60 to 80 tons if necessary. At the three levels, 4th, 6th and 8th, where tramming is done to the main shaft, there will always be at least three boxes used as ore passes, and if only an average of 30 tons be in each box, this will give 9 boxes at 30 tons, or 270 tons available in addition to the 60 or 80 trucks tramming on the levels. This means a storage of fully a day's supply, allowing for 25% sorting, and a tonnage treatment of fully 5,000 tons per month, apart from the quantity of ore which may be lying in the various stopes. As No. 5 shaft, with 3 ton skips, has a capacity of fully double this amount, there is ample margin for future extension, and additional storage will naturally be obtained as the mine extends. For deeper development No. 4 shaft will be used to sink from 8th to 10th level, a winze and raise will be made under No. 5 shaft, and this shaft sunk as before, the rock falling to the 10th level to be hauled up No. 4 shaft. No. 4 shaft, being equipped with cages, will deal with men, material, and waste rock, while No. 5 shaft will deal with ore.

	Per ton.
The cost of mining and hauling at present is about	10s.
The cost of concentrating, including 2s. for bagging	6s.
The cost of general charges	6s.
The cost of development redemption ...	5s.
Total cost per ton treated ...	27s.

This is on 2,000 tons per month. When treating 5,000 tons per month the cost per ton should be considerably less.

Natural ventilation supplies a good current of air to all the workings. The old shafts from the surface acts as downcasts, and No. 4 shaft is an upcast. No. 5 shaft is also a downcast, so that fresh air is always available at each station. By fixing up split screens on the main roadways, the air is directed and controlled. These split screens are hung from timbers across the roadway, so that trucks can pass through by opening the screens in the centre. The lowest level is usually hot during the early development until a second winze is sunk, but by carrying 12 in. galvanized pipes in the ladder compartment 200 ft. up the shaft to the next level, and blowing a small jet of compressed air into the lower end of these pipes, the smoke and fumes from the development faces are cleared out, even though the second winze is not connected. The total amount of water in the mine is about 120,000 gallons per day, nearly all of which is caught at 4th level. The roadways are nearly all damp, and there is practically no dust.

The ore after reaching the surface, passes over a rubber sorting belt, where about 20% waste rock is sorted out, thence to the rock breaker, and to the mill storage bin. From this bin it is fed to rolls, crushed to ½ in. size, and elevated by rubber belts with buckets to the trommel screens, where it is sized, and each size falls into its respective jig, where the separation of the concentrate and middling takes place. The fine passes through cone classifiers to the Wilfley tables. The middling has been stored, and will be treated in the future either by regrinding or by smelting.

The old mill has a capacity of fully 2,000 tons per month, and the new mill, just erected, will treat 5,000 tons per month. Two reverberatory smelting furnaces have just been erected, which will deal with the accumulated middling from the old mill, of which there are about 10,000 tons on hand of an average value of 14% copper. Later, when the new mill and smelting plant have had a proper trial, they may form the subject of a future paper giving details of the process.

The old mill has been driven by a Tangye suction gas engine of 110 h.p. This engine has run for fully five years very successfully. It runs without stoppage from Monday morning to Sunday morning, when the mill is stopped for repairs, and general clean-up of the gas plant. The engine drives the whole mill, including the sorting belt, 10 in. × 7 in. rock breaker, two 24 in. × 12 in. rolls, one 8 in. bucket elevator 42 ft. high, four trommel screens, four 3-compartment jigs, one return water 4 in. centrifugal pump, throwing about 250 gal. per min. 50 ft. high, and two Wilfley tables. It also drives a small 3-drill compressor, and a small 7 h.p. lighting dynamo. This total is reckoned to be about an average of 85 h.p.; and during the month the charcoal consumption is 740 bags, which at 2s. per bag is equal to a total of £74. Taking the total time run during the month as 26 days, it gives an average cost of 0·33 penny per h.p. hour, and a consumption of about 1 lb. charcoal per h.p. hour. The new plant is to be run by electric power, the generating plant being driven by steam. The question of installing gas engines for the enlarged plant was considered, but as the supply of charcoal was decreasing, and likely to become more costly, and the economy of coal burning gas plants had not yet been sufficiently proved, it was decided to install steam boilers and quick running generators to supply the necessary energy for workshops, new mill, and pumping plant at the river, 4 miles distant. A small ice plant is running, capable of producing 400 lb. of ice per day, which is a great boon during the hot season.

Prof. G. H. Stanley (*Vice-President*): I am sure you will join with me in passing a very hearty vote of thanks to the author of this most interesting paper. I had the pleasure of going up to Messina and spending a few days there some five or six years ago in the early days of the mine. Although at that time it showed great promise, no one could have foreseen that it would attain quite to its present day development, for it is now an important copper producer. The paper breaks fresh ground, so far as the Society is concerned; I think this is the only producing copper mine in the Transvaal, at any rate, and I am sure when we have had an opportunity of going through the paper carefully, there should be some most interesting and appreciative discussion on it. I have pleasure in proposing a very hearty vote of thanks to the author for his paper.

Prof. J. Yates (*Past-President*): I have very much pleasure in seconding that, and in congratulating the author on having written a paper which is both interesting and instructive.

The President: I think the Society is fortunate in getting such a full description of one of our important base metal mines; and I hope the author will add to our indebtedness by giving us, later on, the paper on smelting operations which he has hinted at.

GOLD PRODUCTION IN RELATION TO HUMANITY; A STUDY OF THE QUESTION: HAS GOLD DEPRECIATED IN VALUE?

(Read at April Meeting, 1913.)

By Eustace M. Weston (Member).

DISCUSSION.

Mr. T. Donaldson (*Member of Council*), on behalf of Mr. Cullen, read the following contribution to discussion:

Mr. Wm. Cullen (*Past-President*): With Mr. Weston's main conclusions I am in agreement, though I must confess that some of his arguments seem a bit far-fetched. For instance, on p. 474 he says that: The one great cause of the downfall of the Roman Empire was the exhaustion of its mines of gold and silver. Now, this may be one of the contributing causes, but there were others much more important. However, this is a point of detail. The author attempted to answer the question, as many have done before him—has gold really depreciated in value? Of course the obvious answer is that it cannot do so seeing it is the standard, but it is quite another matter when we come to consider its relative purchasing power. Here we are on perfectly safe ground when we say that it has not the same purchasing power to-day as it had only a few years ago, and hundreds of reasons may be given for this. Take one example from England. A few years ago an eight hours' working day was made compulsory for all coal miners—and a very excellent Bill it was in its way—but the net result is that the consumer pays to-day 3s. a ton more for his coal. In Johannesburg to-day shop assistants are clamouring for shorter hours and higher pay. I express no opinions on the merits of the case, but I do say that if the assistants get all they want it is again the consumer who will have to pay, and his sovereign will not go so far. Even the recent strike provided us with some excellent examples in elementary economics, but these must be left for another occasion, only I cannot help saying that the public bodies of South Africa seem terribly keen on depreciating the value of the sovereign by encouraging projects in their public capacity which they would never even contemplate in their private capacity. White labour is a nice, full-mouthed political battle cry, but the poor old consumer has again to pay it out of his sovereign. Experiments in social legislature in the aggregate have precisely the same effect on the soverign, and we sometimes wonder what is going to be the end of it all. Like the author, I am a pessimist with regard to the future. However, although gold is the foundation of the world's credit, and although its increased production appears to stimulate credit in the way of making it more fluid, the actual amount of gold in reserve as coin seems to have scarcely any relation to the volume of credit transactions. What I mean is this. Take the clearing house returns of any large English city to-day as compared with ten years ago, and without giving specific figures I am certain you will find that these transactions have increased many fold, in spite of the fact that the gold reserve has been to all intents and purposes stationary during that period. Notwithstanding the enormous increase in the aggregate wealth of the world within the past two decades, there is no doubt that on the whole there is not enough in the form of credit to go round. Think of the untold millions that have been swallowed up within the past nine months through the Balkan troubles alone. The direct amount is large, but the indirect—through depreciation of securities, preparation for war, etc., etc.—must be many times greater. Then again new countries, as the author points out, in every quarter of the globe are shouting for millions, but in most cases these millions are not forthcoming. Capital is getting just a bit shy, and it is even discriminating, but that is the natural order of things when it is scarce. Then, think further of the enormous aggregate of capital which is being sunk annually in non-productive ventures such as fleets, armies, even roads and sanatoria, to say nothing of the extravagant administration of Government Departments. A very good instance of this is afforded by the English and German Workmen's Insurance Acts. No, I am convinced from a study of the question that the recent large increase in gold production has very little to do directly with the so-called depreciation of the metal, but that is only one of many causes. Things tangible, such as food, clothing, and even mealies, have, after all, got to bear burdens of taxation and direct and indirect charges of every kind. Fortunately, as in the case of mealies, we can pretty well account for a rise or a fall in price, but it is not so easy when we come to consider aggregate facts. Economists to-day are still divided into two schools of thought on this very important question, and it is well that it has been brought to our notice here, on the Rand, where 40% of the total world's supply of gold is produced.

THE ACTION OF OXIDISERS IN CYANIDING.

(Read at February Meeting, 1913.)

By Morris Green (Associate).

REPLY TO DISCUSSION.

Dr. J. Moir's explanation makes the statement of the reaction between potassium ferricyanide and potassium cyanide more intelligible and his general remarks in this connection supply the *rationale* of the actual results obtained.

Nascent cyanogen (liberated in a potassium cyanide solution) acts energetically upon gold, the usual aurocyanide being formed. Although potassium ferricyanide may accelerate the dissolution of gold in KCN Aq. due to the liberation of nascent cyanogen, yet this possible explanation does not necessitate a modification of the author's general suggestion relative to the action of oxygen and oxidisers. It is not easy to answer Dr. Moir's question, namely, the difference between two (?) reactions, one of which does not take place. Although this is not a case of concrete logic, the difficulty of reconciling the two facts (*i.e.*, that one reaction proceeds and one does not), considering the similarity of certain factors, is apparent.

Mr. Chomley, by his opening remarks, evidently regards the question from one view point only when he states that because "cyanide itself is a strong reducing agent, it appears . . . that the action of an oxidiser, in the presence of KCN, cannot do anything but harm." The value of bromocyanide need not be referred to any reaction with the tellurium, the action of nascent cyanogen obviously providing a more likely reason.

That ammonia is formed in cyaniding is well known. The oxidation, in the presence of air, of potassium cyanide to cyanate is followed by the decomposition of the latter, ammonia being produced. Hydrolysis of the cyanide also results in the formation of ammonia. An ammoniacal odour, too, may be due to the presence of methylamine and other compounds. Mr. Chomley's experience of the destruction of cyanide in an actual treatment is extremely interesting and shows the absolute necessity, in such cases, of more experimentation than the mere titration of the solution with silver nitrate. If HCN is the result of the destruction of the potassium cyanide, making the solution suitably alkaline will recover the cyanide in the desired form.

The author is sorry that Mr. A. F. Crosse's promised contribution to the discussion did not materialize.

CYANIDE PRACTICE IN INDIA.

(*Read at March Meeting, 1913.*)

By Hugh M. Leslie (Member).

REPLY TO DISCUSSION.

I have to thank you for the manner in which you have received my paper, and for the opportunity it has afforded me of placing on record what has been done in the past in India in the working of the cyanide process.

What I have written is but the briefest outline of the work accomplished in the long period under review, and it will be noted that I have made but very brief mention of what may be termed "more recent metallurgical practice." This has been done purposely for not being now on the spot; I would not have been in the position to reply definitely and with authority to the criticisms which doubtless would have been made had I taken up this section more fully.

In considering the question of what is, and what is not, the best and most efficient metallurgical practice for attaining a certain end, I think it is a common failing with metallurgists to judge and pronounce upon the practice of others by the conditions of the place in which they themselves work. That these conditions vary in different parts of the globe will be admitted. This being so, the best practice must surely be determined by the conditions of the locality. The question of capital expenditure on metallurgical development is another very important factor, and one which is also governed largely by local conditions, which include character of the ore, nature of the deposit and adjacent rock, coarseness of the gold present and associated minerals, climate, water supply, and many others. Criticisms, such as Mr. King's, which fail to take these points into consideration, serve no useful purpose.

Mr. H. A. White calls attention to a point in my paper which I had expected might be touched on, viz., the low extractor box capacity of the Mysore mine cyanide plant. When he refers to what to him seems the high value of the liquors after passing extractors, which, according to his view, must have resulted in an appalling loss of gold due to a too rigid economy in zinc, he is under a misapprehension, as there was no loss of gold from this source. I am afraid I have not made myself sufficiently clear in regard to this point, as showing what was actually my practice. The method adopted was usually to run the last washing direct to the sumps without letting it pass through extractors, and then to wash the ore with fresh water to replace the gold-bearing-liquor remaining in the ore. This too was run direct to sumps. By this means all the dissolved gold was replaced by non-gold-bearing water and the ore discharged free of any dissolved gold. The table showing the assays of liquors before and after extractors is only an example of what happened in the extractors during the first 45 hours in the treatment of a charge, and does not include the final weak and water washings which formed a very important part of the treatment. All the liquors in the works were used over and over again, so that any gold remaining in solution after passing through extractors was ultimately recovered. The whole object of the treatment was to use the smallest quantity of liquor consistent with the highest extraction obtainable by percolation, and the smallest quantity of zinc necessary to recover the gold extracted. This object was attained, and I am glad to see that there are others working on the same lines.

Mr. F. W. Watson mentions latent acidity in the water. This varied so much on the different mines and with the seasons of the year that no figures are given. The water to be contended with chiefly, was that held up in the ore before treating with cyanide. The drier the ore the less trouble in this respect, any ferrous iron present being oxidised during weathering to ferric iron. Magnesium salts were the chief trouble, and the addition of caustic soda to the first cyanide solution added to the ore was found to be the most efficient method. Aluminium salts were not present in sufficient amount to cause any serious trouble.

ADDENDUM.

Mr. H. T. Thomas *(Associate)*: The author of "Cyanide Practice in India" has given an account of the earlier development of the cyanide process on the Kolar Field and more particularly at the Mysore Mine. In recent years considerable modifications have been introduced in the methods described by Mr. Leslie.

For example, at Champion Reef, where I am employed as head of the Cyanide Department, the following variations in the practice described by the author have been introduced. The use of caustic soda has been discontinued and replaced by the more ordinary employment of lime; although an increased consumption of cyanide follows this, the gain in extraction is very marked and justifies its use. A further advantage of the use of lime is in its binding effect on the discharged residues, the dust nuisance being much diminished during the high winds prevalent in the monsoon.

Slime treatment has received particular attention and is now carried out by agitation and vacuum filtration in an up-to-date plant. The very short treatment given to the sand in the older method is now much prolonged; greater volumes of solution per ton of ore are used, with proper intervals for draining and aeration of the charge. The largely increased volumes of solution demanded by agitation and filtration of the slime, and by changes alluded to in the sand treatment, etc., are affecting precipitation methods. The small zinc area permitted to a given tonnage is much increased, the solutions being lower in gold and cyanide contents. Precipitation is falling into lines more in common with modern practice; the zinc areas allowed, however, are still below those of many places. The assays of solutions leaving the extractor boxes, from 2 to 7 gr. per ton, as given by the author, would be far too high where barren washes have to be sent to a slime filter plant and the present effluent does not exceed ½ gr. per ton. Clean-up methods were changed, acid treatment is used followed by retorting or roasting, and the gold slime, usually of high bullion content, is quickly and rapidly smelted in oil or gas furnaces. The introduction of these furnaces has greatly helped this part of the process, being quick, clean, and economical.

There are many details in which local practice has developed along original lines, but in response to a paper giving general information only, these would be out of place.

With regard to the apparent backwardness of this goldfield in adopting the methods indicated by the author in the last paragraph of his paper, and also by the Inspector of Mines to the Government of Mysore in his recent report for the year 1911, it must be borne in mind that the rate of extraction on the Kolar field was high with the methods in use; for the small amount of extra recovery there is not the same inducement to spend large sums of money on plant as there would be on the Rand, where such expenditure is attended with more profitable results.

The extraction obtained at the Champion Reef Mine is as follows:—

Milling.			
	Dwts.		
Original screen value ...	11·50		
Tailing value	2·50		
Amalgamation recovery ...			78·2%
Cyaniding.—Sand.			
Original value	2·66		
Residue value	0·88		
Sand percentage recovery		66·9%	
Slime.			
Original value	1·75		
Residue value	0·16		
Slime percentage recovery		90·9%	
Total Cyanide Recovery (Sand and Slime).			
Original value	2·50		
Residue value	0·75		
Total recovery on sand and slime		70·0%	
Or on the original ore value			15·2%
Total recovery, including amalgamation			93·4%

The method by which extraction will be further improved is by a clean classification of sand and slime, and also by classifying the sand into two grades, coarse and fine, the coarse sand being taken to tube-mills for regrinding and subsequent amalgamation, and then returned to the main stream of pulp and collected with the fine sand and slime. The slime separated is now treated and disposed of within a few hours of leaving the mill. Extensive large scale experiments have shown that the sand may receive its treatment directly after collection and transfer to the sand plant, if sufficiently fine ground. Thus the sun-drying and weathering alluded to by the author will in future be avoided.

VACUUM FILTRATION AT THE VICTORIA MILL OF THE WAIHI GOLD MINING CO., LTD., NEW ZEALAND.

(Read at May Meeting, 1913).

By William Macdonald, A.R.S.M.
(Associate).

DISCUSSION.

Mr. J. M. Neill (*Member*): In this most excellent paper the following points especially draw my attention:—

1st.—Chemical nature of ore containing 2·48% of pyrite.

2nd.—Chemical nature of slime containing fully 1% of pyrite. This is equivalent to

0·8009% based on total ore. Therefore fully 32% of original pyrite finds its way into the slime.

3rd.—The high residue value of the slime both in gold and silver, the difference in percentage extraction of those metals, and the high value of the washed residue.

4th.—The by-passing of 25% of the total ore direct to the slime plant.

The relationship of the first three points to one another may largely depend on the last point. This is likely to be the case if (as is liable to happen in an ore of the chemical nature described) ferrous silicate has been formed and minute particles of gold become encased thereby. Again, should the pyrite be, as it generally is, a carrier of gold values, the relationship is further established. Although the author says that the pyrite is sufficiently fine to be successfully treated with the main body of slime, there are no assays showing the value of this portion of the slime before and after treatment, and naturally one wonders whether "successfully treated" is to be taken only to mean that this fine material does not cause any difficulties in the subsequent formation, washing and transferring of the cake.

Examination of the tables relating to point 3 shows that some portion of the ore does not yield up its gold to the current cyanide treatment, hence the high gold contents of the washed residues. Therefore, turning to the fourth point, it seems that the installation of one or more plates, with or without blanket strakes, or some form of slime table and regrinding device to treat this material prior to cyaniding is worthy of consideration. This could probably be managed where dewatering of the by-passed product takes place.

Regarding the difference in percentage extraction of gold and silver, the following may be of interest to the author and to members generally. In this case the ore being treated differed from that under review, in that the percentages of iron sulphide and magnesia were greater, silica less, and copper, zinc and lead were replaced by carbon from traces to 1%. The trouble was the very low extraction of gold from the slimes pulp settled in an ordinary decantation plant. Having only very limited time at my disposal the exact cause was not located, but in effect it was found that the gold came into solution very rapidly, the silver rather slowly, but before the point of maximum solubility of the silver was reached, gold began to precipitate. As there was not much silver present in the ore originally, a few pennyweights left in the residue was of no account, and excellent results were obtained by cutting down the time of treatment to a minimum.

NOTES ON THE ASSAY OF MINE SAMPLES.

(Read at June Meeting, 1913.)

By Robert Dures (Member).

DISCUSSION.

Mr. John Watson *(Member of Council)*: The author's notes and experiments cover many of the difficulties which crop up in assaying mine samples.

Fine v. Coarse Sieving.—His first tabular statement shows the results obtained from a sample put through sieves of 100 mesh and 40 mesh. The difference of 2·5 dwt. in favour of the coarser mesh is contradictory to the results obtained by Mr. Whitby,* described before this Society in October, 1904.

In the author's explanation, he refers to flattened particles from Rhodesian ores, such "metallics" would, of course, make a difference. Perhaps I should not use the word "metallics" for a mine sample, but I recently saw a piece of nice-looking quartz from Gatooma district containing little cubes of native gold. Apart from the extra trouble with "metallics," I do not like a very fine mesh, for the following reason :—Let anyone take a new sieve, 90 or 100 mesh ; on holding it up to the light the interstices are clear. After using the sieve for a sample, examine it again and it will be found that a considerable portion of the interstices are choked. These cannot be cleared entirely by washing and drying or by knocking the sieve, and the next sample for which this sieve is used gets more or less "salted" by its use. Possibly, the best plan is to sieve a little waste rock with it, to clear the mesh. Personally, I think − 40 mesh is, in most cases, fine enough for the average Witwatersrand mine sample. "Rand Metallurgical Practice," I see, recommends the use of a 60 mesh sieve. With all respect to Messrs. Dures and McArthur Johnston, "tell it not in Gath, publish it not in the streets of Askelon," methinks if a little bird went along the Reef, from Geduld to Randfontein, dropping in at each little crusher house, it would find, in the great majority of cases, that no sieve at all is used for mine samples where mechanical crushers are in use. The experienced assayer can tell, by the look and the feel of it, if a mine sample is ground sufficiently fine or not.

Foreign Matter on Beads.—The cleaning of beads by tapping on a small anvil is certainly often necessary. I use a small magnifying glass (three lenses), and when I see adhering particles of magnesia from the cupel, always flatten out before weighing. Some assayers argue that this material weighs light! True, its specific gravity is considerably lower than that of gold, but it *does weigh* an appreciable amount as the author's table shows.

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* "Note on the Influence of Fine Crushing on the Assay Value," this *Journal*, Vol. V., pp. 95 and 96.

Allowance for Silver.—The author has found the percentage of silver to vary from 8·5 to 14·5 on different mines along the Rand. Before the war I weighed, parted and reweighed some hundreds of mine sample beads, notably on the old Stanhope mine, and found the silver to vary from 7% up to 13%. I believe Mr. Kyle, who was then assayer on Geldenhuis Deep, tried some thousands of mine samples and found the silver to run practically 10%—so the late Mr. Harry Line told me.

Silver in Gold Bullion.—There is, I think, much to be said in favour of the suggestion that the percentage of silver in the gold bullion may be taken as a criterion for the ore. In Sept., 1904, Mr. G. Andreoli read a paper to this Society drawing attention to the purity of the gold bullion in the Roodepoort district. In the discussion on that paper Dr. J. Loevy* quoted analyses of mill gold bullion from three mines on the Eastern Rand and from five on the West Rand.

The percentage of silver was as follows:—

	Percentage.
East Rand.—	
Geldenhuis Estate	12·54
May Consolidated	11·34
New Goch	10·43
West Rand.—	
York	6·74
Lancaster, Botha	8·18
Lancaster, Battery	8·57
Princess	7·29
Roodepoort United Main Reef	2·98

On the City and Suburban mine, I have worked out the average of the mill gold bullion (43 bars) over 8 months, Jan. to Aug., 1906, viz.:—Silver = 9·36%. The highest was 9·8%, and the lowest 8·5% silver. Again, over 12 months, March, 1909, to March, 1910 (55 bars), the average was:—Silver = 9·48%. Highest = 10·0%, lowest = 8·8%. These figures go to show that a deduction of 10% for silver in mine sample beads, would keep the calculated value of fine gold on the low (or conservative) side.

Silver in Litharge, etc.—We make up a cask of flux at a time, *i.e.*, 56 lb. of litharge are mixed thoroughly with the proportionate quantities of sodium carbonate and borax. Before using any of this flux, I make a rule of putting through two trial lots, using two portions of 3 A.T. each of pure silica, with the proportionate amount of flux and reducer. These trials give a figure varying as a rule from 0·1 dwt. to 0·3 dwt. silver, to be deducted from each assay ton when the bead is not parted. Once, before the war, and once since, I had litharge supplied which contained a very high proportion of silver, and gold = some 2 dwt. per ton. By this system of checking beforehand I detected the presence of these precious metals (where not desired) and insisted on the merchant, who had supplied it, taking his litharge back.

Variation in Reef Values.—The average manager or consulting engineer on the Witwatersrand wants a lot of mine samples assayed each month on a large mine. The variation over the miles of working faces and drives is so great that it is better to have a large number of samples assayed over a large working area, than to have a smaller number (say half the quantity) done with greater precision by parting. By permission of our manager (Mr. Glyn), and through the courtesy of our chief mine sampler (Mr. van Hasselt), I give actual figures from two drives on the City and Suburban mine, sampled every 10 ft.

A Drive West on Main Reef Leader—31st Level

Width Sampled.	Value in Fine Gold.	Average Value.
Inches.	Dwt. per Ton.	In. Dwt.
10	35·9	
Upper 6	1·7	Over 13 = 29·8
Lower 7	53·9	
Upper 6	84·5	Over 12 = 70·1
Lower 6	55·7	
Upper 5	53·9	Over 11 = 26·8
Lower 6	4·2	
8	32·3	
10	10·7	
9	80·0	
5	10·7	
6	103·4	
5	269·0	
5	10·7	
4	26·5	

The point I wish to emphasise by these assay-values is that here, on the Rand, which is generally admitted to be the most regular deposit in the world's history of gold mining, we get great variations both in reef thickness and gold values. The second case shows a reef where values for 80 ft. to 100 ft. are comparatively regular, then

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* See this *Journal*, Vol. V., p. 153.

A Drive on South Reef—29th Level.

Sampled Over.	Fine Gold per Ton.	Sampled Over.	Fine Gold per Ton.
Inches.	Dwts.	Inches.	Dwts.
36	14·7	7	7·0
28	14·6	7	18·7
37	14·4	6	1·6
32	9·9	6	0·7
26	14·5	3	1·6
28	18·1	15	4·1
38	13·1	13	3·4
34	10·5	13	1·7
34	6·9	12	1·8
16	20·0		

this reef does a "variety turn" by dropping to a thickness of merely a few inches with unpayable values

If all mine assay beads had to be parted, it would mean employing more assayers or doing fewer samples. In the face of such great variation in reef-contents I do not think we should quibble about a slight variation in percentage of silver, which could only effect the accuracy of a 10 dwt. sample by about 0·30 dwt. fine gold at most. Up to 15 or 20 years ago many gold mines in America (North and South) were worked without any assays being done, the mine manager depending on the prospecting-pan, or a cow's horn (sawn across lengthwise), to show him the value of his samples. Now the cyanide process has *arrived* and changed all that.

We assay practically all our reduction work samples in duplicate and part the beads, but for mine samples on the Central Rand I consider such practice to be a work of supererogation.

In conclusion, I fully appreciate the value of the many points touched upon by the author in his excellent and practical paper.

Mr. T. Graham Martyn (*Member*): Our thanks are due to Mr. Dures for a very concise paper, which must have involved a good deal of painstaking work.

Variations from True Value in Portion Weighed Out for Each Assay.—It appears to me, if I might offer a word of friendly criticism as to the arrangement of the figures, that the lessons to be gleaned from the fifth and sixth tables would have been much more obvious had the examples given been arranged in order of values. It is, perhaps, superfluous to reprint them, but I would suggest that individuals who are specially interested should rearrange them for themselves, when they will find that whereas differences in duplicates of as much as 12½% from the mean were obtained in some cases where the values did not exceed 7 dwt., and up to 8½% between 7 dwt. and 48 dwt., no difference exceeding 2% from the mean occurred in the five samples whose values lay between 70 dwt. and 250 dwt. Now, any accidental losses of lead would show up as heavily on a percentage basis on rich samples as on poor ones, hence we may judge that such losses were avoided in these assays: neither is it likely that an assayer who to one's personal knowledge is so scrupulously careful in cleaning and weighing the prills would quote in this table results vitiated by loss of fragments of brittle beads: hence it may be assumed that the differences are *mainly due to unequal distribution of the values in the original and duplicate assay portions weighed out from each sample.*

How then do we account for the remarkable drop in the percentage difference somewhere between the 50 dwt. and 70 dwt. values? In accordance with well established principles concerning sampling errors it seems probable that the values were mainly present in a moderately rich concentrate, and that the rich samples had such a substantial amount of that concentrate present that it was easy to get it fairly evenly distributed through them, whereas the poorer samples had so little of it present that it was practically impossible to do so. Had there been any appreciable amount of free coarse gold present, even supposing it had been abraided so as to pass a 60 mesh sieve (which appears, by comparison with Table I. to be about the grade of these samples), I do not think anything like such close agreement as shown between the duplicate rich samples would have been possible.

Limit of Accuracy in Reading the Balance.—It should be noted, by the way, that the 12½% difference in one case represents 2 dwt. actual variation, and in the other 1 dwt. only, and so far as my experience goes of balances ordinarily employed on the Rand for weighing the results of samples from underground 0·25 dwt on a 2 A.T charge is the utmost limit of accuracy and that is only to be reached by reading carefully several vibrations over a moderate range of the scale, not by simply releasing the beam without bias and watching a few moments to see if it turns. Assuming the more accurate method were adopted, the 7 dwt. may have been 7·25 dwt., and the 9 dwt. may have been 8·75 dwt., reducing the percentage difference from the mean to 9 instead of 12½. In the other extreme case this source may have been responsible for half the error. Making due allowance for this, and considering the *actual* instead of the *percentage* difference shown in the fifth table, and bearing in mind that this limit of balance reading error would in some cases aggravate and in others minimise a difference between duplicates, it appears that between 3·5 dwt. and 22·5 dwt. the actual discrepancy due to other causes may have varied only between 0·5 dwt. and 2 dwt., or about 7% from the mean. Above 22·5 dwt. this source of variation would be negligible, except that in weighing these higher values one would not ordinarily be so extremely careful with the balance. Allowing for a little relaxation of vigilance, one might suppose that 1 dwt. of the difference (0·5 dwt. each way) might be due to this in the higher values, but this only serves to accentuate the contrast between the wide discrepancies in the intermediate values and the negligible ones in the high values, for which an explanation has already been suggested.

The practical impossibility of weighing out successive portions of equal value from the same sample is, of course, exemplified very clearly in the first table, but the author has left it to his hearers to find an explanation of the remarkable improvement shown when he substituted 100 for 40 mesh, the extreme difference from the mean being reduced from 15% to only one quarter of that amount. No doubt a considerable proportion of the 40 mesh sample would pass without crushing through the 100 mesh and the remainder would, theoretically, only have to be broken into about six times as many pieces as before to pass the 100 mesh; but, if the values were present, as I think, mainly in a pyritic concentrate, then the particles of the latter would probably crush very much finer than the low grade rock, and their numbers would be increased very many times more in proportion whereby their even distribution would be greatly facilitated. The author has not told us how he divided his sample before crushing the quarter to pass 100 mesh, but supposing the very best means possible were adopted I should not be surprised to find a difference of 2·5 dwt. between one quarter and another in material which shows a difference of 12·4 dwt. between separate lots of 2 A.T. Suppose for illustration one eliminates the 41·0 dwt. value, which is nearest the mean, and plays with the remaining eight. Take them as they come in pairs, then the first pair equalling one quarter of the eight samples assays $\frac{47\cdot6+35\cdot6}{2} = 41\cdot6$ dwt., the next pair (omitting the 41·0) averages 36·9 the next 43·0 and the last 38·6 or an extreme difference of 6·1 dwt. Of course the method of quartering down the original would have been less liable to error than this and the fact that the original bulk would have been at least four times as great would also diminish the liability to error, but allowing for this it seems to me that the 2·5 dwt. difference is more likely to be due to unavoidable error in the original quartering down than to metallic gold left on the 100 mesh sieve. At any rate I have on several occasions collected such metallics as could be found on the sieve from samples from I think the same mine and have never found them assay higher than the sample under treatment; there being apparently no coarse gold present, the metallics consisting of fragments of iron or steel.

Frosted Beads.—If the sixth table be re-arranged in order of assay values an interesting fact appears, namely, that the frosted beads between 17½ and 30 dwt. lost between 7% and 8·6% on the first recupellation, whilst those between 35 and 56 dwt. lost between 10·9% and 11·4%, with two exceptions, one of 47 dwt. losing 13·2% and one of 55 dwt. losing only 8·2%. The big beads apparently freeze at an earlier stage than the smaller ones. I do not quite see why; perhaps some one will offer an explanation. I think the result of test No. 5 should have been eliminated from this table, as the loss of 4·5% on the third cupellation surely indicates that the second one was in some way faulty. I should prefer to take the 1·6% of No. 3 as representing the cupellation loss proper and adding the balance of the 4·5%, *i.e.* 2·9% to the 8·3% making 11·2% the "probable difference," but both these tests are out of harmony with those from 17·5 to 30 dwt.

In averaging the results of a number of such tests, it appears to me preferable to total up the weights before and after and find the % difference on the total weights rather than simply to divide the total % difference by the number of tests made. Applying that method to the whole series we obtain 10·13% average difference against 9·96 (not a serious matter I admit), whilst the average % loss on the third cupellation becomes 0·83, giving a "probable difference" due to retained lead of 9·3% instead of 8·75% as calculated by Mr. Dures. Taking the two groups, already mentioned, separately:—The 17·5 to 30 dwt. beads lost on the average 7·41% on the second cupellation and 1·03% on the third, giving a "probable difference" of 6·38%, whilst the 35 to 56 dwt. beads lost 10·84% and 0·56% or a "probable difference" of *10·28%* against the *6·38%* for the lower values.

Effect of Change in Fluxes.—The results of experiments on the variation of fluxes need close investigation before their significance can be properly appreciated.

In the work summarised in the second table, the attempt appears to have been made to substitute litharge for soda in chemically equivalent amounts in the slag, but this was frustrated in a measure by the presence of reducers in excess, which threw down almost all the lead into the button in three cases, leaving the slag more acid in character as the amount of soda was diminished. The results all come within the ordinary limits of sampling error in weighing on the charges, so we may conclude that the gold recovery was the same whether the slag (ignoring the borax) contained seventeen times (as in the first case) or only one and a half times (as in the last case) as much sodium oxide as lead oxide. This confirms my own experience that lead oxide to the amount of 30 gm. or so per 2 A.T. charge may safely be permitted to enter the slag greatly increasing its fluidity, enabling the borax to be reduced to a minimum. proportioned to the alumina or other refractories in the ore (much to the advantage of the crucibles), as well as permitting greater latitude in the adjustment of the reducer or oxidiser to the sample.

The Deleterious Effect of Borax on the Crucible does not appear to be realised by those who use it *ad lib* in order to obtain a fluid slag at a low temperature. The refractoriness of a crucible in the presence of a silicate slag depends chiefly on the insolubility of the alumina therein, but if borax be present in excess of that required to flux alumina or other refractories (such as zinc oxide) in the charge itself, the alumina of the crucible will dissolve and rapid corrosion ensue. It is admitted that borax is a great flux for the oxides of iron and copper, but the slag produced is one of the worst conceivable in its corrosive action, the bases readily releasing the boric acid for its attack on the alumina whilst they themselves attack the silica. A little borax seems to be useful in fluxing ores containing much oxide of iron even when alumina is absent, but silica should be added as well if not present in the sample in considerable amount. A very fluid slag will result if care be taken to reduce the iron oxide to the ferrous state by using an excess of reducing agent and no more lead oxide than will give the desired button (see Text Book of Assaying, Beringer, Ed. 6, p. 138).

In Mr. Dures' third series of experiments as in the second, the reducing power of the furnace gases, plus that of the ore itself, overshadowed that of the charcoal so that the leads vary but little and the results of these charges are also within the limits of sampling error. It would have been interesting to see the effect of substituting 5 gm. of nitre for the charcoal so as to obtain a button of between 10 and 20 gm. instead of 50 to 60. From my own experience, one may sometimes get as high a result in gold (though not in silver) from a 10 gm. button as from 60 gm. provided no lead is lost mechanically, but as this is the main risk where much of the work has to be entrusted to Kafirs a big button is advisable.

THE RECOVERY OF BLACK SAND AND FLOATING PARTICLES OF METALLIC MINERALS.

(Read at March Meeting, 1913.)

By J. M. Neill (Member).

DISCUSSION.

Mr. G. A. Robertson (*Member*) : It is difficult to see how the float described by the author can increase both quantity and quality of the black sand caught compared with the amount caught by routine methods. This is explained by a previous contributor, and if it be accepted as a fact that the float roughens the amalgam and thus allows fine particles of amalgam to be recovered with the black sand, then the float is also responsible for amalgamating fine particles of gold which would otherwise escape This is feasible.

Gold certainly does become associated (or coated) with air. I remember when attempts were made to amalgamate in the overflow with a closed circuit, that on one occasion the stamp-mill was hung up, during which time water was circulating through the classifiers and tube mills. Around the edge of the classifiers was to be seen considerable quantities of gold—a sight which would have gladdened the eye of any prospector had he seen anything approaching it in his pan. I scooped up a quantity of this rich sand and dropped it into a bucket of water standing near by. The quantity of gold that floated was surprising. This gold was however of an unusually flat shape, as if it had been hammered, in any case the air must have become associated with the gold during the dewatering process in the scoop or in its fall through air into the bucket. It is easy then to imagine the same phenomenon with gold discharging through a stamp mill screen or flowing in a launder with a subsequent drop and splash.

I have not seen this floating apron in use but I can quite well conceive that it breaks the velocity of the stream over the plates and rearranges the zone in which particles of gold or sand flow. I had an amalgamated plate under my observation some years ago. The first portion of the plate was about 3 ft. long, then there was a drop of 2½ in., and then a 9 foot plate. It was noticeable that the last 6 in. or so of the top plate yielded less amalgam in the routine scrape than the same distance on the top of the bottom plate. At the drop the plate was liable to scour, thus showing small patches of copper. A piece of insertion here did not meet the case as the pulp shot off it and scoured the plate further down. I then placed a piece of wood at right angles to the flow just at the drop and raised the wood half an inch from the plate. A piece of canvas was tacked to the wood and when the pulp was not flowing half an inch or so of the canvas rested on the plate. Naturally the flow had to pass under the canvas. The subsequent dressings of the plate showed that black sand obviously adhered to the plate about from 3 in. to 6 in. down from where the canvas floated. There was difficulty in getting a reliable "head" or "tail" sample at this plate, and as the amount of black sand caught on the plate was small, to go by actual results especially when there was only one plate after the fashion described, was possibly misleading. An objection to this device was that the canvas soon became covered with a scale. The idea was not tried on a straight plate, but obviously there was room for development and I am pleased to see that someone has perfected an apparatus which has resulted in increasing the mill recovery in one case by 5·88%.

Mr. W. R. Dowling, our late President, deprecated the idea of plates with drops in them because it destroyed the "essential true plane" of the plate. I may here point out that it is a matter of little consequence whether a plate is a true plane or not provided the flow is not concentrated to one side of the plate. If there is an even delivery from a mortar box or launder on to a plate the flow has to pass far down the plate before the flow becomes uneven. If amalgamated plates be in several pieces it matters not whether any succeeding piece of plate forms a slight obstruction to the flow, or is one step down, black sand prominently collects near these joints. Similarly, plates in front of a mortar box which have seen 10 to 15 years service usually contain a series of dents due to the usual accidents in a stamp mill and their true plane has long ago been destroyed, but no one will admit that their gold recovery has gone down. As a matter of fact by the author's reasoning and that contained herein, dents or obstructions on a plate are more liable to break the surface tension of the stream than a true plane.

No paper is complete nowadays without some more figures. It is therefore reasonable to ask the author :—

1. What is the cost of the apron?

2. How long does it last before becoming coated with scale?

3. Is any extra water required?

I believe some mines are experimenting with this apron. Their results should be submitted, and I should suggest that, until they are received, the final discussion on this paper should be delayed.

The meeting then terminated.

Obituary.

The death is announced with much regret of Mr. JOHN PATRICK FULLER, a member of the Society. Mr. Fuller, who was also an associate of the Institution of Mining and Metallurgy, joined the Society in April, 1910, when resident in Sumatra. About two years ago he came to South Africa to act as a Consulting Mining Engineer to Willoughby's Consolidated in Bulawayo. From thence he went to London, where he passed away on the 21st of April last The sympathy of the Society is extended to his wife (who is resident in New Zealand) and relatives.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

COLOUR REACTION FOR THE DETECTION OF THIOSULPHATES.—"Five cc. of concentrated sulphuric acid are added carefully to a mixture of 1—2 cc. of the solution under investigation, and an equal volume of a 10% solution of ammonium molybdate. The presence of thiosulphate is shown by a blue colour above the zone of separation of the liquids; 0·00005 grm. of sodium thiosulphate can be easily detected in this way."—E. POZZI-ESCOT, *Journal of the Society of Chemical Industry*, May 31, 1913, p. 532. (F. W. W.)

ZIRCONIUM.—"The preparation of metallic zirconium by means of magnesium or by the aluminothermic method was unsatisfactory, but good results were obtained by a method devised by the author and H. Kuzel. Pure zirconium oxide was intimately mixed with an excess of metallic calcium as finely divided as possible. The mixture was placed in an iron tube which was exhausted to about 0·5 to 0·1 mm. The bottom of the tube was heated with a blowpipe till the reduction began, after which the tube was cooled by blowing air on it and finally dipping in water. When quite cool the contents were rapidly pulverized and treated first with water which had been boiled and cooled, next with acetic acid, finally with very dilute hydrochloric acid. Air should be excluded during the whole of the treatment. The residue was finely powdered and again digested with boiled dilute hydrochloric acid. The powder was filtered off and washed with air-free water till the spectroscopic calcium reaction was no longer given. The water was removed by acetone and the powder dried in an exsiccator at ordinary temperature and then dried in a high vacuum at 250° C. to 300° C. The portion for analysis was further dried in a completely exhausted porcelain tube at 800° C. to 1000° C. This was done in a small kryptol furnace. A regulus was thus obtained which on filing and polishing gave a fine metallic surface which did not oxidize in air. The yield was about 97·5%. The metal contained 97·7% of free zirconium. The specific gravity of the powder at 18° C. was 5·98 and of the fused (sintered) metal at 17·5° C., 6·204. The metal was completely melted (not sintered) by making an arc between zirconium electrodes *in vacuo* in the Weiss-Neuman furnace. The specific gravity of the completely melted product was 6·44 at 16° C. The calcium method has been applied to other refractory metals,—*e.g.*, titanium, thorium, tantalum, etc. The melting point of this titanium was 1530° C., according to G. K. Burgess and 2350° C. according to v. Bolton. Burgess melted the metal on platinum, but his method has given good results with iron, chromium, etc., and it is probable that v. Bolton's result is too high, owing to the formation of zirconium oxide, with a melting point of about 2500° C. A sample of zirconium hydride, considered to have the formula ZrH_4, was found to contain some oxide, and the formula ZrH_2 was found for the hydride. The oxide obtained on oxidation does not appear to be ZrO_2, nor does it correspond to the suggested Zr_2O_3. A new compound, the nitride, Zr_3N_2, was obtained by heating the metal in nitrogen at 1050° C. On passing hydrogen over the nitride for 15 minutes at 1050° C., 8·7% (87%?) of the theoretical yield of ammonia was obtained. The so-called amorphous zirconium was obtained by Berzelius's method by heating potassium zirconium fluoride with potassium in an iron tube. On heating this powder it often becomes pyrophoric. It contains only about 35% of free zirconium. Its characteristic feature is its great surface; it is like a colloidal gel and retains water with great tenacity. With acids it easily passes into a hydrosol. The hydrogel precipitated from this hydrosol is very similar to the amorphous zirconium."—(*Annalen*, ccccv. 149), *Journal of the Franklin Institute*, Vol. clxxv, No. 6, June, 1913, p. 672-673. (J. G.)

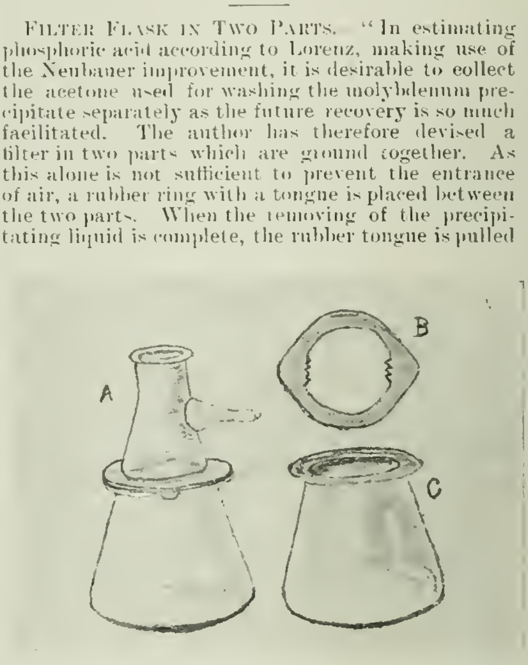
AMMONIUM SULPHATE INDUSTRY.—"Germany leads in the production of this chemical; through increasing use of by-product coke-oven and producer gas plants. A comparison of the production in all countries is made by the *Journal of Industrial and Engineering Chemistry*, for the year 1912:—

	Tons.
Germany...	465,000
United Kingdom	379,000
United States	155,000
France	68,500
Belgium	49,500
Other Countries	170,000
	1,287,000

The Chemical World, June, 1913, p. 205. (J. W.)

CLAY PURIFICATION BY ELECTROSMOSIS.—"A new method of clay purification is noted in the *Australian Mining Standard*, March 13, 1913. If colloidal solutions or liquids in which extremely fine particles are suspended are subjected to an electric current, these finely distributed particles begin to move about. This is not to be confounded with electrolysis, since no decomposition takes place, and is called electrosmosis. The new method is based upon this process and is patented in Germany and abroad. A direct current is passed through a deposit of clay, the clay separating on the anode. The importance of the process consists in the clay being not only purified, but also refined. Even with clay containing considerable portions of sand and pyrites, the result is a product of great binding power and plasticity. It does not burn, neither does it show any spots. Osmosed clay is much in demand by makers of paper, colours, pencils and crayons. In ceramic factories it is employed for improving the plasticity of clay for crockery, etc. There is another feature inherent to the process of osmosis, this being the property, under proper conditions and when a direct current is used, to remove water from clay, which must be regarded as a colloidal substance, far more effectively than may be done mechanically under heavy pressure."—*Engineering and Mining Journal*, May 10, 1913, p. 953. (H. A. W.)

FILTER FLASK IN TWO PARTS. "In estimating phosphoric acid according to Lorenz, making use of the Neubauer improvement, it is desirable to collect the acetone used for washing the molybdenum precipitate separately as the future recovery is so much facilitated. The author has therefore devised a filter in two parts which are ground together. As this alone is not sufficient to prevent the entrance of air, a rubber ring with a tongue is placed between the two parts. When the removing of the precipitating liquid is complete, the rubber tongue is pulled out until the air passes into the flask when the top part can be removed. If acetone is to be used for washing it is not necessary to use the rubber ring. To prevent any liquid from running into the joint it is advisable to let the funnel reach well into the flask and thus prevent the liquid from running down the sides. A tap can be brought on at the bottom of the flask so as to facilitate the removal of the liquids." F. MACH, *Chemiker Zeitung*, May 29, 1913, p. 651. (T. D.)



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(1) RATE OF DECOMPOSITION OF OZONE IN AQUEOUS SOLUTION. (2) DETERMINATION OF OZONE AND HYDROGEN PEROXIDE.—"(1) A study of the decomposition of relatively strong solutions of ozone in dilute acids (see this *J.*, 1912, 1030) at 0° C. has shown that, in all cases, oxygen is the sole ultimate product, though the rate of decomposition, under apparently identical conditions, varies widely. In alkaline (sodium carbonate) solution the same irregularities are not observed. Decomposition is slow in relatively acid solutions, but the speed increases as the concentration of the hydrogen ions is reduced, although not in the same proportion. The ozone concentration of the solution is ascertained by allowing it to react with potassium iodide in presence of sodium carbonate, and then acidifying and titrating with thiosulphate. (2) Hydrogen peroxide may be accurately determined by the iodometric method, in the presence of molybdic acid as catalyst, and the method is suitable also for the estimation of ozone (see above), but it cannot be

applied when both these substances are present together. This may be done, however, by allowing the solution to react first with potassium bromide in the following manner. The faintly acid solution (about 0·01 *N*) is treated at 0° C., in a stoppered vessel, with potassium bromide (sufficient to ensure the ultimate presence of about 1%), and then rather more potassium iodide is added than corresponds to the ozone present (about 3 cc. of *N*/10 iodide generally suffices), and the liberated iodine is titrated with *N*/100 thiosulphate. This gives the amount of ozone, and the hydrogen peroxide is determined by subsequently adding 10 cc. of *N*/2 potassium iodide, 1 cc. of *N*/10 ammonium molybdate, and 15 cc. of dilute sulphuric acid (1 : 5), and, after five minutes, again titrating with thiosulphate." V. ROTHMUND and A. BURGSTALLER (*Monatsh. Chem.*, 1913, *34*, 665-704), *Journal of the Society of Chemical Industry*, June 30, 1913, p. 656. (C. T.)

ARGON AND ITS PLACE IN NATURE.—"After referring to earlier work on the action of niton (radium emanation) on various substances, and the formation of helium and neon by the bombardment of the glass of a vacuum tube by cathode rays in the presence of hydrogen (this *J.*, 1913, 221) the author stated that he had confirmed the results of Collie and Patterson, and believed that under the experimental conditions, dry hydrogen polymerised into helium, whilst for the formation of neon oxygen was necessary; this might be derived from moisture or from the bombarded glass. On allowing the electric discharge to pass for 5 or 6 hours between an aluminium cathode and an anode of aluminium coated with sulphur in a vacuum tube into which dry hydrogen was introduced, evidence was obtained of the formation of argon, but no trace of neon or helium could be detected. In an analogous experiment, using selenium instead of sulphur, the gas after the experiment, when examined spectroscopically, showed very faintly the characteristic yellow and green lines of krypton."—SIR W. RAMSAY, *Journal of the Society of Chemical Industry*, June 30, 1913, p. 656. (J. A. W.)

DELICATE REAGENT FOR TIN.—"One gram of molybdenum trioxide is dissolved in a dilute solution of sodium hydroxide, the liquid made acid with hydrochloric acid in slight excess, and the volume made up to 200 cc. with distilled water. A blue coloration is formed when solution of tin chloride is added to the reagent, and the delicacy of the reaction is such that 0·000001 grm. of tin may be recognised."—C. ZENGHELIS, *Journal of the Society of Chemical Industry*, June 30, 1913, p. 675. (J. A. W.)

A RHODESIAN IPOMŒA-ROOT.—"Mr. Prosper H. Marsden, Lecturer in Materia Medica and Pharmacy at Liverpool University, in the 'Annals of Tropical Medicine and Parasitology' for June, gives particulars of a specimen of a root of *Ipomœa* brought by Dr. Yorke from Rhodesia, with the statement that as 'the drug has such a powerful cathartic and emetic action, it would be of interest to ascertain the active principle.' This rough tuberous root (weighing 260 gm. and measuring 20 cm. in length and 24 cm. in dia.) has external characters corresponding to those of *Ipomœa Horsfallia*, described by Power and Rogerson, the hard ash-grey outer portion bearing warts and dark spots of exuded resin. The inner portion was evidently decomposed, consisting of a black pulverulent mass interspersed with greyish vessels. Both simple and compound starch grains were present in quantity, as well as resin-secreting cells in the cortex. The root was found to contain 61·47% of moisture. The outer portion exhausted with hot alcohol gave 6·95% of extractive having an odour of gentian extract. This treated with water left an amount of dark-green resin corresponding to 0·85% of the root. The inner portion treated similarly gave 8·19% of soft extract and 3·7% of resin. An endeavour is being made to obtain leaves and flowers of the plant, so as to establish its identity."—P. H. MARSDEN, *Chemist and Druggist*, July 12, 1913. (J. A. W.)

METALLURGY.

REDUCTION OF TIN ORES.—"I beg to suggest the following treatment to directors and managers of tin mines. I consider that thereby not only would costs per ton be reduced, and some wear and tear be saved, but also that a saving in capital expenditure would be effected.

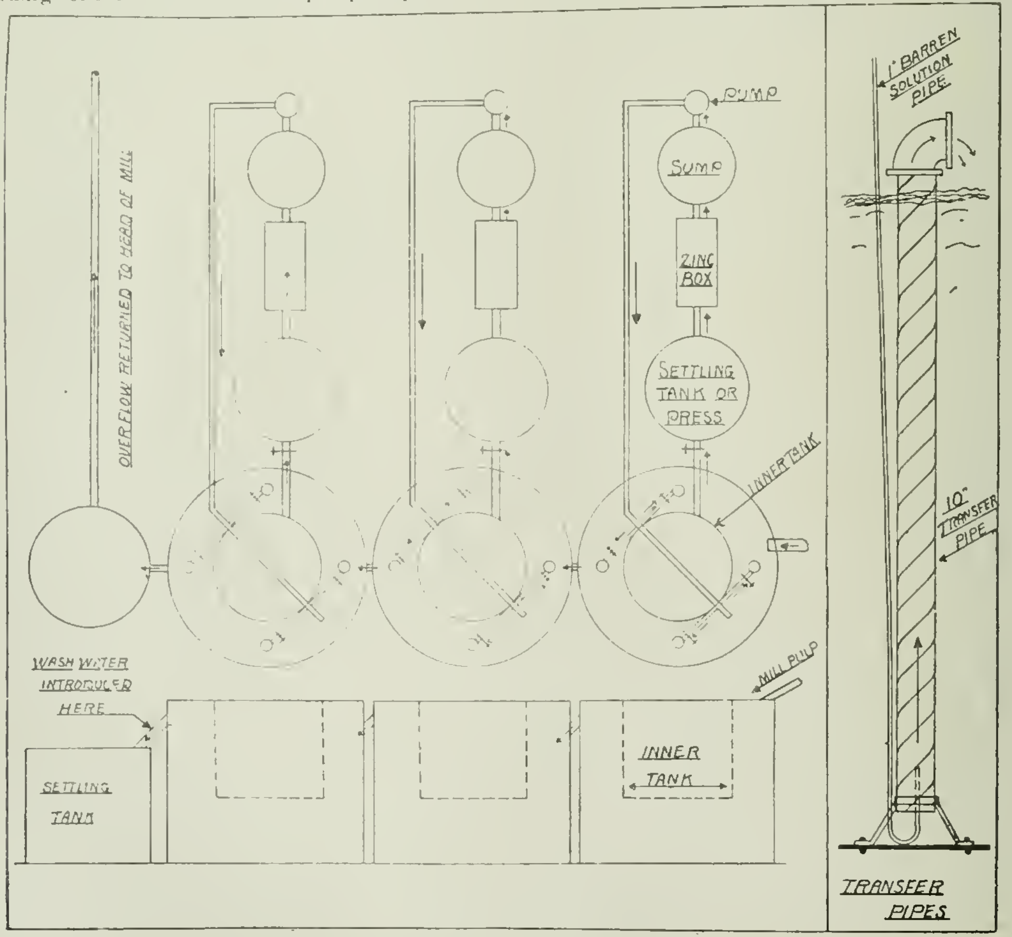
1. The ore should be washed before being subjected to any crushing, the washing sized, that of proper size going straight to a suitable fine-crushing machine, and the veritable slime to frames or settling pits.
2. The delivery from the rock breaker should be so screened that no particle under a certain size should be delivered to the stamp battery, but should be fed forthwith into the suitable fine-crusher.
3. The stamp mill should be used for coarse crushing only, and should have, instead of the ordinary screens, punched or of wire mesh, screens having longitudinal wires only, no closer together than necessary to ensure that the discharge be in size small enough for treatment in the suitable fine-crusher, thus avoiding much sliming by the stamp battery, and gaining a large duty per head of stamps.
4. As a large percentage of the ore will, by the above treatment, have been crushed finely enough for concentration, the product should be sized so as to prevent the finely-ground ore and slime being needlessly passed on to the fine-crusher.
5. The coarser particles of the delivery from the stamp battery should be fed into the suitable fine-crusher, the delivery from which should be carefully sized.
6. Very careful experiments should be made on each respective mine to ascertain whether really fine or comparatively coarse crushing be the more suitable for effective concentration.

Perhaps the day is near when, as suggested some years ago, the stamp battery will be scrapped and replaced by a series of rock breakers and suitable fine-crushing machines. As to what constitutes a suitable fine-crushing machine for ore containing cassiterite, the Huntington mill leads the way for the largest output with the least power, but it is liable to give trouble, while the tube mill slimes the ore. Another machine which I have used gives excellent results but requires too much power for economy."—*London Mining Journal*, April 19. 1913, p. 377. (A. R.)

SHAMVA REDUCTION PLANT. — "The reduction plant of the Shamva mine in Rhodesia was due to be running before the end of the year. Ore will be drawn out in the adits by petrol locomotives and deposited in a large storage bin. From this it will be fed into trucks and lowered by gravity to grizzlies placed over three No. 7½ Gates' stone breakers. After the oversize has been reduced in these the whole product, fine and coarse, will be deposited upon a belt-conveyor which discharges into trommels where the ¼-mesh product will be eliminated and sent direct to the tube mills, while the oversize will be deposited into a bin feeding 56 Nissen stamps each of 2,000 lb. weight and using ¼ in mesh screen, the ore being crushed in cyanide solution. The capacity of the plant will be in the neighbourhood of 50,000 tons per month and the cost of treatment is expected not to exceed 4s 6d. per ton. After leaving the stamps, the pulp will gravitate to the feed cones of eight 22 ft. tube mills, using Osborne liners. The underflow of the cones will pass through the tube mills over short copper plate and blanket tables, and thence to a centrifugal pump to be elevated back to cones again. The overflow of the tube mill cones will pass direct to eight sand cones superimposed over four others in which the sand will be classified out, and mixing with the barren solution from the slime plant extractor boxes will gravitate to eight 50 ft × 8 ft. 6 in. sand collecting and leaching vats in which the sand will be leached, washed, and deposited on the dump by rope haulage. The slime and solution overflowing the sand cones will flow to five 35 ft. × 10 ft. Dorr thickening vats, part of the clear overflow of which will run to the extractor boxes and a portion back to the mill supply vat. The thickened pulp from the discharge of the Dorr vats will be pumped up to five 45 ft. × 10 ft. Pachuca vats, connected in series, where, after air agitation, the slime pulp will run to the storage agitators connected to the Butters filter plant. The latter will consist of 336 leaves designed on the gravity system of working. All solutions will be precipitated in ordinary zinc extractor boxes and the gold slime smelted in two Morgan gas-fired melting furnaces." — *Mining and Scientific Press*, April 5. 1913, p 509. (A. McA. J.)

Slime Agitation.—" The Ogle Mountain Mining Co , of Oregon City, Oregon, will build a mill this coming summer in which the slime will be agitated by circulating barren cyanide solution instead of air. This mill has been designed from data worked up at the 350 ton cyanide plant of the Veta Colorado at Parral, Mexico, where the plan worked admirably, reducing the tailing 12 gm. to 15 gm. silver per ton and also cutting the time for agitation from 72 to 48 hours. I believe that the better extraction and shortening of the time was due to the displacement of the pregnant solution surrounding each particle of silver by more active barren solution during each passage through the transfer pipe

In the figure is shown the flow sheet of the cyanide plant of the Ogle Mountain Mining Co.'s mill which will be from 100 to 125 ton capacity. The agitating



Flow Sheet, Ogle Mountain Mill.

vats are three in number, 20 ft. in diameter, 20 ft. deep, and have flat bottoms. They contain an inner suspended bottomless vat, 10 ft. × 10 ft., in which the slime settles. This inner settling vat is fitted with baffle plates to stop the swirling motion of the pulp as it rises from the outer vat.

The agitating will be done through four 10 in. transfer pipes made of spiral riveted pipe and fitted with L's on top, these to be 4 in. to 6 in. above the level of the pulp surface and to point normal to radius of vat. The discharge of the pulp through these L's gives a circular swirling movement to the pulp, keeping the bottom of the vat fairly well scoured, and preventing rapid settling of the slime. These transfer pipes will be evenly spaced and 2 ft. from the side of the agitating vat. The solution as it settles in the inner clarifying vat, is decanted off through a 4 in. horizontal pipe, passing to a clarifying vat or press, thence through zinc boxes or zinc precipitation presses and to a sump from which it is pumped back to the agitating vat in which it is used for agitation. A centrifugal pump is best for this work as the pressures are low, not over 15 lb. to 20 lb. For the Oregon mill, handling 100 to 125 tons per day, tests show that a 2 in centrifugal pump at each vat, consuming not over 5 h.p., will displace the pregnant solution in the vat with barren solution every 6 hours: equivalent to four complete displacements or washes each 24 hours. Each agitating vat is connected with an independent set of inner settling vat, a clarifying vat or press, a zinc box, sump, and pump. Each unit, being complete in itself, will keep the solution from the different vats from mixing. The mill pulp entering vat No 1 displaces an equal amount, which goes to vat No. 2, and on through the system. The overflow from the last vat goes to a settling vat. The settled tailing is discharged and the overflow is pumped back to the head of the mill, where it is brought up to strength and again enters the system. The make-up water for the mill can well be added to the overflow launder from the last vat where it will act as a wash water.

The decanting pipe is proportioned in size to the tonnage in the vat and the time in which it is desired to replace the pregnant solution with barren solution. In the Oregon mill the decanting pipes are to be of 4 in., and the agitating pipes 1 in. diameter.

Agitating in Barren Solution.—Agitating with barren cyanide solution has many advantages over air agitation. For one thing, it is much cheaper. The zinc box for each vat can be arranged so there is not over 12 ft. to 15 ft. drop between the pulp level in the vat and the solution in the sump. This will give a small head to pump against and should not take over 2½ h.p. to 5 h.p. To agitate with air takes 10 h.p. to 15 h.p. to each vat. The cost of piping and zinc boxes in each case is about the same. The cost of the settling and the clarifying vats and pumps is small. This is more than off-set by the cost of the filterpress for the tailing which is done away with. In the second place, the system permits easy manipulation. There is only one valve to regulate for each vat. This is placed somewhere in the decanting pipe. The plant will run for days without adjusting this valve. The valves in the four agitating pipes are not touched after adjustment. The plant is automatic in action and needs little attention. It costs less to operate for labour, power, supplies and repairs, than a plant agitating with air and including filter pressing. One man on a shift should be able to attend to a plant up to 300 tons in size. A third important factor is the increased extraction. The recovery will be materially increased and the time of agitation shortened; due, I think, to the operator being able to carry a more dilute solution and to the displacement of the pregnant solution with barren solution. At the Veta Colorado it was necessary to carry a thick pulp, not over 2 to 1 dilution, because a more dilute solution foamed badly. The best extraction was with a pulp 3 to 1 or thinner. With barren cyanide solution this might be as thin as desired without any foaming. The system also saves the cost of an expensive air compressor and receiver. The agitating pipe is much simpler, being merely turned up at the bottom and using no valves, rubber sleeves, or complicated devices to stop the flow of pulp into the pipe whenever the air goes off. There is no danger of this happening when using cyanide solution for agitation, since the pipe is always full of solution and the pulp cannot back up in it and clog it.

Absence of Foaming.—This passage of the pulp through the transfer pipes is due to two things. The barren solution acts as a jet or hydraulic elevator forcing the circulation, diluting the pulp in the transfer column, it makes the latter lighter than that in the vat. This difference in specific gravity also helps the circulation. It results in a particularly satisfactory agitation, with no fuss, foaming, or troubles of any kind. The flow sheet illustrated includes clarifying vats or presses. In some cases where the slime contains a large amount of colloidal matter and it is necessary to crowd the passage of pulp, these may be necessary. Usually, however, with sufficient decanting capacity these will not be needed, though it might be well to put them in, to provide extra capacity for decanting. If the clarifying vat or press is used the inner (settling) vat can be made smaller in proportion and the decanting may be crowded, the clarifying device taking care of any colloidal matter which passes over.

Mr. McKinnery, of Ward, Colorado, who has a plant somewhat on these lines, except that he agitates mechanically and keeps the decanted solution passing through the system of vats against the flow of pulp, finds that the settling is practically perfect. His extraction on heavy sulphide ore is good. By the time the pulp reaches the last vat and is ready to be discharged, the dissolved gold remaining is so small in amount that it does not pay to use a filter press. Tests show that in the Oregon plant the dissolved gold in the tailing when ready to be discharged will not be over 8½c. per ton. This can be thrown away cheaper than a filterpress can be put in and operated.

Modifying Existing Plants.—An existing plant using Pachucas for agitation can easily and cheaply be modified to use barren cyanide solution for agitation, and it would probably cut out the cost of filter pressing. In making such a change, build the inner vat 8 ft. to 10 ft. diameter, 10 ft. to 15 ft. deep, and construct it from 1 in. lumber, corrugated iron, or any handy material. It does not have to be especially strong or heavy. Leave the transfer pipe in the centre as usual, cutting into the air pipe a connection from the barren solution pump, or, better, putting in a separate pipe for the barren solution. Do the settling outside the inner vat instead of inside it as shown in the flow sheet. That was the system used at the Veta Colorado mill during the summer of 1911. So far as I know, the process of agitating with barren cyanide solution is not patented and has never been used by anyone except myself. I do not intend to patent the process."—C. F., SPAULDING.—*Mining and Scientific Press*, March 1, 1913, p. 342. (A. McA. J.)

Acid Solutions in Cyanide Work.—"The successful application of the cyanide process for the recovery of gold from unroasted antimonial ores is one of those metallurgical problems the solution of which, because of the many failures made in attempting to solve it, we have come to regard as more or less impossible in the light of our present knowledge of chemistry. It is, therefore, interesting to learn that a process has recently been brought out in Australia which, it is claimed, will not only treat ores of the antimonial type, but those containing copper and other base metals.

The Gitsham process, as it is called, makes use of what is termed an 'acid solution.' This solution is prepared by acidifying an ordinary cyanide solution with sulphuric acid. The effect of the acid is to decompose the cyanide with the liberation of hydrocyanic acid, and it is the latter which is supposed to be the active agent in rendering the gold soluble. In practice, the various operations very much resemble those of the ordinary cyanide process. The sands are leached in practically the same way, the only difference being in the substitution of an acid solution for one containing free cyanide. When it comes to precipitating the soluble values, we have a return to the old conditions—the free acid is removed by the addition of lime water—after which the solution is passed through the boxes and the gold recovered in the usual manner. As in the regular process, the barren solution is restandardized before being used again, but with this difference, that after the addition of cyanide, sulphuric acid is added in sufficient quantity to render the solution distinctly acid.

Such in brief is the process which has been brought forward as a solution of one of our most perplexing problems. It is unquestionably a very simple one—a point which will commend it to a great many—and if it will successfully treat, as it is claimed it will, ores of the type mentioned, a most notable advance will have been made in the chemistry of the cyanide process. It is, of course, too early to venture any opinion as to its intrinsic merits, and we can only discuss in a general way some of its most striking features.

At first glance it seems somewhat absurd to employ a gas of the character of hydrocyanic acid for dissolving gold. To render it available, the gas must be absorbed by the solution in which it is generated as fast as it is formed, otherwise, not only would the cost of treatment be prohibitive, owing to the excessive loss through volatilization, but its escape into the air would be a constant menace to the life of those employed in the plant where the the process was used. The first point to consider, therefore, is the solubility of hydrocyanic acid in impure acid solutions. This is a subject on which the evidence is rather conflicting. Julian and Smart state that it is only slightly soluble in water, and less so in solutions of salts, a statement which, if true, would indicate the early collapse of any scheme based on the utilization of hydrocyanic acid as a solvent. In another paragraph, however, they state that in the case of dilute solutions which have become acid, the hydrocyanic acid escapes so slowly that hours after the addition of the acid the loss of hydrocyanic acid is hardly perceptible. This latter statement is perhaps much nearer the actual truth than the first one; for it agrees with the observations of a number of chemists who, in treating highly acidic ores, have failed to meet with the loss of cyanogen which the character of the ore would lead them to expect. There is but little doubt but that in commercial work when the cyanide is decomposed by acid, the hydrocyanic acid evolved is largely retained in the solution, and the chemist in making his titrations, is unaware of the change in the form of his cyanogen compound, and reports it back as an alkaline cyanide. In discussing the Gitsham process it would be only fair, therefore, in the absence of any positive evidence to the contrary, to admit that so far as utilizing hydrocyanic acid absorbed in an acid solution is concerned, the process is quite feasible.

The next point to consider, and the most important one, by the way, is the solubility of gold in impure solutions containing hydrocyanic and sulphuric acids. So far as we know, there has been very little experimental work conducted along this line, and what little has been written on the subject is very conflicting, and not at all in accordance with the experience of those who have had to work with solutions of this type in the treatment of ores. Julian and Smart state that 'when a solution becomes acid its dissolving action on gold and silver almost ceases.' This may be true in some cases, but it is a well known fact that tailings have been treated successfully with so-called 'acid cyanide solutions' where, owing to the presence of various products of decomposition, it was impossible to maintain the solution in an alkaline condition. If the effect of free acid and acid compounds such as would exist in an old bed of tailings would be to decompose an alkaline cyanide solution and liberate hydrocyanic acid, then we must infer that all of the so-called 'acid cyanide solutions' which have been used in the past were in no way different in composition from the solution which Gitsham proposes to use in his treatment of base ores. Reasoning in this way, we are inclined to think that there may be something in the process after all. At any rate, a new field of investigation has been opened up—the use of acid solutions—and it might be well for some of our own investigators to look into the subject. It might lead to important discoveries or—nothing."—*Mining and Engineering World*, April 5, 1913, p. 654. (A. R.)

Precipitation Temperatures.—"I have just noted the communication in the journal of March 15, 1913, from Edward H. Smith on the effect of temperature on precipitation. I have never known of serious difficulty in precipitation due to low temperature, although it has often been noted that with cold solutions the amount of metal retained by the zinc is greater than with warm solutions. I have not seen any cases where the tailing from zinc boxes, assayed higher than the head, but with the use of zinc dust in a precipitating plant equipped with presses not suited to the work, I have seen marked redissolving action take place more than once. The gold in a zinc box precipitate can be dissolved in cyanide solution even in the presence of an excess of zinc. I note herewith the results of some tests made on zinc box products.

A fine black precipitate that settled on the frames for holding down the zinc in the last compartment, was collected. This compartment had been filled with lead-coated zinc. Analysis of this precipitate was Au, 1·92%; Ag, 2·99; Cu, 3·30; Pb, trace; Zn, 39·53; Bi, trace; As, none; Sb, none; CaO, 13·30; MgO, trace; Fe_2O_3 plus Al_2O_3, 3·20; Insol., 2·88; loss in heating to 100° C., 13; loss in ignition, 21·8%. This precipitate treated for 48 hours with a 5 lb. cyanide solution gave 96·3% gold dissolved, and with a 10 lb. solution, 97·9%. Unfortunately no results on the silver extraction were determined.

Another test was made on a low-grade precipitate of light-grey colour obtained from below the screen of the fifth compartment of a light-flow box. This precipitate was formed under normal conditions that occasionally existed in the plant. Analysis was: Au, 0·20%; Ag, 0·26; Cu, trace; Pb, none; Zn, 57·80; Bi, none; As, none; Sb, none; CaO, 19·70; Fe_2O_3 plus Al_2O_3, 0·70; insol., 0·6; loss in heating to 100° C., 17; loss in ignition, 28; soluble in hot water, 5; soluble in KOH, 33·5 (of which 39·4% is zinc). This precipitate treated for four hours with a 5 lb. and a 10 lb. cyanide solution gave respectively 80·3% and 82·7% gold dissolved.

On another occasion the products from the different compartments of a zinc box were tested for the dissolution of gold. The boxes had seven compartments filled. From one of the boxes the precipitate was kept separate and was sampled and weighed. The samples were assayed and partially analysed. A sample of the total clean-up was also taken. In Table I. are given the results:—

TABLE I.—Zinc Box Analysis by Compartments.

Compartment	Weight	Analysis % Au	% Ag	% Cu	% Zn	CaO
1 ...	42	5·93	7·30	8·4	21·5	4·0
2 ...	28	5·76	4·43	9·1	31·5	2·1
3 ...	17	3·63	1·21	4·1	44·2	9·2
4 ...	5	3·48	2·81	6·3	49·2	4·2
5 ...	2·5	2·61	1·54	trace	41·0	3·6
6 ...	2·5	2·63	2·78	trace	46·8	4·2
7 ..	3	3·11	5·34	2·5	38·5	10·2
Averages		5·12	4·92	7·2	31·3	4·5
General sample		6·71	7·88	4·5	36·5	7·3

One gram of each of the above samples was subjected to two hours' treatment in a beaker with 200 cc. of 2 lb. cyanide solution, giving results shown in Table II.:—

TABLE II.—Cyanide Retreatment of Precipitate.

Compartment.	Mgs Gold in Residue.	Mgs Gold in Solution.	Calc. Gold Value of Solution.	Per cent. Gold Dissolved
1	36·26	19·42	\$56·60	34·9
2	36·06	18·22	53·10	33·6
3	22·77	11·10	32·40	32·8
4	20·33	11·40	33·20	35·9
5	16·56	9·62	28·10	36·8
6	14·76	12·73	37·20	46·3
7	13·12	15·81	46·10	54·7
Averages ..	31·39	16·75	\$48·80	34·8
General sample	35·03	2·703	\$78·80	43·6

It is not possible that the cyanide solution used could have dissolved all the zinc in the samples. The solution used contained 200 mg. KCy. If either of the following reactions takes place,

$$Zn + 4KCy + H_2O + O = K_2ZnCy_4 + 2KOH$$

$$Zn + 4KCy + 2H_2O = K_2ZnCy_4 + 2KOH + H_2$$

it is seen that 200 mg. of KCy are capable of dissolving only 50 mg. of zinc, and assuming further that the KOH liberated will dissolve more zinc,

$$Zn + 2KOH = K_2ZnO_2 + H_2$$

another 50 mg. of zinc will be dissolved. By the entire possible dissolving action of the solution with the production of K_2ZnCy_4 and K_2ZnO_2 but 100 mg. can be dissolved. By referring to the table of zinc analyses, it will be seen that the samples contained from 215 mg. to 492 mg. of zinc in 1 gm. The amount of cyanide actually used for the dissolution of the gold amounted in one case to 18 mg., and assuming that an equal amount was utilised for the formation of silver potassium cyanide, it would have been possible to dissolve only 80 mg. of zinc. It is obvious, therefore, that a considerable portion of undissolved zinc remained in the precipitate. It is also evident that, even in the presence of metallic zinc, gold (and very likely silver) in zinc box precipitates is soluble in cyanide solution. This may possibly be due to the mechanical condition of the zinc. Another series of tests made on another sample of precipitate showed that the weaker the cyanide solution the higher the dissolving power compared to the theoretical amount possible to dissolve.

It seems possible, therefore, that should the precipitation fall to, or approach zero, the precipitate below the screens might be attacked and the values redissolved. The tails under such a condition would carry greater values than the entering solution irrespective of temperature, although the temperature may have a slight effect on the amount dissolved. It is quite conceivable that a certain amount of dissolving action goes on under any condition below the screens on the precipitate collected there, with immediate reprecipitation as soon as the solution comes in contact with the zinc above the screens.

The redissolving action can take place in a zinc dust press to an alarming extent under certain conditions, irrespective of temperature. I have seen solutions having a value of \$2 gold precipitated with zinc dust and pumped through a press coming out with a value of \$20 gold. This happened, not once, but several times. The trouble in this case was mechanical, and due entirely to the faulty construction of the presses. After such a period of high effluent solutions the presses were opened and the bulk of the precipitate found to have settled in the bottom of the chamber. The cloths were coated with a thin coat of dirty yellow precipitate that assayed 0·34% Au, 0·45% Ag, compared with 5% Au and 4·5% Ag usually contained in the regular precipitate. The silver, as well as the gold, had been redissolved from the precipitate in this case. Is it not possible that the temperature of the solution may have little or nothing to do with the poor results in precipitation?"—JOHN GROSS.—*Engineering and Mining Journal*, May, 1913, p. 911. (H. A. W.)

RATE OF SOLUTION OF GOLD IN CYANIDE SOLUTION.—"In some testing work recently done it was found desirable to investigate the dissolving power of cyanide solutions on free gold, some of which was fairly coarse. The treatment time possible to give to the sand leaching was seven days.

In portions of the mine coarse free gold was encountered that would cause more or less loss in tails in the plant if a satisfactory extraction could not be obtained on this material. A sample of ore was taken from one of the workings where free gold was known to exist. This was carefully panned and the gold washed free from the rest of the ore; the free gold consisted of particles varying in size from the very finest to a piece about $\frac{1}{16}$ in. across. The gold was known to consist of, approximately, 60% gold and 40% silver.

The treatment outlined called for a solution strength in the plant of 4 lb. KCN per ton, and this strength of solution was used in the experiment. The free gold was placed in a small beaker and agitated with 25 cc. of solution; at regular intervals the solution was removed and the gold contents determined by assay, a fresh solution of equal quantity and strength again being agitated with the gold. At the completion of the experiment the residue, which consisted of particles of all sizes, was assayed and the gold determined, this, together with the total gold dissolved, giving the gold content of the original material.

Rate of Dissolving Free Gold in Cyanide Solution.

Treatment Time Days		Gold Dissolved Daily Mgm.	Gold Dissolved Daily Per cent.	Total Gold Dissolved Mgm.	Total Gold Dissolved Per cent.
1	...	2·23	4·67	2·23	4·67
2	...	3·45	7·23	5·68	11·90
3	...	3·80	7·96	9·48	19·86
4	...	3·55	7·44	13·03	27·30
5	...	3·69	7·72	16·72	35·02
6	...	3·33	6·96	20·05	41·98
7	...	2·50	5·25	22·55	47·23
8	...	2·21	4·62	24·76	51·85
9	...	2·37	4·99	27·13	56·84
10	...	2·38	5·00	29·51	61·84
11	...	2·14	4·48	31·65	66·32
12	...	1·65	3·45	33·30	69·77
13	...	1·31	2·74	34·61	72·51
14	...	1·16	2·43	35·77	74·94
15	...	1·20	2·51	36·97	77·45
16	...	0·99	2·08	37·96	79·53
17	...	0·84	1·76	38·80	81·29
18	..	0·75	1·57	39·55	82·86
19	..	0·61	1·28	40·16	84·14
20	...	0·52	1·09	40·68	85·23

Gold in residue, 7·06 mgm. : gold dissolved, 40·68 mgm. : total gold in sample, 47·74 mgm.

The results of this experiment, which was carried much beyond the economic point in the mill, are given in the accompanying table. It will be seen that the maximum daily extraction was obtained after three days and at the end of 20 days an appreciable amount of gold was still going into solution.

Based on the results of this experiment, together with other data to verify it, the plant was equipped with blankets placed after the stamps (amalgamation not being desirable owing to crushing in a cyanide solution), the production from this source amounting to 27% of the gold and 3% of the silver produced in the entire plant."—JOHN GROSS.—*Engineering and Mining Journal*, April 12, 1913, p. 749. (H. A. W.)

COARSE GOLD IN SAMPLING AND ASSAYING.—"A considerable discrepancy between the amount of gold won and the quantity which should apparently have been forthcoming from the direct cyanide treatment of a large quantity of low-grade ore made a thorough investigation necessary of everything which might have affected the estimates made of the quantity of gold contained in the charges and remaining in the discharged residues, and it was also intended by a correction of the possible causes of error to ensure future estimates being of a more satisfactory character.

The possibility of loss by robbery or any irregular leakage was considered and put on one side.

The weight of the crushed ore forming the charges was checked and established with reasonable accuracy.

The system of sampling the ore was examined and improved, the assay office operations were found to be satisfactory, the balances and weights were checked and found to be correct and efficient.

As the ore was of low grade, varying between 2 dwt. and 5 dwt., it had been the practice to make the assay of each sample in three fusions of two assay tons each, to weigh the parted beads together and to record the return as one-third of the total weight. This system was discontinued, each button being weighed and the weight recorded separately. The variation between the weights of the buttons was often considerable, as will be seen from the following examples taken at random :—

	ORE CHARGES.			RESIDUES.		
	Triplicate Assays in Dwt.			Triplicate Assays in Dwt.		
	1	2	3	1	2	3
	3·500	3·250	3·000	0·833	0·791	0·750
	5·250	3·750	3·250	1·125	1·125	0·791
	3·000	3·000	3·000	1·125	1·000	0·875
	6·000	4·250	4·250	1·166	0·916	0·875
	5·000	5·000	5·000	1·208	1·000	1·000
	6·250	5·375	3·500	1·500	1·250	1·125
Averages	4·883	4·104	3·666	1·159	1·013	0·903

The difference in the average of the charge assays given in Cols. 1 and 3 was 1·167 dwt., or an increase of 31·8%, and in the residue assays 0·256 dwt., or 28·3%.

Another series of six gave the following results :— 6·043 5·125 4·583 and 0·999 0·944 0·916 or difference of 31·8% as regards the charges and of 9·06% as regards the residues.

The variations are greater in the charge assays than in the residue assays. If, therefore, the value of any charge were taken as the average of the three assays, there would be considerable risk that it would be over-estimated by an error probably 10% to 15% greater than that which would affect the residues, and the theoretical extraction or difference would be in excess of the actual possible. On a large monthly tonnage, a 10% or 15% excess error would easily give an apparent shortage of 300 oz. to 400 oz. of gold on a 5 dwt. average charge assay.

As the coarser ore was always poorer than the fines, experience showed that the charge values could be easily reduced by a system of 'judicious sampling,' which would give too much coarse material and an apparently more accurate return would be obtained. This, however, was not considered a satisfactory way of overcoming the difficulty, and an endeavour was therefore made to ascertain the cause or causes of the differences shown in the triplicate assay returns.

As some members of the Institution may at some period find themselves in a similar situation, the writer has decided to place the results of his investigation on record, in the hope that they may be of some assistance. Many of the older members may have had similar experiences, but as new and younger members are continually joining who have not yet obtained their experience, the writer hopes that to them especially the views which will be set forth will be of use, and will encourage them when dealing with ores of special character, after due study, to devise methods to suit the ore which may have to treat rather than to be content with dealing with it by the generally accepted methods.

Class of Ore and Preparation of Samples.—The ore was of a somewhat special character. It was easily crushed, being a mixture of quartz in veins varying in thickness from ¼ in. to several feet; running irregularly through a crushed mass of oxidised and crumpled iron shales and bands of chert. There was a considerable quantity of oxide of iron present, derived from the decomposition of the iron pyrites with which undoubtedly the gold was originally associated. The gold itself was chiefly in very fine grains, but it was also frequently found re-deposited in the form of leaf or flake gold, and on rare occasions coarse gold was found in the quartz.

The ore was crushed dry by rolls and passed through revolving screens of four meshes to the linear inch. The crushed product would vary according to the number of revolutions made by the screens, or with the moisture in the ore, moist ore clogging the wires and causing a reduction in the size of the particles passing between them. The average product would consist of particles about ¼ in. diam. and less, 60% of the total passing a 200 mesh sieve. The finer material was always richer than the coarser.

The crushed ore was transferred dry to the cyanide treatment vats, each of these holding 200 tons.

The final ore samples, on which all calculations depended, were taken from the trucks as they were filled at the shoots of the finished ore bins. About 10 lb. of material would be taken up with a scoop, from different places, from each 16 cub. ft. truck. The samples from each vat for each 24 hours were then well mixed and quartered down to about 100 lb., which was taken to the assay office for final manipulation.

This 100 lb. would therefore represent the average of the ore treated in each charge as closely as could be obtained in a reasonably economical manner.

As the writer finally decided that the cause or causes of the varying results shown by the triplicate assays had to be looked for in the final sample, this material was thoroughly examined and tested and the conclusion formed that the variations were due mainly to the accidental presence in some of the prepared samples either of coarser grains of gold or of flaky re-precipitated gold, such particles, however, not being distributed throughout the whole of the charge of ore in the same proportion or number as were found in the portion of ore used in the fusion.

The discrepancies would be more noticeable in the charge than in the residue assays, as the larger grains of gold would, to a considerable extent, be dissolved during the cyanide treatment.

In order to prove whether the grain or two of coarse gold which might have unduly raised one of the triplicate assays were present in the same proportion throughout the whole of the charge, or whether they were comparatively isolated occurrences, two tests were made.

Test No. 1.

	Triplicate Assays.			Average.
	dwt.	dwt.	dwt.	dwt.
Charge ...	6 25	3·5	3·5	4·41
Residue ...	0·875	0·833	0·883	0·847

The first of the charge assays indicates a high value due to the presence of coarse gold (or some other cause) equal to 2·75 dwt. per ton, or, say, to 27·5 oz. in the 200 ton charge.

Two tons of the cyanided residue were then carefully concentrated, and free and undissolved gold equal to 0·172 dwt. per ton, or but 1·72 oz. in the whole charge, was collected.

Test No. 2.

	Triplicate Assays.			Average.
	dwt.	dwt.	dwt.	dwt.
Charge ...	7·00	5·25	5·00	5 91
Residue ..	0·958	0·916	0 916	0·930

The difference between the first and the second and third assays is equal to 1·875 dwt., or 18·75 oz. of gold in the whole charge.

Two tons of washed residue gave undissolved gold equal to 0 504 dwt. per ton, or 5 04 oz. in the whole charge.

These and other tests of similar nature indicated that the grains of coarse gold which caused the abnormal rise in some of the triplicate assays were not present in the whole mass of ore in similar proportions, and some method had to be devised for counteracting their influence based on some well-founded reason, not by the simple but empirical method of crossing out or rejecting any assay considered as being too high.

Although a rule-of-thumb method such as this when tested by experience could perhaps be used for what might be called the 'home consumption' of the mine management, it would not be permissible in cases such as when payment or royalties depended on the value of the ore or when an accurate knowledge of the results of the cyanide operations was necessary.

Test No. 3.

The effects of the presence of an unusually large particle of gold was ascertained by adding to a sample of known value some particles washed out from cyanided residue with the following result :—

Assay of Residue Sample.	With Addition of One Particle of Gold	With Addition of 2 Particles of Gold.
dwt.	dwt.	dwt.
0·916	1·250	1·400
0·875	1·400	2·250
0·541	1·500	2·200

These tests show how the value of a sample can be unduly raised by the accidental presence of one or two particles of coarse gold which may not exist in corresponding quantity in the bulk of the ore which the sample is supposed to represent.

Having ascertained what seemed to be a probable reason for the variations found in the triplicate assays, the next problem to be solved was how to avoid in a legitimate way the effect of these on the calculations designed to show the results of cyanide operations in the monthly run or campaign.

Trials were made of various systems for quartering down or reducing large samples to a quantity which, while representing the true value of the ore, should be sufficient for the work in the assay office. No satisfactory result was obtained, however.

'Quartering down,' however carefully carried out, affords no guarantee that the gold particles will be equally divided, especially when they are somewhat large in size. The selection by the assayer of a portion of the sample, the value of which should theoretically be equal to that of the bulk of the ore which has to be valued, is therefore merely a matter of chance, the probabilities being against a satisfactory selection being made.

The investigations were then continued in a practical or working manner

Two average samples of ore charges were obtained and sub-divided as before, the final 16 products of the fourth sub-division being assayed.

The results obtained are shown in Tables 3, 3a, 4 and 4a.

Table No. 3.

Details of Assays given in Fig. 3.

Assays, in triplicate of 16 portions of crushed ore, obtained by 'quartering down' an average sample of an ore charge of 200 tons.

Original Value, by Assay, 4·50 dwt.

Left Side.	Triplicate Assays.			Average.
	dwt.	dwt.	dwt.	dwt.
No. 16	4·000	3·750	3·750	3·833
,, 18	4·375	4·125	4·000	4·167
,, 20	4·500	4·250	4·125	4 292
,, 22	4·000	3·750	3·750	3·833
,, 24	4·375	4·000	3·875	4 083
,, 26	4·125	4·125	4 125	4·125
,, 28	4·000	3·750	3·750	3·833
,, 30	4·250	4·000	4·000	4·083
Averages ...	4·203	3 9 9	3·922	4·031
Right Side.				
No. 17	5·750	3·750	3·500	4·333
,, 19	4·375	4·375	4·250	4 333
,, 21	4·375	4·375	4·000	4·250
,, 23	4 250	4·125	4·125	1·167
,, 25	4·500	4·500	4·500	4·500
,, 27	3·750	3·500	3·500	3·583
,, 29	4·625	4·500	4·250	4·458
,, 31	4·625	4·500	4·375	4·500
Averages ...	4·513	4 203	4·062	4·265
General Averages ...	4·358	4 086	3·992	4·148

The assays in triplicate of each sub-division are also given so that the variations can be noted.

It appears to the writer that if the gold is evenly distributed there is no necessity for any elaborate quartering down. If it is not evenly distributed naturally, and if there are present a comparatively small number of larger particles, the odds are against a correct sample being obtained by any means.

It is not probable, even in the latter case, that the average of the assays of the whole of final quartering down will be correct, as the too high results will still be causing their effect. The probabilities are in favour of the average assay being *too high* rather than *too low*, as the smaller and more evenly distributed particles of gold will be in much the same numbers in every case, whereas the larger pieces, less in number, will be distributed more irregularly.

Table No. 3a.

List of triplicate assays, arranged in numerical order, with number of occurrences.

Dwt.	No. of Times.
3·500	3
3·750	8
3·875	1
4·000	8
4·125	7
4·250	5
4·375	7
4·500	6
4·625	2
5·750	1
	48

Average of 16 assays in triplicate, 4·148 dwt.

Average value of ore, by original assay, 4,500 dwt.

It is clear that the original value was too high, and also that the averages of the 48 triplicate assays is unduly raised by the higher assays.

As a rule therefore, when considerable variations are noted, it will be safer to consider that the correct value of an ore sample is *below* the average of a series of assays rather than above it; but the difficulty which now presents itself is—on what basis or system is any reduction in value to be made?

Before considering this, however, there is another matter which demands attention, and that is—to what degree of fineness should the ore sample be crushed before it is considered to be in a proper state for the assayer to take his one or two assay tons for fusion?

Table No. 4.

Details of Assays given in Fig. 4.

Assay, in triplicate of 16 portions of crushed ore, obtained by 'quartering down' an average sample of three ore charges, or 600 tons.

Original Value, Average of Three Assays, 2·916 dwt.

Left Side.	Triplicate Assays.			Average.
	dwt.	dwt.	dwt	dwt.
No. 16	2·625	2·625	2·500	2·583
,, 18	2·750	2·750	2·750	2·750
, 20	7·000	4·000	3·250	4·750
,, 22	4·500	3·000	2·625	3·375
,, 24	3 500	3·125	3·125	3·250
,, 26	4·250	4·000	3·625	3·958
,, 28	3·000	2·625	2·625	2·750
,, 30	3·125	2·750	2·625	2 833
Averages ...	3·843	3·109	2·890	3 281
Right Side.				
No. 17	3·250	2·750	2 625	2·875
,, 19	8·000	3·500	3·000	4·833
,, 21	3·000	2·750	2·750	2·833
,, 23	5·125	4·250	3·000	4·125
,, 25	3·000	3·000	2·875	2·958
,, 27	3·750	3·000	3 000	3 250
,, 29	3 000	3·000	2·875	2·958
,, 31	2·625	2·625	2·625	2·625
Averages ...	0·968	3·109	2·843	3·307
General Averages ...	3 905	3·109	2·866	3·294

In the opinion of the writer this is overdone in the majority of cases and the samples are crushed too fine.

When the gold particles are of fairly equal size and well distributed there is little need to crush the sample to pass through, say, a 120 mesh sieve. A 60 mesh or 90 mesh would probably be fine enough.

When there are larger particles, or 'metallics,' present, and these do not exist in large numbers throughout the mass, the final crushing up of these metallics until they *would* pass a 120 mesh sieve would only be introducing a source of error, as the particles accidentally present in one lot would be distributed over the whole sample and an apparently more even and correct set of assays would be obtained, although in reality they would be less correct.

Table No. 4a.

List of triplicate assays, arranged in numerical order, with number of occurrences.

Dwt.	No. of Times.
2·500	1
2·625	10
2·750	7
2·875	2
3·000	11
3·125	3
3·250	2
3·500	2
3·625	1
3·750	1
4·000	2
4·250	2
4·500	1
5·125	1
7·000	1
8·000	1
	48

Average of 16 assays in triplicate, 3·294 dwt.

Average value of ore, by original mine assays, 2·916 dwt.

The average of the 48 triplicate assays is unduly raised by the abnormally higher assays.

As an example of this, we may take the triplicate assays recorded in Tables 3 and 4, Nos. 17 and 19 respectively, where we find the following:—

5·750 dwt. and 3·750 and 3·500.
Numerical average 4·333 dwt.
8·000 dwt. and 3·500 and 3·000.
Numerical average 4·833 dwt.

The true values of these two samples are probably about 3·625 dwt. and 3·250 dwt. respectively, rather than the averages given above.

The arithmetical average of each set of three assays is, in reality, the equivalent of what would be obtained by crushing up the samples until the gold is so fine that it would be equally distributed. The material would give more uniform weighings, and they would be undoubtedly too high, but they would 'look better.'

In cases such as these, the more safe and correct plan would be, in the opinion of the writer, to use a fairly coarse sieve, say, 40 mesh, make the assays in triplicate, and to repeat with a fresh portion any assay in which the weight of one bead was notably in excess of the other two.

The following experiment shows the effect of crushing a sample to different degrees of fineness before fusion.

A sample of an ore charge was taken, well mixed and divided into four portions, which were crushed to pass through different sieves:—

Mesh of Sieve Used. per Linear Inch.	Assays in Quadruplicate.				Average.
	dwt.	dwt.	dwt.	dwt.	dwt.
15 mesh	3·000	2·625	2·625	1·500	2·437
30 ,,	5·500	2·900	2·800	2·750	3·487
40 ,,	2·875	2·875	2·800	2·700	2·812
60 ,,	2·875	2·800	2·800	2·800	2·812

The assays from the 30 mesh product are disordered by the first high assay (5·5 dwt.), which is clearly abnormal. If this is eliminated, the average would be 2·816 dwt. instead of 3·487 dwt. The correct value of the sample would probably be about 2·75 dwt. The last two, which are really the higher, appear to be more equal, due to the gold particles being broken up and more equally distributed.

When the repeated assays show more uniformity and more normal figures, the high one could, with propriety, be rejected.

When, however, the 'metallics' are really abundant, or can be found by panning in nearly every sample, it is clear that they are in sufficient quantities to appreciably raise the ore value, and they should be therefore taken into account with due caution.

General Conclusions.—The conclusions to be drawn from these experiments and tests are, in the opinion of the writer, as follows:—

Where only a comparatively small number of grains of gold of unusually large size are present in an ore, the sample for assay should be crushed to pass sieves of 30 mesh or 40 mesh only. The fusions should be made in triplicate, and in the case of notable variations repeats should be made, the abnormally high results being rejected and not included in the averages. It could with reason be considered in such cases that there would be a tendency for any 'average' to be in excess of the true value.

It may be considered that this would involve considerable work and expense, but when important interests are at stake, and time and money has been spent in collecting and preparing samples, it would be short-sighted not to obtain as full and accurate information as possible just for the sake of economising a comparatively small number of assays.

There is more chance of excess errors in the assays of ore charges than in those of the cyanide residues, and this would increase the probability that the 'monthly return' will show an apparent loss of gold, or 'unaccounted for,' greater than is actually the case."—Franklin White.—*Institution of Mining and Metallurgy*, April 10, 1913. (A. McA. J.)

Nomenclature in Ore-Dressing. — "In the paper, Gelasio Caetani gives revised definitions of the words sand, slime, and colloid, revoking his system of nomenclature published in the *Transactions* of the American Institute of Mining Engineers, Vol. 37.

There exists only one well marked division, that is, between crystalline matter and colloid. The crystalline matter includes all particles of minerals, of whatever size, that show a crystalline or granular structure; it has little or no chemical affinity with water, and its solubility under ordinary ore-dressing conditions is negligible. The word 'sand' includes all crystalline matter that is not too coarse to be treated on reciprocating-tables of the Willey type. Sand may be sub-divided into 'coarse sand,' 'fine sand,' and 'fine.' 'Coarse sand' may be defined as including all sand that is sufficiently coarse to settle rapidly and that is not carried in suspension by water running at moderate speed. Any sand coarser than 100 mesh Mr. Caetani designates as coarse sand. 'Fine sand' will settle in a short time in still water; in water running at a moderate speed it is kept in suspension; it will all pass a 100 mesh screen, and much of it is found in the undersize of a 250 mesh screen; it can be successfully concentrated on a Wiltey table. 'Fine' includes all crystalline matter so minute that it cannot be included in the fine sand: the diameter of the smaller particles is measured in ten-thousands of an inch At this extreme state of sub-division the crystalline matter loses many of the attributes of the coarser sand. The viscosity of water and electrostatic repulsion, in great part, or even completely, offset gravity and the particles will remain in suspension for a long time. These very small particles probably have some chemical affinity with water.

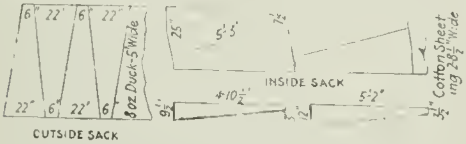
A 'colloid' is an amorphous substance of gelatinous nature having a well marked chemical affinity. A colloid reacting with ammonia or caustic soda becomes soluble, forming a turbid suspension that will not settle clear. The ammonium or sodium combines with the colloid, forming very large and complex polymerised molecules in which the alkaline radical is an exceedingly small part of the whole. This is the 'sol' form of a colloid. By adding sulphuric acid to a sol, the acid combines with the sodium or ammonium base and the colloid passes into the 'gel' form, which is insoluble and precipitates. By adding barium chloride or lime to a sol, the barium or calcium displaces the sodium or ammonium, forming a new compound which is insoluble and precipitates.

Colloid particles carry also an electrostatic charge which in most cases is negative; under the influence of this charge the particles repel each other; besides this, with most clays, the negative OH ions have a marked effect in causing the colloid to assume the sol form.

The mixture of colloid and fine Mr. Caetani terms 'slime,' though, as he says, this term is usually applied to all material that will pass through a 200 mesh screen, regardless of the amount of fine sand it may contain. In some plants it is applied in a still broader sense to all material that is treated in the slime department, even if it contain some comparatively coarse sand." —*Mining Magazine*, May, 1913, p. 379. (A. R.)

Handling Cyanide Precipitate at Lluvia De Oro. "As explained in a previous article on the practice at the mill of the Lluvia de Oro Gold Mining Co., Lluvia de Oro, Chihuahua, Mex., zinc shavings precipitation is giving way to the zinc dust method. The latter has proved much more satisfactory, since experience has enabled the correction of faulty operation. The substitution of a centrifugal for a triplex plunger pump has also been of advantage.

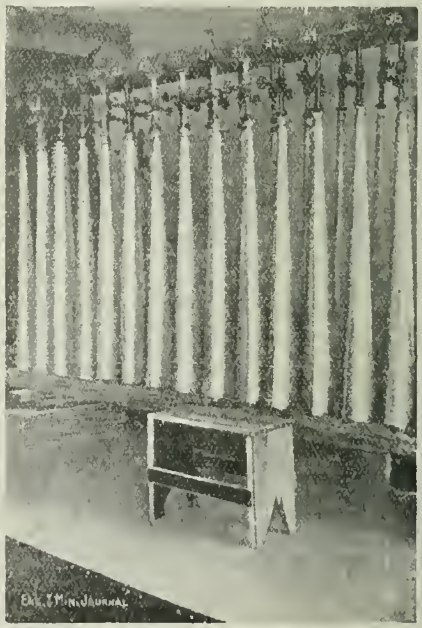
The emulsion of zinc dust and solution produced in the centrifugal pump is filtered through a press, as in the usual practice, but additional capacity was required, and a system of sacks is now used, which is proving satisfactory. After many experiments the sacks are being made double, with the outside sack of 8 oz. duck, and the inner sack of plain cotton sheeting. The inner sack is made larger than the outer sack, so that it does all the filtering, while the outer sack takes all the pressure. Sacks are made tapering, and by using the patterns as shown in the accompanying sketch, are cut from the cloth without any waste. Inner sacks are sewed on a regular household sewing machine, and outer sacks on a heavy belt-stitching machine, with heavy linen cord. Both are sewed by doubling over the edge and sewing through the four thicknesses. Inner sacks are placed inside the duck sacks by means of two long slender sticks similar to a pair of shears, and are carefully spread so that when solution is turned on, the inner sack immediately fills out against the outer sack, without any spaces between.



Patterns for making Filter Bags for Cyanide Precipitate at Lluvia de Oro.

Each of two sets of sacks is about equivalent to a large press in capacity, and consists of 44 sacks, suspended from ¾ in. pipes, which are tapered into 4 in. pipe headers and spaced 8 in. centres. The two 4 in. pipes are suspended about eight feet above the floor, with centres about 10 in. apart, and each sack connection to the header consists of a short ¾ in. nipple next to the header, with a valve, nipple, union, nipple and sleeve below. Sacks are tied at the top with heavy linen twine, the inner one first, and the outer over it, the sleeve on the lower end of the pipe serving as a shoulder to prevent the sack slipping off when under pressure.

Underneath each set of sacks is placed a flat pan or launder on a slight grade, which catches all the dirt and is piped to the barren solution vat. Each sack holds about 50 lb. of precipitate when taken off, and costs less than 50 cents.



Bag Precipitate Filter, Lluvia de Oro.

In operation, two, three or four sacks are replaced each day, and precipitation is not stopped for cleaning up. The cycle of a sack is about 11 days. Beginning when its valve is opened, the flow of solution is rapid, but after the first day, becomes much less. After its valve is closed, it is allowed to hang one day and dry, and is then disconnected at the union and taken down. The binding twine is cut, nipple washed, and a new sack tied on, ready for work. The filled sack is set in the strong room for a few days to dry farther, and is then cut open, when the precipitate is in a hard cake. The sacks are burned in the electric furnace when melting precipitate, so that there is no loss.

Regulation of the amount of solution precipitated is obtained by using any desired number of sacks. Contact of the precipitate with air has no effect, and if zinc dust is properly fed, all precipitation has occurred before filtering, so that the efforts of some managers to prevent air entering the precipitate press seem entirely unnecessary. Either in a press or in these sacks, the precipitate builds up from the bottom, and an excess of zinc, once placed in the filter, has no further appreciable effect in precipitation.

The only advantage of the press is that the precipitate is locked and safe against fire, but sacks could just as well be placed in a fireproof, locked room, or isolated building, and in every way are more satisfactory than the press." H. R. Conklin, *Engineering and Mining Journal*, May 17, 1913, p. 1001. (W. A. C.)

Grading Analyses by Elutriation.—"For the classifying of crushed materials into groups of similar properties there are two means at our disposal—that of sizing by screens and that of hydraulic classification. As, however, the difficulties attending the manufacture of a correct 200 I.M.M. mesh (0·0025 in.) have not yet been overcome, and as the commercial 200 mesh screens are unreliable, it will be more advisable to drop this mesh number altogether and to accept the preceding 120 I.M.M. mesh (0·0042 in.) as the limit of accuracy for screen measurement. For the classification of particles passing this mesh, we have unavoidably to rely on hydraulic methods based on the differences in the velocities of bodies falling through water.

Elutriation.—By far the best and most reliable elutriation apparatus is that devised by E. Schoene, which though many years in use in agricultural laboratories for the mechanical analysis of soils and of clays in the ceramic and other industries, has, so far as the author is aware, not up to this date been adopted by metallurgists.

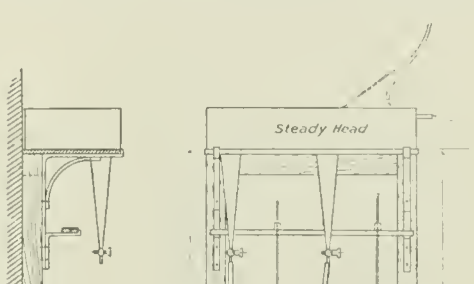
The special feature of this apparatus, illustrated in Plate I, is the provision of a piezometer immediately above the small discharge opening at the bottom of the bend in the overflow pipe. The water coming under constant pressure from a steady head, is introduced from below into the vertical classifying tube in which the material under examination is placed. Passing through this material the water rises with a velocity adjustable by a cock regulating the inflow and consequently also the overflow. Before the apparatus can be used some preparatory work must be undertaken to establish once for all the exact relation between the pressure read on the piezometer and the corresponding velocity in the classifying tube. This velocity, in addition to depending upon the pressure and the area of the discharge, depends upon the sectional area (A, in sq. cm.) of the column in the classifying tube. As this is usually not perfectly cylindrical, the average area of the upper portion is more accurately determined by dividing the number of cc. required to fill up the space between two marks by the distance between them measured in centimetres than by direct measurement of the diameter.

With regard to the relation between the pressure read on the piezometer and the volume discharged, theoretically this volume through a given orifice varies as the square root of the heights under which it takes place. Under the particular conditions of the present case the effect of this height is considerably reduced by the frictional resistance of the abnormally narrow discharge opening and by the capillarity of the comparatively small diameter of the piezometer itself. According to Schoene, for the purposes in view these particular conditions are sufficiently allowed for if from the observed pressure readings a factor (c) which he claims to be constant for all heights be deduced and applied. With this correction the above law of the relation between pressure (h, in cm.) and flow quantity (Q, in cub. cm. per sec.) is then expressed by the formula—

$$\frac{h-c}{h_n-c}=\frac{Q^2}{Q_n^2} \qquad \text{whence } c=\frac{Q^2h_n-Q_n^2h}{Q^2-Q_n^2} \text{ (A)}$$

$$\text{and } h_n=\frac{Q_n^2(h-c)}{Q^2}+c, \text{ or } Q_n=Q\sqrt{\frac{h_n-c}{h-c}} \text{ (B)}$$

To establish the direct relation between the velocity V (in cm. per sec.) in the classifying tube of a sectional area (A, in cm²) and the quantity (Q, in cub. cm. per sec.) of the overflow, we substitute in the above formula (B) for the values of Q and Q_n, those determined by the relation between the velocities and the discharge quantities. They are—



$$V=\frac{Q}{A}, \text{ and } Q=V\times A, \text{ or } Q_n=V_n\times A$$

the above equations (B) then become:—

$$h_n=\frac{V_n^2(h-c)}{V^2}+c, \text{ and } V_n=V\sqrt{\frac{h_n-c}{h-c}} \text{ (C)}$$

If, therefore, the ratio of the velocity (V) to the pressure (h) has been ascertained for one single case by actual measurement, any other pressure (h_n) required to effect a given velocity (V_n) or *vice versa*, the velocity produced by a given pressure can be established by calculation.

The constant (c) must be determined experimentally for each combination of sorting cylinder and discharge opening by the measurement of the flow quantities for two different heights of pressure according to formula (A). As the writer had no opportunity to satisfy himself personally that the factor (c) is really constant for all pressures, it seems to him to be just as expedient, and in any case safer, to establish by actual measurement for a few heights of pressure the corresponding velocities, and, by plotting the values, to draw a curve from which any intermediate functions might be read.

The apparatus being thus prepared for use, the material to be examined is placed in the cylinder, and the cock regulating the water inflow is slightly opened so as to obtain the lowest required velocity. When the water in the upper part of the cylinder is no longer turbid the beaker, in which the overflow containing the finest particles has been collected, is removed to make room for another, which receives the overflow of the next following required pressure (or velocity), and so on as long as the length of the piezometer allows such an increase of pressure. The displacement of the finest slime takes a long time, but as the level in the piezometer keeps extremely steady the apparatus can safely be left, and other work undertaken in the meantime. With the finest portion elutriated, the sand of the next following fine grades—whatever may be its fineness—settles with a remarkable rapidity, and, being perfectly permeable, the water can immediately be drawn off to the last drop.

The results of actual measurements are in perfect accord with the theoretical deduction, that for obtaining a regular reduction scale at the rate 1 : 2 in volume, or weight, the velocities determining these grades must vary at the ratio $\sqrt[3]{4}$.

The grades obtained by elutriating Rand ore (density 2·7) at velocities of 10·08, 4·0, 1·59, 0·63 and 0·25 mm. per sec., are fairly equivalent to those determined by the 120, 200, (300), (500) and (800) I.M.M. mesh respectively (the existence of the latter being theoretically assumed.

The small percentage of pyrite contained in Rand ore does not appreciably affect the results, as might be expected, because they, too, are like the quartz particles, classified and distributed over the different grades, with the only difference that in consequence of their greater specific gravity the comparatively small amount contained on each remains in size somewhat behind the quartz particles. As an example, Plate II. gives the results obtained by screening and by elutriation of a 1,200 mesh battery pulp.

By Schoene's apparatus, in its ordinary form, a standard of accuracy may be attained which is beyond the ordinary needs of metallurgy. Without sacrificing too much of this standard, the writer modified the apparatus in the following directions, all having the object of rendering its operation quicker and less troublesome.

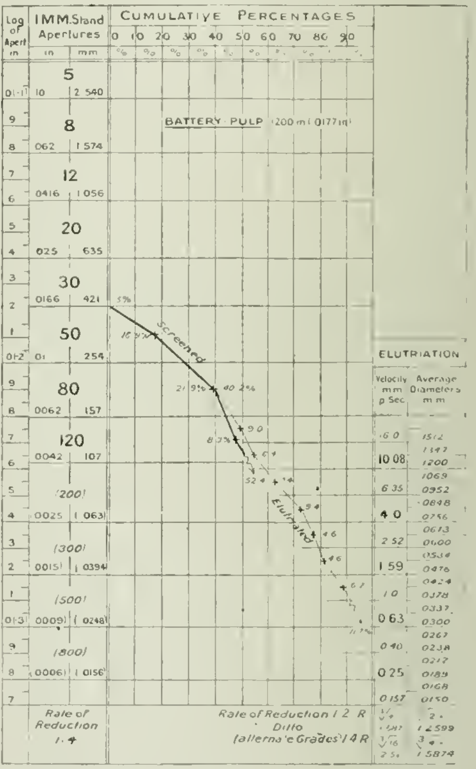
1. By the use of the funnel the sample to be examined can be placed in the classifying tube without removing the stopper, and as the material falls through water the rough classification which takes place prevents the choking of the inflow opening. The upper portion of the tube remaining comparatively clear during the operation of filling, elutriation can at once be started.

2. The setting of the slime, the drawing-off of the water, and the drying and weighing of the overflow products, involves much trouble, which can be avoided by weighing the residues after running them into a glass flask filled to the mark with water.

3. The different openings of the exchangeable overflow nozzles allow the use of the same apparatus over a wide range of grades within reasonable limits.

The method of grading by elutriation has the advantage that the samples need not be dried, as the dry weight can be ascertained by the specific gravity method.* When dry samples are to be elutriated, it is advisable to take their quantity in grammes according to the value obtained by the formula :—

$$gm = \frac{100 \times \text{density}}{\text{density} - 1}$$ (for Rand ore 2·7 density, therefore 158·8 gm.). This amount, if weighed in water, will then show on the scale exactly 100 gm., so that the weights of the residues, also taken in water, give directly the percentages.



With this elutriation apparatus it is easily possible to continue the classification of mill-products as many grades above the 120 I.M.M. mesh as the 30 mesh is below that mesh. Considering that this mesh marks about the limit of accuracy of screening, and that with the present tendency to ultra fine grinding the final pulps are frequently as fine as to leave only about 10% on the so-called 200 mesh, we have of necessity to turn to elutriation to obtain a reliable control over tube-mill discharges, final pulps and slime pulps. The method of grading by elutriation is also much more in accord with practical working conditions, as the pulps on their run through the cyanide plant are exclusively subject to hydraulic handling.

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* This *Journal*, Vol. XII., Nov., 1911, pp. 166-171.

The introduction of this apparatus in the mining industry will greatly advance our knowledge of the physical and metallurgical conditions of the finest mill products including slime. As far as the average Rand ore is concerned, it appears that a still further reduction of sand already passing 120 or 200 I.M.M. mesh is waste of energy, and the determination of the amount of over-ground slime will therefore be a very good check on the efficiency of grinding machines from this economical standpoint. For instance, it would be interesting to ascertain by elutriation how far the higher efficiency of the Nissen stamp can be accounted for by the lower percentage of overdone slime contained in its discharge. The possibilities of this apparatus will have to be ascertained by further experiments, but already it can be seen that it promises to become a most valuable instrument of control in ore dressing.

Theory.—Whilst classification by screening is exclusively dependent upon the size, and to a small extent upon the shape of the particles, many other factors make their influence felt in hydraulic classification. With this method the time of settling and consequently the terminal velocity, in addition to being dependent upon size and shape, are affected also by the density of the mineral, the inner friction of the fluid (viscosity), the temperature (by its effect on viscosity), and by numerous other factors of minor importance. The simultaneous sediments may therefore contain very heterogeneous components, but having taken the same time for settling through a given space, they are said to be of equal *hydraulic value*, and hydraulic classification may therefore be defined as the separation of crushed minerals into groups of equal hydraulic value.

There are two conditions in the settlement of particles which are recognised as distinct from each other, and therefore to be studied independently. R. H. Richards* defines them as follows:—

'*Free settling* is where individual particles *fall freely*, either in still water or against an opposing upward current, without being hindered by other particles. (The classifiers and settling tanks are instances of this principle.)' Settling in a horizontal flow may be considered as a special case of free settling

'*Hindered settling* is where particles of mixed sizes, shapes and gravities *in a crowded mass*, yet free to move among themselves, are sorted in a rising current of water, the velocity of which is much less than the free falling velocity of the particles, but yet enough, so that the particles are in motion. . . . (The jig bed, as well as hindered settling classifiers, are instances of this principle.)'

For the purpose of these notes, only the first case of free settling is to be considered, and here we may distinguish between the two following conditions of settling. *Sedimentation* is based on the difference of the velocities of free fall in still fluids, in which case the falling particles have only to overcome the resistance of the opposing medium, whilst in *elutriation* the particles are in addition subject to the opposing force of the upward rising current, which may retard their fall, keep them in suspension, or even make them overflow.

Accepting the correctness of the axiom laid down by Rittinger, that the velocity of fall of a body in a still fluid is equal to the velocity of an upward rising current sufficient to keep it in dynamic equilibrium, the theoretical part of these notes refers alike to both methods.

In practice a clean separation of the particles according to their hydraulic value is with the method of sedimentation for various reasons unobtainable. On examination it will be found that the lower portion of the sediments, which should consist of particles of the hydraulic value, contains also smaller ones of lower value which, having been nearer to the bottom at the commencement of settlement, had to settle down through a shorter space. Another reason is that the fall of the coarser particles takes the smaller ones with them or at least prevents by the agitation of the water a uniform settling. By a system of successive operations these inconveniences can be reduced to a minimum, but the results will never be so accurate as those obtained by the quicker and easier process of elutriation.

The velocity attained in settlement is governed by two different laws. Small grains settling at a low velocity follow the law of *viscous resistance* (translation) and the larger grains falling with greater velocity obey the law of *eddying resistance* (turbulence). The change from the one law into the other is rather pronounced, as the influence of either law on the other diminishes rapidly with the distance from the critical point or transition stage. The application of the laws is more nearly true—the greater the velocity, in the case of eddying resistance, and the slower the velocity, in the case of viscous resistance.

(1.) *Law of Eddying Resistance (Turbulence Flow).*—From the above-mentioned axiom Rittinger evolves the formula:—

(I.) $V = C\sqrt{D\,(\delta - \delta')}$ For free fall in water—

Of quartz (density 2·65) C = 87

Of galena (density 7·5) C = 100

where V = velocity in mm. per sec.

D = average diameter of grains in mm.

δ = density of the mineral

δ' = specific gravity of the fluid (for water at 20 C = 1)

C = a constant

Isaac Newton arrived at the same formula from the assumption that the resistance due to eddying motion set up in the fluid may be expected to vary as the square of the velocity.

The general application of this formula is limited, as the constant C includes a factor expressing the viscosity, which increases with the size of the particles in such rapid progression that the falling velocity of the smallest particles practically vanishes. The formula holds good therefore only if applied to grains above a critical size when skin friction and viscous resistance becomes negligible in constrast to the increasingly important part played by the eddying resistance.

For any given material $\delta - \delta'$ (or $\delta - 1$ for water) is a known quantity which may be merged in the constant c, and with this new constant, c, Rittinger's formula becomes—

(II.) $V = c\sqrt{D}$ For free fall in water—

Of quartz (density (2·65) c = 112

Of galena (density (7·5) c = 255

or expressed in words: *The velocity of fall of roundish grains above a critical size, falling at such a velocity that the law of eddying resistance is alone followed, is a constant multiple of the square root of the average diameter of the grains.*

(2.) *Law of Viscous Resistance (translation).*—For the case of small roundish bodies, settling at such slow velocities that the eddying resistance can be neglected, Stokes, from purely theoretical considerations, deduced for the terminal velocity V, the following formula:—

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* "Ore Dressing," by R. H. Richards.

$$V = \frac{2}{9} g \left(\frac{\delta - \delta^1}{\eta} \right) r^2$$

where V = velocity of fall in cm. per sec.
r = radius of the sphere
g = acceleration due to gravity
n = the co-efficient of viscosity for water at 20° C = 0·01).

If V and D are taken in mm. instead of V and r in cm. the formula becomes, by substituting known quantities by the constant K :—

(III.) $V = K(\delta - 1)D^2$ For fall in water—
Of quartz (density 2·65) K = 424
Of galena (density 7·5) K = 631

It must be noted that the constant K, involving n (brought into account at the value of 0·01 for water at 20° C.), changes about 2% per degree for temperatures differing from 20° C.

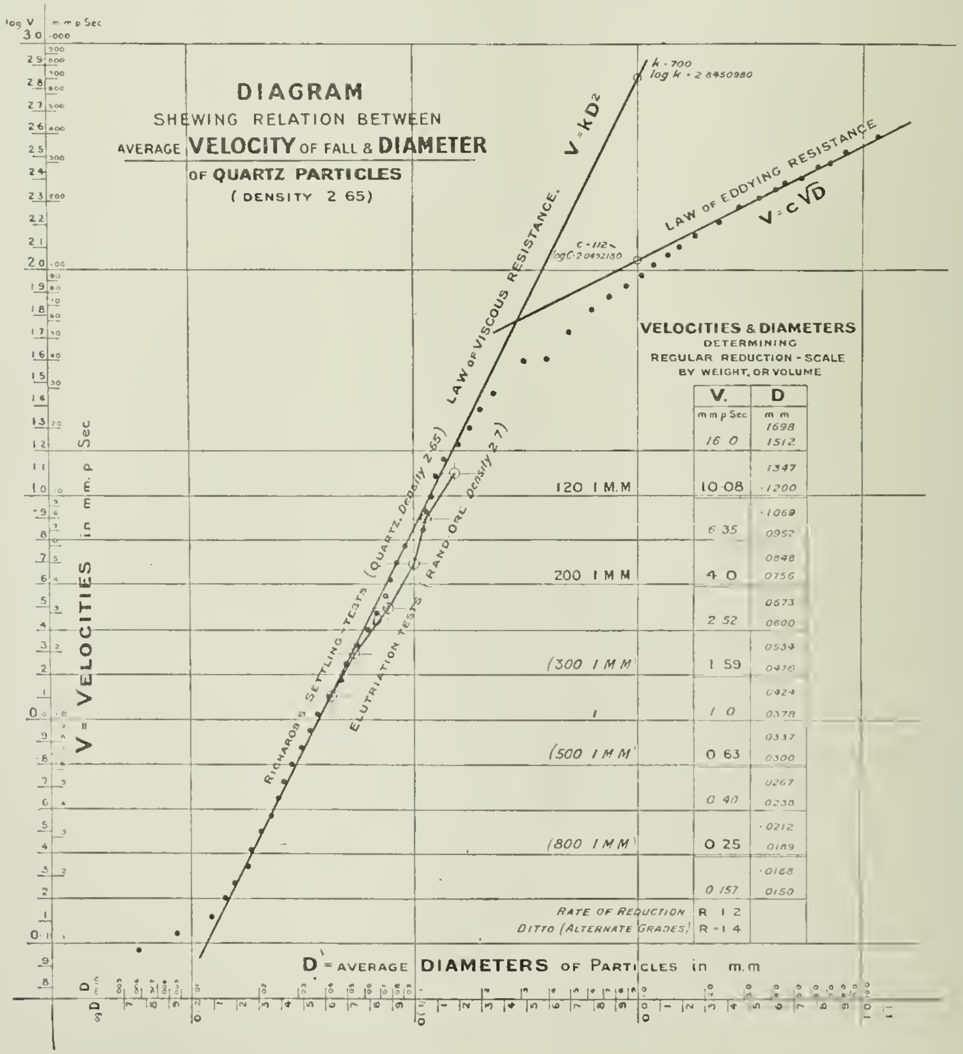
For any given material $\delta - \delta^1$ (or $\delta - 1$ for water) is constant and may be merged in the constant k. By using this new constant, k, the formula becomes :—

(IV.) $V = kD^2$ For free settling in water—
Of quartz (density 2 65) k = 700.
Of galena (density 7·5) k = 4100.

or expressed in words :—

The velocity of roundish particles below a critical size, settling at a slow velocity at which the law of viscous resistance is alone followed, is a constant multiple of the square of the average diameter of the particle.

Since the constants C and K in the general formula (I) of the eddying resistance and (III) of the viscous resistance do not include a factor involving the density of the material, these formulæ should theoretically hold good for all materials of any density. These values are, however, slightly affected by the shape of the falling bodies, and it is therefore not surprising that the values for C and K differ for two materials of such a widely different nature as quartz and galena. They lie, however, on either side of the theoretical value of 550. As it is not likely that a simple formula may be found covering both laws of



VELOCITIES & DIAMETERS
DETERMINING
REGULAR REDUCTION - SCALE
BY WEIGHT, OR VOLUME

	V.	D
	m m p Sec	m m
		1698
	16 0	1512
		1347
120 I M.M	10 08	·1200
		·1069
	6 35	0952
		0848
200 I M M	4 0	0756
		0673
	2 52	0600
		0534
(300 I M M)	1 59	0476
		0424
I	1 0	0378
		0337
(500 I M M)	0 63	0300
		0267
	0 40	0238
		·0212
(800 I M M)	0 25	0189
		·0168
	0 157	0150
RATE OF REDUCTION	R 1 2	
DITTO (ALTERNATE GRADES)	R = 1 4	

resistance and providing for the slightly varying effect of the surface friction on the acceleration, and therefore on the constants, the difficulty must be overcome for practical purposes by determining the constants empirically for different materials as it has been been done by Richards for quartz and galena.

That these laws of settling are closely followed by the actual results of Richards' experimental series is clearly illustrated by the accompanying diagram (Table III.) in which the abscissæ are the logarithms of the average diameters and the ordinates the logarithms of the velocities.

With this diagram before us the conclusions may be summarized as follows :

1. The curves for quartz and galena (this latter not plotted) are exactly parallel, proving that the laws hold good for both minerals of widely differing density (2·65 and 7·5 respectively) ;

2. The constants c and k are affected by the density. They are for free fall of quartz in water as follows :—

Law of eddying resistance (turbulence).—$c = 112$.
Law of viscous resistance (translation).—$k = 700$.

By taking the logarithms in formulæ (II.) and (IV.) it follows that the logarithm of the velocity increases in the case of eddying resistance one-half as rapidly as the logarithm of the diameter, and in the case of viscous resistance twice as rapidly. The slope of the first curve is therefore 1 : 2 and of the second 2 : 1. The constants are determined by the interception of the straight curves with the ordinate at the point where log. $D = O$.

3. The logarithmic plots of Richards' series of tests determine evidently a hyperbolic curve—yet at a reasonable distance on either side beyond the critical point or transition stage, the arms of the curve have flattened out to such an extent that they can without great error be represented by straight lines over a fairly wide range. In the case of slow-settling particles, the straight line representing the law of viscous resistance, according to the formula for quartz, $V = 700\ D^2$, covers, with a quite sufficient degree of accuracy for all practical purposes, a range of velocities from about 16 mm. per sec. (0·15 diam.) down to about 0·157 mm. per sec. (0·015 mm. diam.).

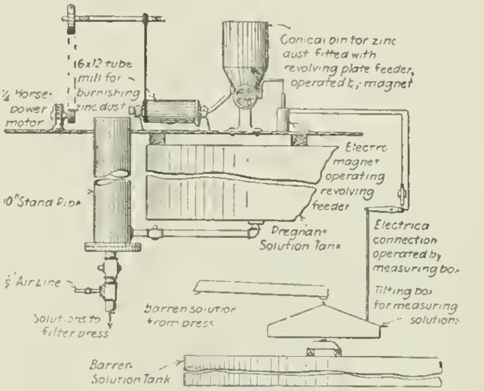
4. With such a simple law, the velocities required for classifying finely-ground minerals into a series of groups determining a regular reduction scale in volume or weight, at the ratio 1 : 2, can easily be taken from the diagram or calculated, and the values for V in Table (I) are those obtained by taking the velocity of 1 mm. per sec. as starting point.

5. The curve of a series of elutriation tests carried out with Rand ore by the author with a primitive laboratory equipment at his disposal coincides remarkably well with the curve of Richards' series of settling tests, confirming the correctness of Rittinger's assumption that the terminal velocity of falling grains in fluids is equal to the velocity of an upward rising current sufficient to keep the particles in suspension. The comparatively small deviations may easily be accounted for by the different shape and nature of the minerals under examination in the two tests, by unequal temperatures, purity of the water (in the elutriation tests mill water had to be used), and especially by the difficulty of eliminating the personal factor in guessing under the microscope a representative average diameter from a few particles out of millions.

(It is difficult to understand why the last two observations of Richards' series should deviate from the law for any other reason than that of experimental errors which at such slow velocities may easily be due to changes of temperature affecting the viscosity of the fluid.)

6. The limit of accuracy is without special precautions apparently reached at a velocity of about 0·15 mm. per sec., but for all mining purposes this limit is already far beyond our most pretentious requirements." -H. Stadler, *Bulletin, Institution of Mining and Metallurgy*, May, 1913. (A. R.)

Zinc-Dust Feeder. — "The solution having settled is admitted into a 10 in. standpipe beside the clarifying vats, over which is placed a 6 in. × 12 in. tube mill, through which the zinc dust is fed automatically into the standpipe by means of a magnet connected with the tripping bucket at the zinc press in the precipitation room below. As the bucket is filled with discharge from the press, an electrical connection is made with the magnet at the zinc mill, and it in turn operates the automatic feeder which lets the required amount of zinc dust enter the mill, where it is mixed, burnished and discharged into the standpipe. This arrangement is shown in figure."—H. A. and R. T Sill,—*Engineering and Mining Journal*, April 12, 1913, p. 745. (H. A. W.)


Conical bin for zinc dust fitted with revolving plate feeder, operated by magnet
16x12 tube mill for burnishing zinc dust
¼ Horse-power motor
Electric magnet operating revolving feeder
10" Stand Pipe
Pregnant Solution Tank
Electrical connection operated by measuring box
½" Air Line
Barren solution from press
Tilting box for measuring solutions
Solutions to filter press
Barren Solution Tank

Zinc-Feeding Device at Zombona Mill.

The Pinder-Berry Stamp Mill.—"In an effort to improve upon current stamp mill construction, a type of stamp has been developed that involves such differences from the usual form as to warrant, perhaps, special description. It is the outgrowth especially of the experience of J. W. Pinder, and the first stamp of this type was built and placed in operation in Shasta county, California. Details as to its work are given below, following a general description of the stamp as now built.

The mortar has a base of standard thickness and width, is circular, with the screen around the circumference and equally distant from the stamp, the front upper part of the body cast with the base, the back upper part removable and secured by toggle clamps for easy handling. The screen is in two pieces, rectangular in shape, permitting reversal when worn at the bottom, and held by rubber-lined steel bands secured by toggle clamps. Screen changing is easily and quickly done, and the entire mortar can be opened for changing shoes and dies, and for cleaning out. The feed water is introduced, under pressure, at six points around the bottom of the mortar, in order to keep the material loosened to facilitate the rapid settlement of free gold through the mass. When quicksilver is used, it will be protected from being splashed about by the stamp and lost.

The concrete foundation shown is reinforced, and is provided with a tunnel to allow access to the lower ends of the bolts. Cast-iron sole plates are used under the battery posts, and the posts are fastened directly to the foundation by two heavy rods.

Boss-head and shoes are proved with holes forming finally a central channel to deliver the ore to the centre of the die, resulting in a maximum crushing capacity. This feature, combined with the long screen and increased number of drops, permits a heavy duty, as has been proved by test.

The guide shown is iron, bored to fit the well finished stem, thus reducing vibration. It is adjustable for alignment with the stem in both directions, has no nuts or threads in contact with it, and is split so that the stem can be taken out by slackening the nuts on the back side of the girth and turning the bolts part way round.

With the present form of tappet, held on to the stem by friction, the practical limit in the number of drops per minute has been about reached. This is due partly to the difficulty of preventing slipping of the tappet and partly to the violence of the blow struck by the cam. With the construction employed, a broad, shallow thread on the stem, a sectional tappet provided with clamping keys, and a simple, practical cushion between a cast-steel wearing plate and the body of the tappet, this limit has been greatly extended. It is now feasable to make as many as 138 six-inch drops per minute, by proper cam design. The question of cam design for any reasonable number of drops is a simple one, no difficulty at all being found with the above number.

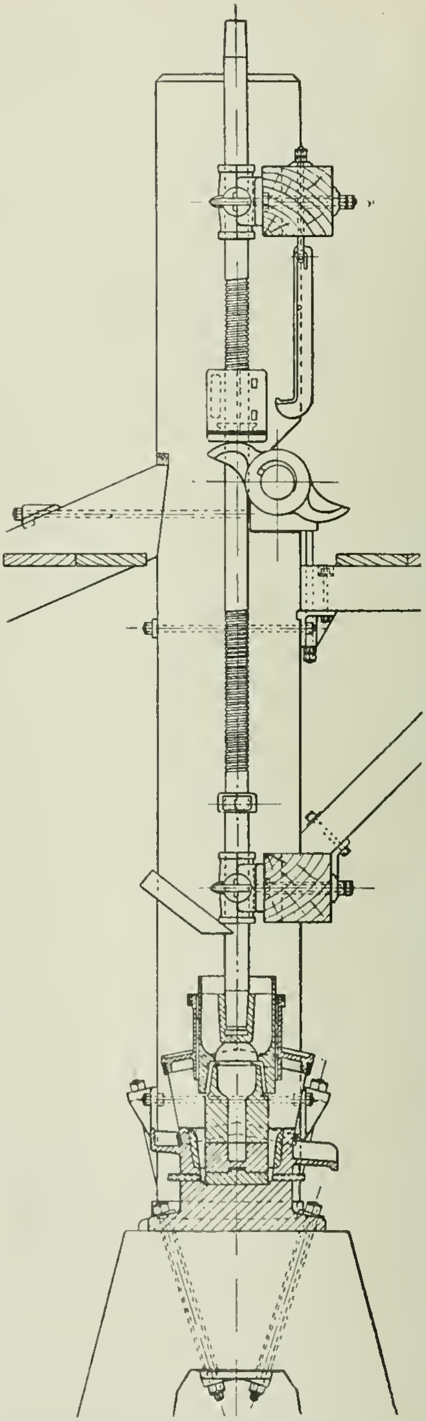
With this mill there may be used any kind of cam desired, the standard, the noiseless, or the high-speed, the selection depending upon the results desired. The threaded tappet would be used with all styles, the cushion feature with the high-speed, and with the standard if quieter operation is desired. The advantages of the threaded tappet are found in the greater ease of adjustment and the greater security against slippage. Two taper keys are used, and, as the frictional element has been removed, these do not require to be set up with anything like the same degree as with the old-style tappet. Adjustment for wear consists in slackening the keys with a machinist's hammer, turning the tappet on the stem, or the stem in the tappet, and keying up. No sledging is required.

In general, the battery frame is of the size and construction used for a five-stamp battery, but an improved method of bracing it has been worked out, leaving it free from the influence of a settling ore bin. Two 1,350 lb. stamps can be placed in a frame having the posts 60 in. apart.

In mills large enough to warrant it, it is proposed to use a short crane having a span of about 8 ft., and available for handling cam-shafts, pulleys, and stamp parts. It is the intention to change shoes and dies, and other stamp parts, from the floor behind the battery, leaving the front entirely clear for plates or other machinery. For this purpose the feeders would be moved back and part of the upper floor, which is left loose, removed.

Advantages of this Type.—The advantages of this type of construction are:—

I. The feed is, at all times, even and regular, being distributed instantly under every part of the crushing surface of the shoe, giving the greatest possible duty to the stamp at each blow and in-



Pinder-Berry Stamp.

creasing the crushing capacity to a remarkable extent.

2. The wear of shoes and dies is comparatively true and even.

3. There are no flying pieces of ore to cut and break the screens. This is, at times, a serious detriment to the standard stamp, where ore is fed to the outside of the die, the larger pieces struck, but not crushed, by the shoe, fly with great force against the screen.

4. The feed being central and even, there is no wrenching strain on the shank of the stem to cause it to crystallize and break, as so often occurs on back-fed or solid stamps.

5. The screen length being so much greater than on the standard battery, also greatly increases the capacity of the stamp by permitting immediate discharge of the pulp as fast as crushed, avoiding unnecessary sliming.

6. The injection of the feed water into the bottom of the mortar, as described, permits the collection of the free gold without amalgamation. The forced injection of the water keeps the pulp soft and agitated, and prevents packing, while the liberated gold will immediately go to the bottom without the aid of quicksilver. But quicksilver may be used if desired, and to much better advantage than in the standard mortar where it is constantly splashed about from under the stamp and much of it lost. No chuck-blocks or inside plates are required.

Recognising the great value of injecting feed water into the bottom of the mortar for the above reason, many efforts have been made from time to time by prominent metallurgists to accomplish this object, but without success. The failure was due to the comparatively small discharge capacity of the standard mortar, and the limited amount of water that could be used for that purpose did not serve to prevent the solid packing of the pulp before it could be discharged, the injection of so small a quantity of water into the comparatively large box area having but little effect. The angular form of the mortar was also a preventative, the packing, beginning to form in the corners and between the dies, soon blocked the inlets.

The comparatively small size and weight of the Pinder-Berry mortar offers a great advantage in economy of hauling and transportation.

A Comparative Test.—The following is the comparative result of a test run, of 23 days, of the Pinder-Berry and standard stamp, in a Shasta county mill, September, 1909.

The original mill was provided with ten stamps. After preliminary experiments, Mr. Pinder had a mortar, boss-head, cam, shoe, and die cast in Redding. The cam-shaft was long enough to allow the new cam to be put on the overhanging end by moving it endwise as much as possible. Five stamps and the single one were run 21 days at about 109 drops per minute, the five stamps crushing 5·2 tons per stamp and the single one 11·5 to 12 tons. For two days the five were hung up and the single run at about 138 drops, crushing 16·2 tons. The ore was measured by cars and was hand-fed to the single mill. During the run the battery water was fed in at the usual place in both mills, the experiments with the water entering at the bottom following later. The stem was not threaded, but the tappet was held by set-screws in addition to the usual keys. A cushion was used, consisting of rubber packing protected by boiler plate ¼ in. thick. Details of the run follow :—

	Pinder-Berry.	Standard.
Weight of stamp, lb. ...	1348·000	1000·000
Drops per minute	138·300	109·600
Horse-power per stamp ...	2·650	1·750
Ore crushed per stamp, tons	16·200	5·200
Ore crushed per horse-power, tons	5·890	2·970
Horse-power per ton of ore crushed	0·169	0·336

Screen, 30 mesh steel wire, drop 6 in. ; ore, black slate with quartz stringers.

Actual running time: Pinder-Berry, 22 days, 23 hr., 40 min.; standard, 20 days, 19 hr., 35 min. The standard was hung up 53 hr. 25 min. for screens, tappets, and clean-outs. The Pinder-Berry was hung up 20 min. on the 15th day for clean-out, and to adjust the tappet for shoe and die wear. Since the foregoing test the mortar and stamp details have been remodeled and greatly improved.

In addition to the greatly increased tonnage of this stamp, the important saving in lost time, as compared with the standard, on account of slipping tappets, breaking screens and stems, may be placed at 15%, and in cost of labour and attention at 50% in favour of the Pinder-Berry. —S. L. BERRY, *Mining and Scientific Press*, May 17, 1913, p. 739. (A. McA. J.)

THE CRUSHING-SURFACE DIAGRAM.—"*Synopsis.* — An effort to reduce to graphic form the exposition of the energy absorbed in reducing rock from one size to a smaller one. The diagram is based on Rittinger's law, which is believed to be more applicable to the problem than that of Kick. The diagram presents a convenient means of comparing the crushing efficiences of various machines, a system which will be valuable in many branches of metallurgy. The conclusions are contrary to those found and expressed by other investigators.

In view of the importance of crushing efficiencies in various industries, and the frequent discussion taking place on this subject, the diagram which may be called the crushing-surface diagram is submitted as a means of comparison and study of crushing conditions. The scheme was discovered during the summer of 1910, when trying to understand why one tube mill did not do the work expected of it on the basis of what another was doing on 'a similar ore just as hard.'

Diagram Based on Rittinger's Law.—First it should be stated that this diagram is based on Rittinger's law, which is interpreted to mean that the work done in crushing is proportional to the surface exposed by the operation, or better expressed for this purpose,

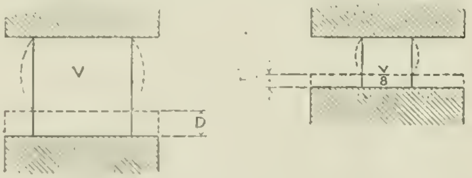


Fig. 1.—Kick's Law.

100 Mesh Cube — 200 Mesh Cube

Area one face = A — Area one face = $\frac{A}{4}$

Average resisting force per sq. in. = F

Energy = FAD — Energy = $F\ \frac{A}{4}\ \frac{D}{2}\qquad\frac{FAD}{8}$

∴ Energy Proportional to volume.

the work done on a given mass of rock is proportional to the reciprocal of the diameter of the final product, assuming that all the mass has been reduced to one exact size, which is only theoretically possible.

Kick's law is frequently referred to in connection with this subject, especially since the publication of H. Stadler's work, and is thus expressed: 'The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies.' This law does not apply as much to crushing as to deformation of bodies before rupture takes place.

Kick's Law Not Applicable.—In Fig. 1 are represented two particles of ore 'of equal technological state' shown as cubes between the faces of a crushing or testing machine. Assuming the theoretical mesh, equivalent to the reciprocal of the diameter, and using concrete values, we have a 100 mesh particle with eight times the volume of the 200 mesh particle, and with an area per face four times that of the 100 mesh particle. The dimensions are as 2 to 1, and the bodies being similarly deformed within the elastic limit without fracture, the energy that must be applied in each case to produce this deformation is the product of the average resisting force per square inch, the same in both cases, into the area worked against and into the distance through which this average force works. As shown in the figure, in this particular instance the energy absorbed is proportional to volume and it can be similarly shown for the general case. On the gradual release of the external pressure the energy absorbed is given back to the machine producing the deformation and the body returns to its original shape. It should be noted that the body has been deformed only by a gradually increasing pressure, the first increment of deformation not requiring so much pressure as the last. In case the body has been deformed beyond its elastic limit either the whole mass of particles have been reduced to the molecular state by the freeing of their bonds with adjacent particles, which never happened, or fracture takes place along a few surfaces by the breaking down of some of the weaker bonds, and the new particles thus formed are free to resume their original shape in so far as they are not held between the machine surfaces. The energy given up by them is probably used in some sort of lever action in making fracture planes. So the energy absorbed according to Kick's law does not stay in the particle after pressure is released and therefore this law does not govern to any great extent the amount of energy absorbed in crushing.

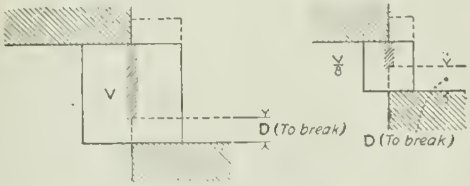


Fig. 2.—Rittinger's Law.

100 Mesh Cube	200 Mesh Cube
Area one section = A	Area one section = $\frac{A}{4}$
Average resistance to shear per sq. in = F	
Energy = FAD	Energy = F $\frac{A}{4}$ D
Surface produced = 2 A	Surface produced = 2 $\frac{A}{4}$

Energy Proportional to Surface.

Energy Proportional to Surface Produced.—Reference to Fig. 2 will explain how Rittinger's law applies. Supposing that it were possible to hold similar cubes between the two offset faces as shown, and that forces were applied until the deformation shown by the dotted lines was obtained, it will be seen that only the molecules along the vertical centre line are stressed and deformed, the mass of the cube away from this surface receiving practically no pressure or deformation. The energy in this case required to produce rupture will be the product of the average resistance to shearing per square inch by the area along which rupture takes place, and by the distance the two offset faces move together. The average resistance to shearing is a variable quantity, as the deformation increases up to rupture. To reduce to cubes, this amount of energy must be multiplied by three, as three similar fracture planes must be made to produce cubes. And it will not be hard to see that the distance through which the offset faces must move in either case must be the same and not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement regardless of the thickness of the piece.

It will be noted that if eight of the half-diameter cubes be sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared into half-size cubes, and also that the energy required in the case of the smaller cubes is double that required in the case of the larger cubes. This should demonstrate that energy applied to crushing is proportional to the surface produced. ∝

Surface Proportional to Reciprocal of Diameters.—In this scheme we start from an infinite mass, say the side of a mountain, and consider that particles are sheared off in the form of cubes, each particle having new surface added to it equal to that exposed before the new shearing operation was started. This eliminates the 'minus one' part of the equation given in Richards' 'Ore Dressing' on the subject. For example, if we cut off from a cubical corner of this infinite mass 100 slabs 1 in. thick and 100 in. sq., and then cut the slabs up into 1 in. cubes, 6,000,000 sq. in. of new surface is produced. If instead, 200 ½ in. slabs were cut from the same corner and likewise cut up into ½ in. cubes, the area would be 12,000,000 sq. in., just double the surface of the 1 in. cubes. This shows that the surface produced on equal masses of rock is proportional to the reciprocal of the diameters, as can be proved in a similar manner for any sizes. Also the surfaces of two lots of the same sized particles will be proportional to their values, and if of the same substances, proportional to the weights of the lots.

Energy May be Represented by an Area.—From the above we see that the energy absorbed by a lot of given sized particles is proportional to the product of their surface by their weight, which can be shown graphically by an area. And if we have a series of these areas, placed side by side, and representing the summation of the energies of the different sizes produced by a crushing operation, the total area is proportional to the energy expended on the rock. And when the cumulative weights of the different sizes are plotted consecutively, as in Fig. 3, the area between the sizing-analysis curve and the zero lines is proportional to the work expended on the rock in breaking it down from the infinite mass. And then if further crushing takes place on all or part of this rock and the new sizing analysis is plotted, the area between the two curves is proportional to the further work done. This is the crushing-surface diagram.

For rough work, comparing the work of one day with that of another, the screen analysis may be plotted, mesh against the percentages and areas compared, etc. For accurate work, such as the determination of constants for different rocks, or for the comparison of the efficiencies of different crushing machines, etc., the actual reciprocals of the diameters should be plotted. In fact, it is no hard matter to locate on the diagram the reciprocals of the diameters for the screens used; or better still, screen manufacturers could be sufficiently induced to furnish pads of coordinate paper, on which are drawn these lines corresponding with their standard sets of screens.

Several Uses for the Diagram.—Two uses are already suggested; the first, determination of crushing resistances of various rocks, and second, the determination of crushing-machine efficiency. Only with a knowledge of the former, can comparisons be made of the operation of two machines at different places. And I believe that this diagram will make necessary some revision of the constants for the crushing resistance of rocks. All tests of rocks have been made on single pieces and the crushing strength determined from an average of a number of tests. Where surface has been determined it is only that of the larger pieces involved in the fracture. No calculations based on the surface of the very fine particles have been made. One thing which will be noticed in nearly every screen analysis that is plotted as a crushing-surface diagram, that the curve is of the form of a hyperbola, indicating a strong probability of the presence of particles of 1000 and even 10,000 mesh. Some allowance for the probable area of these very fine particles must be made, and when that is done it will very likely be found that there is a greater area in the minus 200 mesh particles than in the plus 200 mesh material. There is no question but that the surface of the minus 200 mesh particles must be considered in crushing calculations. The neglect of this surface, which usually is not wanted, makes any reasonable comparison impossible, between crushing operations that are not very much alike. The agriculturists in the study of soils do not hesitate to measure and discuss particles of clay as small as 0·0002 in. dia., and so I see no reason why milling operators should avoid these sizes and talk of colloids as soon as they get past the 200 mesh line. The use of logarithmic paper can be invoked to find out about the minus 200 mesh particles as I may show later.

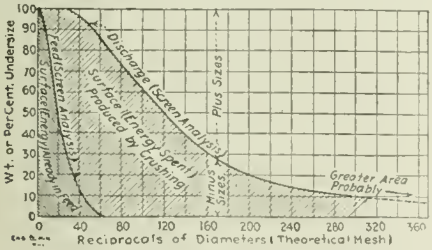


Fig. 3.—The Crushing-Surface Diagram.

Important Use in the Cyanide Plant.—Other uses for this diagram should be in the cyanide plant, where time of dissolution depends upon surface exposed, and the amount of surface is a factor in filtering operations. In the cement industry, surface must play an important part, for each particle of stone or sand or fine material must be coated with cement particles before a good bond can be made. The amount of cement could be made proportional to the area of the diagram for the material used.

In concentrating-mill operations, surface is fully as important in the concentrating department as in crushing operations. It is surface that limits the tonnage that can be put through a given machine, only a certain number of layers of particles can be treated at one time on a given surface, which explains the small capacity of fine concentrating machines. Based on surface, the distribution of feed to different machines can be made more efficiently. Also the character of the crushing operations can be modified to give the largest reduction possible with the minimum production of useless fine surface.

The first experiments to prove some of the facts in the case were made in Salt Lake City, in 1910, when I used a small gravity stamp of 22 lb weight, but the foundation was not satisfactory and other conditions compelled the dropping of the work which was only taken up again when I came to Purdue University a year ago. Here I found in the laboratory for testing materials, a hydraulic testing machine capable of exerting and recording pressures up to 150,000 lb. which just suited this class of work.

Diagram of Tests Made.—The two diagrams, Figs. 4 and 5, are made from the logsheets of this test, all records of crushing pressures being plotted directly on the sheet against the movement of the faces of the machine. The triangular curved areas are the records of the work on the machine, and are a measurement of energy. Each crushing movement was continued until the pressure was rapidly going up with little movement of the faces, then the pressure was gradually reduced and we got the return curve showing that some energy was being given up by the rock. The machine was then opened, the finer sizes screened out and the coarse returned and crushed again. After the crushing was complete as far as was considered advisable, the screen analysis was made and plotted as the weights were determined. One hundred grams was used in each case, its original screen analysis being the first thing plotted on the paper.

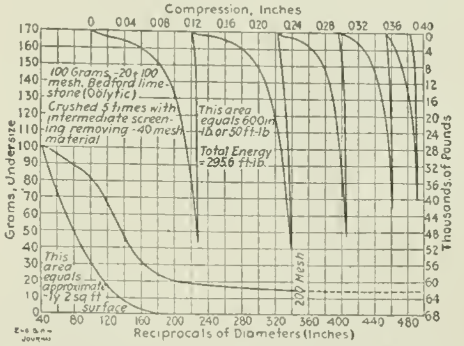


Fig. 4.—Diagram of Tests on Testing Machine.

Without going into the calculations in detail, the energy absorbed in the case of Fig. 4 was 166·3 ft.-lb., while in the case of Fig. 5, the energy absorbed was 295·6 ft.-lb., about 80% more. And yet the area of the surface under 200 mesh was greater in the case of the former. The only way to account for the energy is to go beyond the 200 mesh line, and doing this, using logarithmic paper and assuming that the curve continues its law indefinitely among the smaller particles, I find that out to 5,000 mesh for Fig. 4 the energy is equal, per unit area, to that in Fig. 5, out to 11,000 mesh. Both are calculated to have one-half of 1% at those sizes.

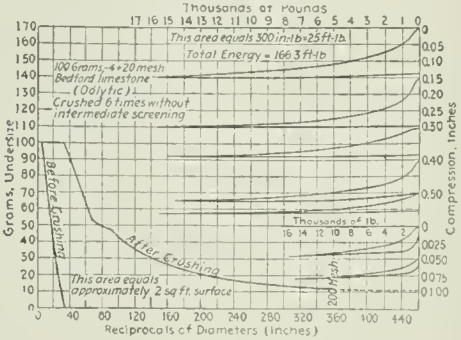


Fig. 5.—Diagram of Tests on Testing Machine.

I do not see any other way of accounting for this energy, but it is so revolutionary as to make a serious study necessary before it will be universally adopted. It ought to be definitely known whether Rittinger's or Kick's law, or either of them, applies to the crushing problem in mill work, and if I have started a discussion which will produce this result, I shall be satisfied."

[Fig. 3 shows a curve much more like the well known 'frequency curve' than a hyperbola. Editorial Committee.]—A. O. GATES, *Engineering and Mining Journal*, May 24, 1913, p. 1039. (H. A. W.)

ELECTROLYTIC METHODS FOR PREVENTING THE CORROSION OF METALS.—"The corrosion of metals can be lessened or prevented in two ways: (1) By connecting the metal to be protected to a more electro positive metal, so that a primary cell is produced; (2) by making the metal to be protected the cathode in an electrolytic cell supplied by an external electrical pressure. Various types of primary cell arrangements that could be employed in practice were classified. The efficiency of the cell for protection will depend on the current density at the cathode, and this will be controlled by the resistance of the cell and the effective voltage. The importance of overvoltage in determining the effective voltage was discussed. The history of Sir Humphry Davy's application of zinc and iron protection for the prevention of the corrosion of the copper sheathing of ships, and subsequent inventions for the protection of condensers and pipes were detailed. The patents of Harris and Anderson, in which aluminium alloys are used for the prevention of the corrosion of condenser tubes, were shown to be primary cell methods. The use of zinc in boilers involves a knowledge of the electrolytic resistance of the boiler waters and the effective voltage at temperatures from 150° C.—200° C., concerning which there is great need of experimental data. The amount of zinc used in some marine boilers is as great as from 400—600 lb. of rolled zinc per annum. If the zinc is efficient in producing electrical currents then the average current may be from 17 to 25 amperes. It is obvious that such currents would be obtained much more economically by the use of a dynamo. The direct use of electrical currents has been the basis of a number of patents. Those of Mr. Elliott Cumberland were especially described. Iron anodes are placed in the water of the boiler, which latter is made the cathode. A low voltage of supply provided by a small motor-generator is used. The method has proved effective not only in preventing corrosion, but also in removing scale from the heating surface and preventing its formation. Experiments carried out at the Manchester School of Technology have shown that the current densities necessary to protect iron, copper, and other metals from the corrosion of fresh and salt water are of low value, and hence in the case of boilers and condensers the annual cost of the electrical energy required is a small item. The chief cost will be in the renewal of anodes. Harris and Anderson have also applied electrical currents for the prevention of the corrosion of condensers. They find that a condenser with a cooling surface of 1,025 sq. ft. requires only 2 volts and 2 amperes, and the special anodes used by them cost from £3 5s. per 1,000 sq. ft. per annum. The use of electrical currents may also be applied in chemical works to prevent the corrosion of metallic screens and vessels from acid liquids."—PROF. W. W. HALDANE GEE, *Journal of the Society of Chemical Industry*, May 31, 1913, p. 520. (F. W. W.)

LOSSES IN TIN DRESSING.—"At the meeting of the Association of Cornish Mining, Mechanical, and Metallurgical Engineers, held on April 5, William Thomas read a paper discussing the losses incident in the dressing of Cornish tin ores. After reminding his hearers that the introduction of the chemical assay was not the occasion when these losses were first disclosed, but that the assay only confirmed in a scientific manner a matter which was of common knowledge, he proceeded to refer to the disinclination of most mine managers to investigate the causes and remedies. The average manager in the old days, and even now, did not care about sampling his tailing. He was keener on mixing it with tailing from other mines as soon as possible and so sharing his burden with his neighbour, by discharging them into the streams, or better still, by sending them over the cliffs. However, systematic sampling of tailing had been introduced at some mines of late years.

In addition to sand and slime losses and losses after roasting, there were other losses to be reckoned, which in mines handling complex ores were considerable. Some of these losses had been substantially reduced of late years For example, not long ago it was not unusual for a parcel of wolfram, as sold, to carry 15% to 20% of black tin. This has now been reduced generally to 2% or 3%. Black tin as sold a few years ago from three or four of the leading mines carried from 6% to 9% of tungstic acid, and such parcels of tin were heavily penalized by the smelters. Mr. Thomas quoted correspondence and notes relating to endeavours made, on the initiative of the late professors Le Neve Foster and Bauerman, to introduce the Wetherill magnetic separator into Cornwall in 1898. In September of that year, the lecturer called on East Pool and other mines with Alfred Martin, of Frankfurt, who then represented the Wetherill patents in Europe. The efforts were unavailing as far as Cornwall was concerned, but Mr. Thomas exchanged ideas with Lake and Currie in 1899, and they, with Joseph Richards as manager, erected on their San Finx mines, in Spain, the first Wetherill machines applied commercially to tin and wolfram ores. Complete success was obtained, and the San Finx example was promptly followed at the Clitters mines in east Cornwall.

If all sources of loss were taken into account and due allowance made for inaccurate weighing of the ore, the average loss of tin in Cornish mines at the present time was not less than 40%. He did not know a Cornish mine where the loss was less than 30%. He knew of some where it exceeded 40%, and in two cases he had investigated it approached 60%.

Mr. Thomas then proceeded to indicate lines along which improvements might be made in the treatment of tin ores in Cornwall.

(1). By avoiding unnecessarily fine crushing. This involved substituting a coarser screen in the stamps, and adopting stage crushing. The first tube-mill ever inserted as a permanent part of an automatic plant in Cornwall was that put in by Mr. Thomas at King Edward mine about 10 years ago. One of his successors removed it; but at South Crofty, East Pool, and several other mines tube-mills now formed a permanent and efficient feature in the plant. The use of tube-mills at this stage of crushing necessitated the selection of a suitable concentrator immediately below the stamps. He had strong reasons for declining to favour any concentrator below the stamps that could not be made to yield one finished product at the outset, that is to say, valueless tailing. There were concentrators in plenty on the market that, on most Cornish tins ores crushed through a 10 to 15 mesh, would yield a considerable portion of coarse tailing running under 5 lb. It was a great advantage to get rid of this at once No profit could be made out of it. If ground to slime, it ran to waste at 10 lb. instead of 5 lb., stealing tin from the richer portions of the ores.

H. W. Hutchin proved with his heavy solutions that certain portions of the matrix carried practically no tin. This entirely confirmed his own observations on lode-structure in general, and the distribution of tin throughout the lodes. As a rule, the distribution was not uniform. Tin was confined, in the main, to leaders, branches or parts of the lodes, while other parts were practically barren. If in a granite capel, the tin was not in the quartz, but in the altered felspar. It was essential to select a concentrator that should discard a substantial portion of those barren grains, in as large a size as possible; thus following up, automatically, the hand selection begun on the spalling floors or the picking belts; and for precisely the same reasons.

It was notable that, following the lead of South Crofty, several other mines in various parts of the country had abandoned the 20, 25, and 30 mesh screens, or their misleading equivalents in the form of punched grates, in favour of 10, 12, and 15 mesh screens. In every instance, satisfactory results had followed the change. There was a limit, of course, to the range of coarser crushing, and in the author's opinion the limit was about 10 mesh.

(2). It was time to substitute automatic plant for the hand-treatment of roasted ores. Automatically-fed machines, whether grinders or concentrators, would handle more stuff and maintain a more uniform set of conditions. Experiments in this section were enforced at King Edward mine in the early days, because no other plant existed. They were entirely successful. The first mine to select such a plant on commercial lines was Stormsdown, in Devonshire, by Ernest Terrell, followed just after by Josiah Paull, at South Crofty, with considerable elaboration. Stage-crushing formed a main feature in that plant.

(3). The roasting furnaces in use in Cornwall were defective for complex ores. The Brunton furnace was not sufficiently supplied with oxygen when treating such ores carrying a high percentage of sulphides. Those in charge of the furnaces did not always distinguish between temperature and combustion; and, finding something going wrong, were apt to 'stoke up,' and so commit a folly in innocence. Another serious defect in the Brunton was its inefficient stirring gear. Portions of the charge were never in actual contact with the heated air. It was not surprising that arsenic stayed in the charge and gave trouble to the smelter.

Temperatures were, as a rule, raised much higher than they need be in Brunton furnaces in Cornwall, and he strongly suspected that the consequent evils extended farther than he had intimated.

(4). Classification at the outset was essential. It seemed late in the day to refer to this first law of ore-dressing. It had become fairly general in Cornish mines, but a few still maintained the old-fashioned disregard of this essential.

(5). An investigation of the burnt leavings question was important. In this matter those who were attempting to work out a commercial method of chemical extraction had, in his opinion, excellent opportunities for a start, much better opportunities than the raw ore afforded."—*Mining Magazine*, May 1913, p. 376. (A. R.)

INFLUENCE OF SULPHUR ON CAST IRON.—"A series of test bars was prepared from pure materials and was examined chemically, physically, and metallographically. The results show that: (*a*) With low silicon content a small amount of sulphur causes nearly all the carbon to become combined. With high silicon content the effect of sulphur in this respect is much less marked. (*b*) There appears to be a critical point, which varies with the silicon content, at which the effect of sulphur changes suddenly. (*c*) The addition of sulphur increases tenacity, transverse strength, and hardness. This influence is due partly to its effect on the condition of the carbon, partly to its modifying the dendritic character of the silicious irons. (*d*) Manganese neutralizes the influence of sulphur on the condition of the carbon, and, if in excess, eliminates sulphur by separation as manganese sulphide. Sulphur is most dangerous in irons low in silicon and manganese. These results were obtained in the absence of phosphorus, and probably would be modified in the case of commercial pig iron. Cast iron is desulphurized either by the addition of ferromanganese or by Saniter's process, which consists in mixing burnt lime and calcium chloride with the molten charge, thus producing calcium sulphide. For malleable cast iron, if the Réaumur process is employed, 0·2 to 0·4% of sulphur may be present, but for 'Blackheart' castings 0·05% is the maximum permissible, since in the Réaumur process the annealing is oxidizing, and the stability of the carbide as regards decomposition into iron and graphite is of little importance, while the production of 'Blackheart' castings depends essentially on the decomposition of iron carbide. As sulphur prevents this decomposition, it is therefore harmful."—H. I. COE (*Proc. Brit. Foundrymen's Assoc.*, 1911-1912, 78), *Journal of the Franklin Institute*, clxxv, May, 1913, No. 5, p. 534. (J. G.)

GRINDING PANS *v.* TUBE MILLS.—"Experiments in the regrinding of sand at the Homestake mine, South Dakota, have resulted in the adoption of tube mills in preference to grinding pans. The details of the practice with both machines are given by Messrs. A. J. Clark and W. J. Sharwood, in *Bulletin No. 98, Inst. Min. and Met.*

The regrinding machinery consists of seven pans, each 5 ft. in dia., and a tube mill 5 ft. × 14 ft.

The pans are a modification of the Wheeler type, arranged to maintain a continuous discharge at the periphery, the depth of discharge being adjusted by the use of overflow rings of varying height, which are readily mounted upon the permanent lower portion of the shell. This shell extends 20 in. above the base-plate and determines the minimum depth of discharge.

The pulp fed to the pans is introduced in a distributing hopper, cast about the central shaft and forming part of the muller carriage. This hopper is tapped with pieces of pipe, through which the pulp is led to the bottom of the pan and discharged directly in the path of the mullers.

The total weight of moving parts when the pan has been equipped with new mullers, is nearly 2,200 lb., to which total the mullers contribute 860 lb. The pans operate at 58 r.p.m., grind a trifle less than 20 tons per day, and consume 8·4 h.p. each.

Mercury is fed in small quantities to the pans, and the pulp, after overflowing the pans, passes over amalgamating plates, one plate, 4 ft. 6 in. wide and 6 ft. long, being provided for each pan.

These plates are dressed twice daily, and the amalgam is removed on alternate days: the pan is cleaned up when new dies and mullers are to be installed, usually about once in three weeks.

The tube mill, driven by gearing from an electric motor, and provided with a scoop feeder is lined with silex blocks, formerly 3 in. and now 5 in. thick. Flint pebbles are used for grinders, experiments with ore having proved unsatisfactory.

The pulp fed to the tube mill is diluted with sufficient water to carry it freely through the pipes into which the concentrating cones discharge, and is delivered into a dewatering cone, 3 ft. 6 in. × 70° slope, which is mounted above the feed hopper of the tube mill. The thickened pulp discharged from this cone carries fully 62% solids, but it is interesting to note that the cone remains clear of any accumulation of sand, a rod introduced at the top meeting with no resistance until it encounters the bottom of the cone immediately above the bushing.

As this adjustment depends entirely upon the discharge area, it is necessary to frequently renew the bushing, and at this point the life of a bushing is not more than two weeks.

The overflow of the dewatering cone is nearly free from solids, containing but a small quantity of sand, over 90% of which will pass a 200 mesh screen. This is led around the tube mill and unites with the ground sand at the discharge, where it dilutes the pulp sufficiently to carry it over an amalgam table, 12 ft. × 18 ft. of silvered copper.

The mill grinds 73 tons per day, and consumes 37·5 h.p. when driven at 27 r.p.m.

All pulp discharged from the grinding machines unites with the main stream of pulp before arriving at the second battery of classifying cones.

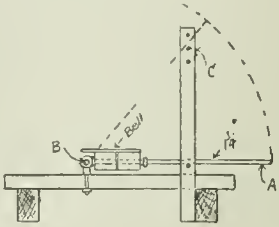
The pans show a slightly greater recovery of amalgam, but the excessive wear of iron more than off-sets all other considerations, and on this account we are preparing to install the tube mills for all future work in this department.

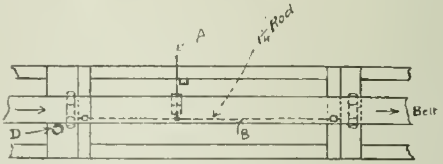
The loss of mercury at this plant amounts to 0·061 oz. per ton ground; the amalgam retorts about 26% and has a gold fineness of 630.

The two types of grinding machines have been operated on similar material with a view to determining which is the better suited for the work, and the conclusion arrived at is in favour of the tube mill. A comparison of the work done shows the pan to be more economical of power, but we consider, and experiments have shown, that if 30% to 40% more sand were fed to the tube mill, and the discharge elevated and returned to classification system, the tube mill would be at least equal to the pan in this respect.

The daily performance of the tube mill and pans is determined by timing the filling of a 5 gal. bucket (2/3 cu. ft.) and weighing the catch. Wherever possible the tipple is used for diverting the flow to the sampler."—*Mexican Mining Journal*, May 1913, p. 230. (A. R.)

SIDE TIP FOR CONVEYOR BELT.—"The accompanying sketches show a very simple and effective side tip for a 14 in. conveyor belt, installed for filling the ore-bins of a 30-head mill. Previously a mechanical distributor was used to deliver the ore at various points in the bins; this had the advantage of being able to deliver at any point in the bin, but was very severe on the belt. As at present arranged the ore from a rock-breaker is delivered by the belt at three points, two of the points being fitted up for side delivery, as in the sketches, and the other point is over the end pulley.





When the side tip is in operation the lever A is raised and held at an angle by a pin at C; when the lever is in its lowest position the ore on the belt is carried past the tip. With this arrangement the wear and tear and strain on the belt is very much less than with the usual mechanical distributor, and it is inexpensive to install, as the apparatus is merely a rearrangement of one set of three troughing pulleys, two of them being used as carriers on the lever A, and the third one as a guide pulley running on a vertical spindle as shown at D."—W. J. NICOL, *Mining and Engineering Review*, May 5, 1913, p. 314. (H. A. W.)

PLATINUM AND ITS PRICE FLUCTUATION.—"Probably no metal has experienced more fluctuation in value than has platinum. In 1874 it was worth $6 or $7 per oz. In 1898 it had increased to between $10 and $20. In 1907 its price averaged $35. Though it fell to less than $30 in 1908, probably through lack of commercial demand, it has risen steadily since then and has now reached the highest point in its history, being quoted at $46 per oz.; $50 per oz. is the present-day value of iridio-platinum suitable for jewellery making. The limited distribution and the steady increase in its demand during a long term of years give every assurance that its value will not depreciate.

The splendid character of platinum may best be expressed by those rare physical qualities which not only surpass the properties of all other commercial metals, but place it regally in a class by itself. First, it has a greater specific gravity than any other known substance, excepting only iridium and osmium, with which it is always found, and to which it has the strongest mineralogical alliance. Secondly, its melting point is far higher than that of any metal known in the history of manufacturing (most metals volatilize at the temperature in which platinum starts to melt). Thirdly, it has a remarkably low co-efficient of expansion Fourthly, no acid but aqua regia can attack it. Unlike all other metals which are used in the arts, it remains absolutely impervious to the corrosive action of oxygen or any other gas. Its untarnishable brilliancy defies the elements.

When it is combined with its natural alloy, iridium, it possesses a hardness that approaches the absolute. The greatest evidence of its superlative worth is that the nations, seeking a medium that would maintain its character forever, chose platinum from which to make the standards of the weights and measures of the civilized world. Platinum of standard hardness weighs 58·5% more than 14-carat gold. And while the gold costs but 64c. per pennyweight, the platinum has a value of $2.50 As a concrete illustration of the difference in values of the two metals, a gold jewel weighing 10 dwt. would be worth $6.40. If duplicated in platinum it would require 15 dwt., 20 gr., and would have a value of $39.58.

The cost of working platinum is far greater than that of working gold. However, the proportion of cost in relation to the value of the metal is so much less that the finished jewel will possess a far greater percentage of metal value than if made in gold."—*Mining and Engineering World*, May 17, 1913, p. 941. (A. R.)

ZINC DUST PRECIPITATION.—"The continued and increasing use of zinc dust as a precipitant for gold solutions in the cyanide industry is gradually extending our practical knowledge of the operation, even though all the chemistry involved may not be definitely known As in the development of any novel process the original advocates of zinc dust precipitation hedged the method about with certain precautions and restrictions which were dictated by their best judgment and reason. But likewise, as in the further application of such novel processes, the actual use of zinc dust in the hands of various metallurgists and under a variety of conditions has gradually altered the original practice. Step by step some of the negative precautions have been disregarded with apparent impunity, while at the same time the original positive instructions have been altered with consequent improvement. Thus, both by accident and design, fallacies have been exposed and our practical knowledge increased.

It will be recalled that one of the points strongly emphasized in the early literature on zinc dust precipitation was the necessity of preventing oxidation of the dust prior to its mixture with the gold solution. The idea was strongly held that efficient precipitation would occur only when the dust particles were fresh and uncontaminated by oxidation products. In fact, small tube mills were used to brighten the zinc just before using and successive charges of dust were constantly undergoing this treatment in order that fresh material might be available for precipitation.

This feature was embodied in patents granted to one metallurgist, who proposed to use a tube mill or other grinding apparatus in the precipitation circuit so that the mixture of dust and solution should pass through it. This procedure was adopted because such precipitants as zinc dust 'are rendered less efficient and active by reason of the presence of a deterrent coating upon the surfaces of the particles thereof, caused in part at least by their well-known property of oxidation.' The mechanical removal of the coating of oxide by abrasion was regarded as essential to efficient precipitation, because otherwise it must be removed by solution before good contact could be obtained.

It has come about, however, in at least one instance that the protection of zinc dust against oxidation by exposure to air is wholly disregarded. At the Homestake slime plant where zinc dust precipitation was first installed on a large scale the dust is exposed to air in shallow pans for periods ranging from several days to a week or more. No provision is made for subsequently grinding the dust to remove any coating of oxide that may have formed, and it is fed dry to the gold solution and pumped to the press. Nor have any ill effects followed this disregard of a generally accepted precaution, all of which suggests the probable fallacy of the original idea.

Another feature of zinc dust precipitation, which was early adopted and later abandoned, was the practice of forming an emulsion of dust and solution which was then added to the main body of the latter and pumped to the presses. The method usually adopted for forming the emulsion was to mix zinc dust with a small stream of barren solution in an inverted cone and to agitate the mixture by a jet of air. The emulsion overflowed from the cone and joined the gold solution on its way to the press. An investigation into this method of precipitation made some time ago by Mr. A. J. Clark, metallurgist for the Homestake Company, led to the condemnation of air agitation in connection with gold solutions prior to precipitation The reaction of the carbon dioxide of the air with the lime dissolved in the solution, formed calcium carbonate which was found to exercise a genuine deterrent effect on precipitation by coating the zinc particles and preventing contact with the solution. On the removal of the air-agitated mixing cones and the substitution of dry feeding direct to the precipitation line the troubles due to lime coating disappeared

The ill effects of calcium carbonate as first determined at the Homestake are now generally recognized and the emulsifying cone with its air agitation is no longer in general use. The original theory seems to be well founded and has not been changed. On the other hand, the exposure of zinc dust to the air before using is not yet generally practised, but it may be only a short time until the old opinion on this point may be broadly regarded as a fallacy."—*Metallurgical and Chemical Engineering*, Vol. xi, No. 6, June, 1913, p. 304 (J. A. W.)

PRECIPITATION OF CYANIDE SOLUTIONS.—"In the method of precipitating cyanide solutions by means of zinc dust, Mr Charles W. Merrill, of Berkeley, Cal., has discovered that it is desirable to maintain reducing conditions not only to the entrance of the press, but also throughout the precipitation and filtration in the press. Owing to the fact that some solutions filter freely, and that in some cases the supply of solution to the filter is insufficient to keep the press entirely full, the upper portion of the press is apt to contain air or other oxidizing gases. The effect of this air is to oxidize the mixture or the solid component, and thus decrease the efficiency of precipitation and the cost of the precipitant.

In presses which discharge their filtrate from the bottom, it is not always possible to maintain reducing conditions in the press, and Mr. Merrill proposes to adopt a discharge at or near the top of the press, so that the filter will always be full of the mixture, and thereby avoid the accumulation of air in the press. Any filter or press from which the atmosphere is completely excluded and which is capable of being maintained full of the mixture during the operation of filtration, may be used in connection with this process. (1,063,567. June 3, 1913."—C. W. MERRILL, *Metallurgical and Chemical Engineering*, Vol. xi, No. 7, July, 1913, p. 414. (J. A. W.)

MINING.

THE MINERS' PHTHISIS PREVENTION COMMITTEE (Interim Report to the Minister of Mines).—"It is now proposed to submit, for your information, a brief account of the investigations which the Committee has carried out, together with a summary of such information gained as seems to have a bearing on miners' phthisis.

Measurement of Dust in Air.—The first point to which attention was devoted was the measurement of the amount of dust in mine air. Investigation has shown that the dust found in a miner's lungs is exceedingly small in size. The majority of the particles of it are 1 micron (0·001 mm.) or less in diameter, only an odd grain or two being as large as 10 microns. Consequently it appeared of first importance that all the fine dust should be caught and measured, whereas the estimation of the coarse dust was of little interest, and the latter was therefore eliminated in these experiments. The earlier methods employed for collecting and estimating dust were not sufficiently accurate, and the committee, after considerable experimental work, has evolved an apparatus which collects the whole of the fine dust, but rejects the coarser particles, namely, those of 60 microns and upwards in diameter. On the whole, it may be said that whilst an apparatus has been evolved which is quite satisfactory for experimental purposes, the committee feels that there is still room for some simple and convenient apparatus which could be used regularly on mines to ascertain the amount of dust in the air at various points.

Amount of Dust in Air.—All the measurements of the committee deal with mineral dust. No attempts have been made to estimate the amount of organic dust in air. In this report, all quantities are given in milligrams per cubic metre.

(*a*) *Drives.*—When drilling in drives with machine drills, without water, certain measurements in 1903 on the Witwatersrand showed as much as 307 mgm. The average of nine measurements in 1910 showed 170 mgm. (ranging from 61 mgm. to 530 mgm.). Experiments in Cornish mines showed an average of 460 mgm.* It is, however, probable that all the above figures are vitiated by the fact that no precaution was taken to prevent the larger particles from getting into the filter tube. All the following determinations refer to fine dust only. In 1912, at the Jumpers Deep mine, comparative experiments were made when dry drilling was being carried on, the sides of the drive being also dry, when an average of nine estimations showed 59 mgm. of fine dust. Another nine estimations, when drilling wet, and with the sides of the drive wet, gave 13 mgm. Four other series of experiments (made chiefly at the City Deep mine in 1912), where water was used, showed averages of from 4 mgm. to 8 mgm. of dust in the air.

The committee is quite satisfied that dust catchers are of little value in drilling; and that the only way to keep the atmosphere reasonably clear of floating dust is to prevent its formation by the abundant use of water delivered *into* the holes whilst being drilled.

Immediately after blasting in a drive a sample taken 200 ft. from the face gave 151 mgm.; and an experiment, where two sprays were installed, showed that the air contained 91 mgm. of dust after passing the sprays; in one hour after blasting the amount had become reduced to 8 mgm., and in two hours to 2 mgm. In another experiment, where a water-blast was used, two estimations, 30 minutes after blasting, gave 8 mgm. and 16 mgm. of dust in the air, respectively; one hour after blasting 6 mgm. were found. These experiments show that, although with a free use of water much the greater proportion of the dust is quickly laid, a certain amount remains in suspension in the air for a longer period.

(*b*) *Stopes.*—The average amount of dust in stopes, whilst drilling was proceeding, varied from 4 mgm. to 6 mgm. No measurements have yet been made with regard to the dust in stopes after blasting.

(*c*) *Shovelling.*—An experiment was made in an engine chamber which was being cut underground, with a view to testing the effect of shovelling rock dry, as compared with shovelling it wet. When shovelling the rock dry, six measurements showed that there was an average of 18 mgm. of dust in the air; when shovelling the rock wet, three measurements each showed 2 mgm. of dust. It would seem, therefore, that the proper wetting of rock, before and during shovelling, is an efficient means of preventing the dissemination of dust by such operations.

(*d*) *Return Air-ways and Upcasts.*—An interesting series of experiments was made at the Aurora West mine, where the current of air passes downwards through a series of stopes to the 10th level, which serves as the return air-way. Two measurements of the dust in the return air-way, in the morning, gave 1·3 mgm. and 1·2 mgm. In the air-way a water-blast, two sprays, and a water-door—that is, a door with coarse wire gauze thoroughly wetted—were placed to see what amount of the dust made in blasting could be laid by these devices. Numerous measurements were made, of which the following may be considered typical. Before the air reached the devices, immediately after blasting, the dust, taken over a period of twenty-five minutes, proved to be 286 mgm.* The air which passed the spraying devices, taken over three consecutive intervals of fifteen minutes each, contained 80 mgm., 40 mgm., and 36 mgm. of dust, respectively. Another series of experiments was made to see how quickly the dust abated by settling after passing through the dust-laying devices. In the first five minutes, there were 164 mgm. of dust in the air, in the second five minutes 84 mgm.; and then in four consecutive intervals of fifteen minutes each, the following amounts were obtained: 24 mgm., 17 mgm., 11 mgm. and 13 mgm.

The results obtained in these experiments are instructive. In the first place, they show how large an amount of dust may be present in mine air after general blasting. It may be that the amount was larger than is usual in other mines, which might have been due, in part, to the fact that the current passed downwards through the stopes. Secondly, the experiments show that water-blasts, sprays and other devices, are powerless to eliminate entirely the dust from the air immediately after blasting. The most important inference is, that the currents of air in a mine, and the sequence of blasting, should be so arranged that men need not travel in—much less remain in—a return air-way, after blasting has taken place.

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*See Transactions of Institution of Mining and Metallurgy, 1904.

*On two other occasions, under these circumstances, the dust was returned at 540 mgm. and 425 mgm., respectively.

An experiment made in 1912 at the Village Deep mine, near the exhaust fan by which the whole air circulating in the mine is drawn upwards, gave the following results :—

Time.	Milligrams.	
2 p.m.	3·5	
2.40 p.m.	1·0	Fan stopped part of time.
3.15 p.m.	9.0	Smoke appeared at 3.35 p.m.
4.40 p.m.	12·5	
5·45 p.m.	9·0	
6·40 p.m.	5·5	
8.45 p.m.	3·0	

The effect of blasting on the amount of dust in the air is here clearly seen ; but the amount of dust is very much less than in the case of the Aurora West return air-way.

The amount of dust in the upcast air of the Simmer Deep mine was measured by Mr. McArthur Johnston at the top of the shaft, both before general spraying was installed underground and afterwards. The results are interesting :—

Time.	Sept., 1911 Without Sprays. Milligram Total Dust.	April, 1912 With Sprays, & Wetting. Milligram Total dust	Sept., 1912 With Sprays, & Wetting. Milligram Total dust
	Mineral.	Mineral.	Mineral.
9 a.m. to 10 a.m. ..	280·0	32·0	3·3
11.30 a.m. to 12.30 p.m.	130·0	21·0	0·2
2 p.m. to 3 p.m. ...	80·0	39·0	2·1
4.15 p.m. to 5.15 p.m. ...	100·0	14·0	17·1

It will be seen that the introduction of sprays in the underground workings had a marked effect on the amount of dust in the air which had circulated through the mine.

More recent experiments in the upcast air of mines, at times other than immediately after blasting, show very small quantities. Thus, at the City Deep, the average of 13 experiments was 1·6 mgm. At the Brakpan Mines, the average was 1·5 mgm.

(e) General.—The amount of *fine* dust in the air of Johannesburg, as ascertained from 18 measurements taken in the streets during April and May, 1912, ranged from 2 mgm. to 16·5 mgm. of mineral dust. If these measurements had been taken in the same manner as in foreign cities, namely, total dust, the results would have been much higher. The average amount of dust (total), present in the streets of Paris, is quoted at 7·5 mgm., or, after eight days of dry weather, 23 mgm. This is reduced to 6 mgm. after heavy rain. Hesse* states that in an iron foundry there are present from 72 mgm. to 100 mgm. of mineral dust, and in an iron mine about 14 mgm. (These estimations are, presumably, total dust.)

Thus it will be seen that the amount of dust present generally in the air of a Witwatersrand mine may, eventually, be reduced so as to compare favourably with that in the air of a city, or in factories, or in other kinds of mines. Of course, at particular points in a mine, and at certain times, the amount of dust may be excessive, but except immediately after blasting these local accumulations of dust in air can be prevented by the free use of water. The amount of dust is not necessarily a comparative measure of the danger to health which may exist, as the character of the dust is also important.

Numerous experiments were made to test whether a solution of molasses in water was superior to water for laying dust in mines. The result showed that water was, on the whole, superior for laying dust in mine workings by spraying, though probably once the dust is caught and coats the sides of the workings, the molassine solution, since it does not dry out, holds the dust more persistently.

(f) Respirators. — Numerous experiments were made with different types of respirators in order to test their efficiency. Where the amount of dust in the air was small, the efficiency was fairly good in one or two cases. The best result was obtained in one case where the dust amounted to 6 mgm. The respirator passed only 0·7 mgm.—that is 88% of the dust was caught. Where the amount of dust in the air was high—say about 50 mgm.—all the respirators failed to catch the dust, and some passed 100% of it. On the whole, the experiments demonstrated that respirators were not efficient as dust-catchers.

(g) Character of the Dust in Mine Air.—The dust in the air of mines is exceedingly small in size, and sharply angular in character. A microscopic examination of the dust in air after blasting, where no water-blast was used, showed that 10% was over 0·007 mm. in diameter, and that the remainder was mostly of a diameter of 0·003 mm., and less. Where sprays were used the average size of the dust was smaller. Several hundred millions of such particles were caught in each of the samples taken after blasting.

Both chemical analysis and microscopical examination seem to indicate that the dust in the air of mines on the Witwatersrand has less silica than the original ore, and that the dust in air which has been subjected to sprays of water has again less silica than that which has not been so treated. The following analyses are of interest in this connection :—

Constituents.	Original Ore.	Constituents.	Dust (Ignited) from Blasting.	Dust (Ignited) Passing Sprays, after Blasting.
	Per cent.		Per cent.	Per cent.
SiO_2	87·4	SiO_2	69·1	66·0
Al_2O_3	9·8	Al_2O_3	22·5	23·0
Fe	1·75	Fe_2O_3	4·5	4·0
CaO, MgO, TiO_2	Traces	CaO	1·0	4·0
S	1·1	MgO	2·0	3·0
		TiO_2	0·2	—

Medical.—Repeated microscopic examinations of the lungs of silicotic victims have been made, and comparisons have been instituted between the physical characters of the extraneous mineral matter in these lungs and those of dust arising from various mining operations. At the present time it may be provisionally stated that the dust originated by blasting most closely resembles the dust incarcerated in the silicotic lung,, and that in order of similarity this is closely followed by that arising from drilling operations—both hand and machine. Dump dust, and dust from compounds and streets, are not characteristically present in silicotic lungs.

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*See Dr. Soper's 'Air and Ventilation of Subways,' p. 201.

(h) Risk of Direct Infection by Tuberculosis in Underground Workings and on the Surface.—For this purpose, samples of expectorated material have been collected from the ladder-ways, stations, and ways underground, and from the compounds, single quarters and beer-halls on the surface. Of 322 specimens examined, 120 have been obtained on the surface, and 202 underground. Of the surface specimens, about 2·5% have been found to contain the bacillus; and of the underground specimens, about 15% have been found similarly infected. As far as they have gone, these investigations appear to justify the assertion that the underground workings are infected, to a certain extent, with the bacillus of consumption, and that the ladder-ways, owing to expectoration which is induced by the great exertion of climbing, are especially infected.

General.—A synopsis is appended showing the improvements in dust allaying effected since 1903. The committee, at a later date, will present a more detailed account of its various experiments and investigations.

We have the honour to be, Sir, Your obedient Servants,

James G. Lawn (Acting Chairman), W. L. Honnold, C. E. Knecht, W. McC. Cameron, Robert Pill, C. D. Leslie, S. C. Thomson, U. P. Swinburne, C. J. N. Jourdan, James Moir, Hugh Mitchell, W. Watkins-Pitchford, R. Raine, L. G. Irvine, M. Trewick, S. Richards, R. A. Barry, Malcolm Fergusson, A. McArthur Johnston, E. H. Bulman, Edward J. Way, B. Madew, A. Spicer (Secretary).

Johannesburg,
13th June, 1913.

APPENDIX.—Synopsis of Progress in Dust Allaying Since 1903.

Quantities of dust (mineral) are stated in milligrams per cubic meter of air, and are rounded off to the nearest 10 or 100.)

References:—M.P.C. = Miners' Phthisis Commission, 1903.
R.C = Regulations Commission, 1907.
E.R.P.M. = East Rand Proprietary Mines' Investigations.
P.I. = Present Investigations.

Working Place and Conditions.	Date.	Dust, Mgm.	Reference.	Remarks.
(A). *Drives.*				
(Dry drilling)	1903	400	M.P.C.	
	(1904	450	Cornwall)	
	1910	1,500	E.R.P.M.	Counts large particles.
	1912	100	E.R.P.M. ...	Normal average.
	1912	60	Jumpers Deep...	Dry conditions. after thorough wetting.
	1912	15 to 30	P.I. ...	'Collaring' in wet drives.
Dry holes in wet or sprayed drives	1903	50	M.P.C. ...	Average.
	1911	10	Simmer Deep...	Spray in wet drives.
(Holes not wetted inside	1912	20 to 40	Robinson Deep	
	1912	10 to 15	Jumpers Deep	
Holes in wet or sprayed drives, with water jets *into* holes	1903	40	M.P.C. ...	Water poured in.
	1907	5 to 30	R.C.	
	1910	5	Nourse Mines	
	1913	4 to 8	P.I. ...	No high results.
(B). *Stopes.*				
(Drilling)	1903	25	M.P.C. ...	Dry.
	1907	3(?)	R.C. ..	Wet.
	1910	20	Nourse Mines...	Wet.
	1912	15	Robinson Deep	Wet.
	1912 1913	3 to 8	P.I. ...	Thoroughly wet
Stopes (Shovelling)	All dates	5 to 30		Dry rock.
		2 to 5		Wetted rock.
(C). *Blasting*				
(No devices)	1903	90	M.P.C. ..	Not much explosive.
	1912	200 to 550	P.I. ...	
With sprays, etc.	1912 1913	30 to 90	P.I. ...	Dust very fine
Settling of blasting dust by gravitation	do.	50 to 60	P.I. ...	In ½ hour.
do.	do.	10 to 30	P.I. ...	In 1 hour.
do.	do.	5 to 15	P.I. ...	In 1½ hours.
do.	do.	6	P.I. ...	In 3 hours.
do.	do.	20	P.I. ...	With ½ hour waterblast.
Spreading of blasting dust	do.	20	P.I. .	1,000 ft. from face.
(D). *Compressor Air*	1912	1	P.I. ...	
(E). Present dust in open parts of wet mines	1912	1 to 10	P.I. ...	Higher quantities only in special places, or after blasting

(J. M.)

NOTES ON SINKING OPERATIONS AT THE SPRINGS MINES, TRANSVAAL.—"The shafts, now in course of being sunk at the Springs Mines, present some new features in sinking through water-bearing strata, so far as experience on the Rand is concerned, and therefore a few notes on the subject may be of interest.

General.—Two shafts are being sunk, each of seven compartments. Six of the compartments are 6 ft. 6 in. × 5 ft., inside timbers, and the seventh, or pump compartment, is 6 ft. 6 in. × 6 ft. The excavation measures approximately 42 ft. × 9 ft., and the arrangement of the shaft is shown in Fig. I.

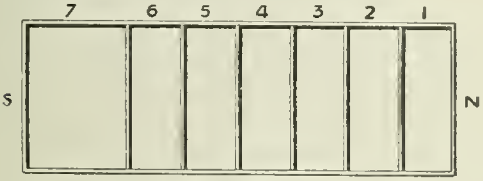

Fig. I.

The sinking conditions to which these notes refer may be divided into two periods, viz.:—

(1) The period from the date the water was struck in the dolomite, of the Black Reef series, to the time the pump chamber was started in the Witwatersrand quartzites.

(2) The period from the installation of the pump station to the time the permanent electric pumps were started.

As the two shafts are equipped in the same manner and the conditions vary merely as to the quantity of water dealt with per day, only the north shaft, in which the maximum amount of water is met with, need be referred to in detail.

FIRST PERIOD.

Equipment.—This consisted primarily of the following plant:—

Wooden headgear, 65 ft. high, erected over compartments 2 to 6.

Two 16 in. × 30 in. double drum Ruston Proctor geared winding engines, set down to haul in compartments 2 and 3, and 5 and 6.

Four 600-gal. tanks with 24 ft. shoes for baling below timbers.

One single-drum winch for hauling rock from the sinkers through No. 4 compartment.

Two horizontal low-lift Worthington pumps, 10 in. suction, 8 in. delivery, size $7\frac{1}{2}$ in. × $10\frac{1}{4}$ in. × 10 in.

The plan of the shaft, with the equipment provided for this initial period, was therefore as shown in Fig. II.

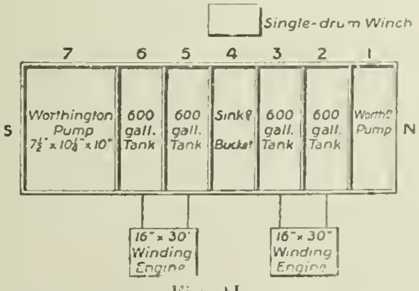

Fig. II.

The winding engines were equipped with reversing lever alarms, which reduced to a minimum the chances of overwinding due to the lever being in the wrong position at starting.

The Worthington pumps were carried on the bottom sett of timbers in such a way that they could be lifted, lowered, or handled, in any way, within their own compartments. They were further equipped with a patent pipe joint on the suctions, whereby the suctions could be disconnected by merely raising a lever. The joint was invented and constructed on the mine. The suctions were further connected by wire ropes to hand winches on the surface, in order that they could be raised when blasting was being carried out. The pumps were used solely for lifting the water from the shaft and delivering it to a ring, whence it was baled by the engines to the surface.

As the pumps were overloaded all the time, the head was never allowed to exceed 30 ft.; and when this lift had been reached, a new ring was installed, the head reduced, and the baling tanks lowered to the new level.

Water was struck at 460 ft. below the surface, and followed the shaft down along the fissure to a depth of 605 ft., at which point the Witwatersrand quartzite was entered.

Altogether, six rings were put in at depths varying from 450 ft. to 619 ft., the deepest, which afterwards delivered the water to the permanent pump sump, being below the dolomite. All these rings were connected by down pipes to run any water caught up by them to the lowest or baling ring.

The construction of the rings was as follows: A wooden bottom, 4 in. thick, was fitted carefully from the outer edges of the wall plates to the walls and ends of the shaft between the blocks. Deal baulks, 9 in. × 3 in., planed and blanketted, were placed on edge right round the shaft flush with the inner side of the wall plates and end plates, and the space between the deals and the shaft walls was filled in with concrete to the top of the deal. Similar deals were then built up on edge to the height of the next wall plate above, thus forming a vat all round the shaft about 5 ft. deep. The sides of the vat were then fitted with two 8 in. sluice valves to each compartment containing a baler, and the ring was ready for use. Crushed ant heap was most successfully used for stopping any small leaks against the shaft wall.

Procedure at Blasting Time.—The sinkers used 24 ft. fuses which were tied to and lighted from the bottom set of timbers. A man stood by each pump, and when the sinkers were ready to light up the fuses signals were given: the pumps were stopped, suction joint levers raised, and the signal to hoist suctions given. On the pumps stopping, the baling-ring gang went to the surface, and as soon as the suction started to move, the fuses were lighted, and pumpmen and sinkers went to the surface in the sinking bucket.

No damage was done to the pumps by blasting, as the water over the holes prevented damage by rock, and the time was so short, between breaking suctions and blasting, that the water never quite reached the pump platforms. After blasting, the bottom set of timbers was examined, marks given to drivers, and the two engines started baling from the bottom until the vats were no longer full. By this time, the pumps having been examined and tested, the engines were clutched in to the baling ring and the remainder of the water got out. After blasting, it took about six hours to get the water out of the bottom, and so allow the sinkers to resume work.

Following is a summary of the work done during Period No. 1, with the equipment as stated. The quantities of water were estimated from the number of 600 gal. tanks pulled per hour while the shaft was being drilled.

Summary.

Month.	Gallons of Water per Day.	Average No. of 600 gallon Tanks Hauled per Hour with each engine	Footage Sunk for Month.	Total Depth of Shaft.
1910				
August ...	1,656,000	57·5	21	459 480
September	2,073,600	72·0	20	500
October ...	1,872,000	65·0	23	523
November	2,304,000	80·0	41	564
December	2,102,400	73 0	19	583
1911				
January ...	2,073,600	72·0	32	615
February...	1,920,000	66·6	35	650
Averages...	2,000,230	69·5	27·3	

The shaft reached the bottom of the dolomite on the 12th of January, 1911, at 605 ft., and work was started on the pump station at 650 ft.

SECOND PERIOD.

Additional Equipment.—This consisted of :—

An extension of the headgear over No. 1 compartment.

A double-drum geared engine to haul 1,000 gal. tanks through Nos. 1 and 2 compartments.

Two 1,000 gal. tanks with 24 ft. shoes.

A double drum winch to haul rock from the pump station through No. 7 compartment.

The provision of false dividers and guides in No. 7 compartment, to allow rock to be hauled by the winch from the pump station.

This additional equipment was put into commission on the 2nd of March, 1911. The ropes of the old North engine were deflected to wind in No. 3 and 4 compartments, the single-drum sinking winch was discarded and the South engine equipped with sinking buckets. The object of the re-arrangement was to relieve the tremendous strain on the Ruston Proctor engines, and to concentrate the water haulage at the other end ; to release one engine exclusively for sinking, and provide another for use at the pump station.

The plan of the shaft at the beginning of the second period was as shown in Fig. III.

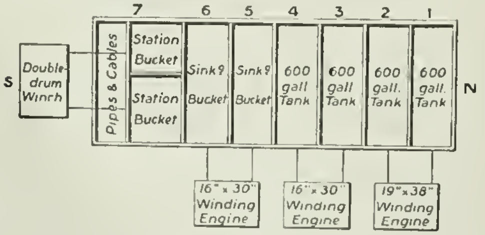

Fig. III.

The final ring below the fissure was put in at 619 ft., and all the water was baled from this level. The actual pump station level was 654 ft., or 35 ft. below the bottom ring.

The pump chamber portion of the station was completed by the 31st of May, 1911, and a temporary centrifugal pump installed with its suction connected to the ring at 619 ft. This pump was driven electrically, had a capacity of 1,500,000 gal. per day under a head of 1,000 ft., and, being situated at 654 ft., with a suction head of 35 ft., was actually working under a head of 619 ft. With an overload of 10%, and the sixth stage of the pump replaced by a blank, the pump dealt with all the water comfortably, the quantity having, fortunately, decreased considerably since the shaft got below the fissure. This pump started on the 19th of June, 1911, when the sump was scarcely begun, and, therefore, when the power went off, there was no storage for the water. This difficulty was got over by keeping the ring manned for baling day and night, with the tanks stationed in front of the sluice valves, and it was found that baling could be re-started in about three minutes.

Occasionally, however, the ring ran over, extinguishing all the lights in the shaft bottom, and this danger was relieved by installing a motor head lamp connected with accumulators on the bottom set of timbers, thus ensuring some light, at least, for the sinkers to get away by. When it is known that 70 natives and two white men constituted the shift in the bottom of the shaft, it speaks well for the efficiency of the system described that only once were the natives in any danger of drowning, and this occurred before the accumulator was provided.

The whole station was completed, and the two permanent centrifugal pumps installed, by the 30th of September, 1911.

The following is a summary of the work done during the second period. The number of tanks hauled with each engine is not given, as the tanks were then of different sizes; but, on the average, about 45 an hour were pulled with the 1,000-gal. tanks and the remainder with the 600.

Summary.

Month, 1911.	Gallons per Day.	Footage Sunk.	Total Depth of Shaft.
March	1,881,600	38	688
April	1,814,400	46	734
May	1,710,000	52	786
June	1,710,000	54	840
July	1,650,000	82	922
August... ...	1,650,000	110	1,032
September ...	1,650,000	110	1,142
Averages...	1,726,571	70·3	—

In conclusion, it might be of interest to mention that the permanent pumping plant has been so satisfactory that no delay has since been caused in the shaft bottom on account of this water, and the only stoppage of the pumps was caused by the power supply being cut off. The average rate of progress in the shaft from September, 1911, to October, 1912, has been 130 ft. per month.

APPENDIX.

Chronology of Sinking of the Two Shafts at the Springs Mines.

The first sod was officially cut 22nd May, 1909.

No. 1 (North Shaft).—

Sinking with windlasses commenced August, 1909.

Hoisting engines started 26th October, 1909.

Hard ground reached end of October, 1909, at a depth of 101 ft.

With regard to the rate of sinking, when not troubled by water, Mr. Bushell, in his notes, gives the monthly average from September, 1911, to October, 1912, as 130 ft.

For the ten months, January to October, 1912, the average number of feet sunk per month was 125.

The depth of this shaft at the end of October, 1912, was 2,829 ft., of which 2,794 ft. were timbered.

The total cost to the end of October was £138,663 0s. 5d., or an average of £49 0s. 3d. per foot.

No. 2 (South Shaft).—

Sinking with windlasses commenced August, 1909.

Hoisting engines started 30th November, 1909.

Hard ground reached end of October, 1909, at a depth of 84 ft.

For the ten months, January to October, 1912, the average number of feet sunk per month was 126.

The depth of the shaft at the end of October, 1912, was 2,569 ft., of which 2,532 ft. were timbered.

The total cost to the end of October, 1912, was £131,728 19s. 1d., or an average of £51 5s. 6d. per foot.

Note.—Since the above was written the reef was struck in the North shaft on March 25th, at a depth of 3,432 ft., and development is now actively proceeding." — B. D. Bushell. — *Bulletin, Institution of Mining and Metallurgy,* May, 1913. (A. R.)

Method of Blasting in the Lake Superior Iron District. "Herewith is a description of a very effective round of holes used for driving in hard rock in the Lake Superior Iron District. This method is used generally for driving large main-haulage drifts through jaspar and diorite. The machines commonly used are of the piston type, No. 3¼ or No. 3½ size. The air pressure at the drill varies from 70 to 90 lb. The holes are drilled large enough to permit the loading of 1½ in. cartridge. It is a well known fact that when large bore holes are used the number required to blast a face can be reduced. The round here described is drilled to pull 5 ft. in the clear and it is a very rare occurrence with this style of round for the cut to be blasted a second time.

The crossbar on which two machines are placed is set up at about the level of holes number 3 and 4. From this set-up it is possible to drill all holes in the face except the lifters. The round is planned to take out a large clean cut from the centre and then extend it on either side to the entire width of the face.

Holes 3 and 4 form the top of a 'V' cut and are given just enough slant to hold water. Holes No. 1 and 2 are started about 18 in. below the collars of holes No. 3 and No. 4 and given a steep pitch. Holes 5 and 6 are started at a height midway between the collars of the two pairs of cut holes and about 1 ft. outside of the sides of the 'V' cut. They are given a steep pitch, their purpose being to clean out the cut proper in good shape. These holes together with the four back holes are drilled with the machine above the bar. Holes No. 7 and No. 8 which complete the horizontal cut, are flat water holes drilled with the machine below the bar. The men go back as soon after blasting as the air is sufficiently clear to allow them to work, set up the bar and start work on the top of the muck-heap, to drill the holes from the upper position of the bar. When the holes drilled from the upper set-up are finished, the muck should be cleaned up, leaving everything ready for drilling the lifters. In short, this system insures the greatest possible economy of time.

The location of the collars on the holes is such that if the lifters are loaded heavy, the muck at the face will not interfere with the drilling of any other holes on the first set-up. The holes are blasted in the same order as they are numbered. As I have already stated, in Lake Superior District, it is a very rare occurrence when the cut holes do not break bottom at the first blast. The steep slant of holes No. 1 and No. 2 form a combination of the 'V' and flat cuts which makes them very effective ground breakers. Holes 3 and 4 complete a good centre cut and the rest of the work is very easy. It can also be seen that after the cut is out, the burden on the two middle back holes and lifters is not unduly great. In short, a careful examination of the arrangement of holes on this round will show that every hole has a good chance to do its proper amount of work.

In the Jasper and Diorites of the Lake Superior districts, the explosives commonly used are 60% and 80% gelatine, which correspond very closely to the 51% and 62% gelignites in that the gelatines are compared to the equivalent strength of nitro-glycerine dynamite, while the gelignites are rated according to their nitro-glycerine contents regardless as to how their strength compares with any other standard. The cartridges used are 1½ × 8 in. which allow a very dense and effective load. In large diameter bore holes, a more effective load can be obtained by using cartridges of a diameter slightly less than that of the bore hole, slitting the shells and ramming them in place, than by the mashing of small cartridges to fill the bore hole."—Charles S. Hurter, *Mexican Mining Journal*, May 1913, p. 229. (A. R.)

Comparative Safety of Electric and Fuse Blasting.—"It it usually urged in favour of electric blasting that on account of its being controlled from a distance it is safer than detonation by fuse. A recent accident in a Catskill aqueduct tunnel, where electric firing is employed, was reported in the daily press as caused by premature explosion. As a matter of fact, it was not, but the report suggested that possibly there is a danger concealed in this method which is not duly appreciated. It is possible to take great precautions in electric firing such as making it necessary to close several connections to complete the circuit, and placing the operation in the hands of an experienced man with the apparatus under lock and key. But the danger of accidental closing of the circuit cannot be entirely eliminated, and the system has the disadvantage of setting off the blast instantly and without warning, whereas a spit fuse, like a rattlesnake, gives fair warning of its presence. In connection with this an abstract by *Engineering Record*, February 15, 1913, of a report by O. Dobbelstein, in *Glückauf*, is interesting.

Many of the new German regulations, he says, are based on the belief that the system of firing by electricity is safer than that of ignition by the ordinary cord fuse. Electric firing, for example, has been made compulsory since January 1, 1912, in certain classes of mines in the Dortmund district. The belief is founded on statistical evidence given below. These figures deal with the years 1908 to 1911, inclusive, and show the number of fatal injuries in that period to have been 43 with electric firing and 51 with safety fuse, the injuries causing permanent disablement being 44 and 66, and temporary disablement 131 and 146, respectively. These figures, Mr. Dobbelstein contends, being absolute are valueless. In place of them he gives a table, in which the figures are related to the number of men at work in the mines. In those in which electric firing only was permitted, 71,555 men were employed, the number in the mines using the ordinary fuse being 112,698.

The proportion per 1,000 is as follows: For fatal injuries 0·111 for electric firing and 0·076 for fuse; for permanently injured 0·101 and 0·105; and for slightly injured 0·328 and 0·220, respectively. Thus these statistics are in favour of the ordinary fuse system. This fact, it is stated, must be attributed to the often imperfect electric fuses and unsuitable firing machines used, and to the ignorance of the miner. It is practically impossible either to misunderstand or to misuse the cord fuse. But in this case also safety depends largely on the quality of the fuse. The relative value of these two systems is said to be becoming a question of the highest practical importance in view of the government regulations which are now being formulated to cover this subject.

Doubtless, as Mr. Dobbelstein states, the accidents with electric firing are largely due to the poor detonators and firing apparatus and to the ignorance of the miners, but even the elimination of these unfavourable conditions would probably only put the method on a parity with fuse firing in point of safety. The choice between the two systems would not seem to be capable of legislative regulation There are many defects in our state mining laws, and some grave omissions, but the choice of electric or fuse firing is one that may be safely left to the judgment of the operators, for the present, at least. Data of wider derivation and especially data on comparative results in America, may eventually show definite advantages for one system. The collection and analysis of such data is a thing highly to be desired." —*Engineering and Mining Journal*, May 17, 1913, p. 1014 (A. R.)

SELECTIVE MINING ON THE WITWATERSRAND.—"A difference of opinion seems to exist still as to the best mining policy to pursue on the Rand. Some engineers maintain that the big mill policy has proved a mistake, while others maintain the reverse opinion.

The past five years' experience on the Rand has been a somewhat peculiar one. Until last year the yield and profits per ton milled were constantly declining, while since 1909 the costs are steadily increasing, and this despite the fact that the tonnage milled was year by year steadily increased. With a higher divisor in the shape of a higher tonnage milled at individual mines, a lower cost per ton with every show of reason might be anticipated. This, however, has not always proved to be the case, and perhaps it is to some extent on this account that the big milling policy has in some quarters been condemned. The year 1902 is the first for which reliable figures as to cost are available, and since that date down to 1909 working costs steadily declined, from 25s. 9d. per ton milled, to 17s. 1d. Since 1909 working costs steadily increased, but what is perhaps more remarkable, is that until last year in face of the increasing costs, the yield and profits per ton milled continued to decline. Last year's improved yield from 27s. 11d. to 29s per ton, does not appear to be due to better sorting, because official figures show that less sorting was actually done. Improved extraction cannot by any means account for it, and the only explanation seems to be that a better grade of ore was sent to the mills.

If the figures published by the Transvaal Chamber of Mines are to be relied upon, and there is no reason whatever to doubt their reliability, some reason must exist for the marked and sudden improvement in the grade of ore sent to the mills. No one fully acquainted with the Rand will for one one moment claim, that taken on the whole, the reefs on the Rand are improving in value. In the majority of cases as greater depths are attained by mining operations, the reverse is the case. Then again on the Central Rand many of the richest mines have practically exhausted the best ore, and have no other alternative but to send to the mill ore of a lower grade. The only explanation that can be advanced to account for the higher grade of ore sent to the mills, is that selective mining on a much more extended scale has been practiced. Proofs of this exist by noting last year's results at the East Rand Proprietary Mines, where not only selective mining as ordinarily understood, but selective development was also practised. Then again it is known that selective mining has been introduced on several mines on the far East Rand, while in addition, the starting up of such comparatively rich mines as City Deep and Modder 'B' must also produce some improvement on the general grade.

At a recent meeting of the Consolidated Goldfields of South Africa, Lord Harris, the chairman, implied in his speech, that 'selective mining' was the same as the old condemned practice of picking the eyes out of a mine. Now such a definition of selective mining must not for a moment be allowed to pass unnoticed, because while it may be true in some instances, it does not convey a correct meaning of selective mining as generally applied on the Rand. Whereas on the Rand payable areas alternate with unpayable stretches of ground, a manager would indeed be foolish to follow a fixed rule of mining, involving the leaving of pillars of payable ore, and the stoping of unprofitable ground, or wilfully work without sufficient reason, any part of the mine known to be unpayable. Both in the eastern and western sections of the Rand, varying proportions of unpayable ground are proved to exist, while on the Central Rand the gold contents occur in a more uniform manner. In the eastern and western sections of the Rand, it has therefore been found expedient to limit the mining operations more or less to the payable zones; leaving the unpayable areas for subsequent working, when working costs may be perhaps lower. Now this is selective mining in its broadest sense, and where mining conditions require it, the practice cannot be condemned, but if on the other hand its adoption is not absolutely necessary, or calculated to damage the future of the mine, then some careful consideration is required before its adoption seems justified.

It will therefore be perfectly clear that the underground conditions of each individual mine must be carefully considered and weighed before recommending any particular method of mining. When Lord Harris made the remark mentioned above, he was replying to the demand made by several shareholders in the Simmer Deep and other low-grade mines that selective mining should be tried, with the object of increasing the profits and earning a dividend. To this request the directors of the Consolidated Goldfields of South Africa, have given their attention even to the extent of sending their London consulting engineer to the Rand, to consider the suggestion on the spot, and to report his recommendations to the board. Now at the Simmer Deep mine the total ore reserves amount to 1,574,989 tons of an average value of 4·2 dwt., and of this amount only 6% or barely sufficient to keep the mill running for two months, assay as high as 4·75 dwt. It is clear therefore that in the case of the Simmer Deep the adoption of selective mining, no matter how carefully carried out, could add but little to the value of the yield, in all probability not sufficient to cover the extra costs incurred by its adoption. It would, it is certain, result in crippling the future profit-earning capabilities of the mine. If the yield cannot be appreciably increased, it follows that the only way to increase the profits, is to lower the working costs, so that the Simmer Deep may be regarded as a striking example of a mine where selective mining cannot be recommended. Figures show that at the Simmer Deep, large scale working lowers the working costs to an appreciable extent, and this method of increasing the profits is about to be tried.

It is probable that the East Rand Proprietary mines afford a more normal illustration of an average Rand mining property than the Simmer Deep, and as selective mining has apparently been in full swing on the East Rand Proprietary mines for the last year, a comparison with previous years may prove interesting. It is well known that for purposes of comparison the results for 1911 ought perhaps to be avoided; so 1910 will be taken as being a more normal year. The results for the two years come out as follows:—

Year.	Stamps.	Tons Milled.
1910	820	2,126,334
1912	820	1,848,050

	Yield		Costs	
	Amount.	Per ton.	Amount.	Per ton.
1910	£2,938,831	27s. 7d.	£1,689,792	15s. 10d.
1912	2,967,443	32 1	1,928,349	20 10

Profits	
Amount.	Per ton.
£1,249,038	11s. 9d
1,039,093	11 3

If the results for 1910 are truly indicative of the capabilities of the East Rand Proprietary mines under normal conditions of mining, the results of last year's selective methods of mining were, to reduce the tonnage milled, increase the yield by 5s. 6d. per ton, add 5s. per ton to the working costs, and reduce the working profits by 6d. per ton. Another effect of adopting selective mining has been to reduce the tonnage of payable ore reserves from 10,000,000 to 6,000,000 and apparently without any advantage as far as profits are concerned. Most mining engineers would therefore conclude that if 1910 results are reliable, selective mining carried on to the extent it was at the East Rand Proprietary mines 1912, can scarcely be regarded as a striking success, or a policy that ought to be permanently adopted. It must not, however, be concluded from what has been stated about the Simmer Deep and East Rand Proprietary mines, that selective mining should be always avoided on the Rand. In some mines some species of selective mining will always be necessary if the best results are to be obtained. At the Simmer Deep for instance, wholesale large scale mining in its fullest sense must be aimed at to keep down the working costs and increase the profits. At the East Rand Proprietary mines a more judicious application of selective mining than was tried in 1912, coupled with large scale mining, seems to be preferable. Generally speaking, the best policy of the Rand to-day would seem to be to take an estimate of the average capabilities of the mine and make the mining policy what seems best calculated to produce the best results spread over the life of the property. The majority of the mines on the Rand to-day are compelled to adopt some species of selective mining, but the advantages of large scale wholesale mining should be recognised and reaped wherever possible."—ROWLAND GASCOYNE.—*Mining and Engineering World*, May 31, 1913, p. 1041. (A. R.)

LIQUID AIR FOR BLASTING.—"Experiments made at the Rudersdorf works, under the auspices of the German Society for the Oxygen and Nitrogen Industries, are officially stated to have given highly satisfactory results. The main advantages of the method are that no danger whatever is incurred in manufacture, storage and transport of the explosive, or in replacing cartridges which have missed fire. Furthermore, the working safety underground is considerably increased, and owing to the presence in the mines of considerable quantities of oxygen in the shape of liquid air, rescue work, if necessary, is greatly facilitated. Lastly, owing to the advantage that the component parts of the new explosive can easily be produced at the mines themselves, the mining industry is made independent of the explosive factories. The only drawback connected with the new method is the at-present comparatively high price of liquid air, but, as the main item in the cost of production is the power required for working the plant, it appears probable, in view of the cheapness of fuel for power generation in coal mines, that processes will soon be elaborated enabling liquid air to compete successfully with the explosives at present in use. The manner of operating with liquid air is very simple. The cartridge with the dry carbon carrier is placed in the borehole without the liquid air. Only at the last moment, when all the preparatory work has been completed, the liquid air is added and at once ignited. Cardboard is used for the cartridge case. A tube of finely meshed wire work placed through its longitudinal axis distributes the liquid air through the whole cartridge mass. The liquid air is kept in cans constructed after the Dewar system and pressed into the cartridge by a hand pump with 0·4 atm. pressure. The cartridge is placed in the borehole with a mixture of kieselguhr and petroleum. The simultaneous filling of three blasting holes does not take more than one minute. The shots proved to be at least as effective as those with ammonium cahücite, the explosive which had formerly been in use. The cost of one shot with the latter would have been about 6d., while the quantity of liquid air consumed amounted to about one quart."—*London Mining Journal*, May 31, 1913, p. 528. (A. R.)

CONGLOMERATES OF THE WITWATERSRAND.—"Opinions regarding the origin of these enormous gold ore deposits still differ considerably. Four principal theories have been adduced which are shortly discussed as follows:—

1. *The theory of the purely mechanical distribution of the gold in its present condition.*—According to this theory, the conglomerates are 'fossil' fluviatile placers; formerly loose, now consolidated pebbles and pebble deposits, whose gold particles have floated in purely mechanically. This view is at once negatived by the microscopical examination of the gold in the conglomerates. The gold particles have absolutely no resemblance to mechanically transported placer gold. Its distribution in the reef is also not that usual in placers. Moreover, the conglomerates, in spite of their approximately low gold content, are as a rule richer than placers. Finally, the absence of gold in the quartz of the pebbles is evidence against fluviatile origin, as also the fact that nuggets which are occasionally found in placers never occur in the bankets.

2. *The theory of the purely chemical precipitation of the gold and the pyrite in the sea.*—According to this precipitation theory, as it may be shortly called, advocated especially by Penning (1888), J. P. Hamil-

ton, De Launay, Stelzner, F. W. Voit, and others, the gold has been precipitated solely chemically from the water at the time of the deposition of the pebbles in a portion of the sea near the coast. This seems unlikely (*a*) because in disturbed water, as in a littoral zone where pebbles could be carried and deposited, such a precipitation is difficult to imagine: (*b*) because it should at least also have occurred during the deposition of the sands from which the quartzite beds are formed; (*c*) because the maximum gold value of the present day sea, as proved by Liversidge and others, does not suffice, and assisttance has to be obtained from auriferous mineral springs; and it is further very unlikely that these springs should have ceased to flow whenever sand was deposited to again break out when pebble masses were brought down. Considering the very great distance between the Main Reef and the Black Reef, these hypothetical springs must have flowed in this wonderfully intermittent manner during a very long period of time.

Recently the possibility of a precipitation of such a very small amount of gold as is contained in sea water has, through the experiments of J. R. Don, been seriously questioned. In no wise was he able in his experiments to precipitate gold from sea water with the reagents occurring in nature. Moreover, it may be mentioned that the adherents of this theory seek to explain the precipitation through the occurrence of finely distributed organic substances. They even consider that the strong movement of the water during the deposition of the conglomerate layers has favoured precipitation. (F. W. Voit.) But this organic substance is not generally distributed.

3. The theory of former placers in which the gold has subsequently been re-deposited.—This view was propounded by E. Cohen and adopted by F. Posepny, but G. F. Becker has more fully attempted to prove it. Mr. Becker compared the conglomerates with the recent marine placers on the coasts of California, Oregon, and Alaska, the material of which undoubtedly originated from the denudation of gold quartz veins. Gold seldom occurs in the pebbles of such recent marine placers; it also evidently occurs in them re-crystallized, and its distribution is, contrary to that in fluviatile placers, approximately even over its whole width; finally, nuggets are here also absent. He connects the re-crystallization of the gold and of part of the pyrite, which he admits occurs on the Rand on an extended scale, with the eruption of diabase. This theory is certainly fascinating, but it does not attempt to remove a weighty objection, namely, that were it correct, the gold should by rights occur primarily in the quartzite sandstone layers, as the material of recent marine placers are generally principally sandy. J. W. Gregory has recently advanced the theory in a somewhat altered manner. He attempts to explain the auriferous conglomerates as shore deposits which have been subjected to the action of the waves during a long interval of time, being thus enabled to retain the minute gold particles within their interstices, while the light sand and mud was continually removed. Iron oxide grains were originally present, which later were converted into pyrite. But why is ilmenite, so characteristic of marine placer conglomerates, absent?

4. The theory of the subsequent transportation of gold to the conglomerates.—The view, well worthy of consideration, which, rightly described, has also been called the theory of subsequent mineralization and enrichment of the conglomerates, has been specially defended by E. Dorsey, A. R. Sawyer, Gardner F. Williams, J. S. Curtis, J. Kuntz, K. von Kraantz, P. Krause, John Hays Hammond, F. H. Hatch, G. S. Corstorphine, and R. B. Young. It can be described as the prevailing view in the circle of Johannesburg mining engineers.

The assumed subsequent penetration of auriferous solutions is connected with the eruption of diabase. The pyrite, according to this theory, is considered as primary and at the same time as a precipitating material of the gold, as actually demonstrated by Johansson and Liversidge, who obtained a precipitation of gold in very dilute solution of chloride of gold through pyrite; or at least the precipitation of pyrite occurred a little sooner than that of the gold. It certainly cannot be proved that the amount of gold is proportional to that of the pyrite. There are even higher pyritical lodes without gold. Hatch and Corstorphine have therefore suggested organic substances which actually occur in the conglomerates as a precipitation medium. I have already mentioned, however, that these are not generally present. The uneven distribution of the gold within the conglomerates and the occasional enrichment of the lodes in the vicinity of eruptive masses favour the infiltration theory.

The fact that elsewhere in the Transvaal, in the Lydenburg district, undoubted sediments, here dolomite of the Malmani horizon, have been mineralized through solutions which deposited quartz and gold, together with copper, is corroborative of the infiltration theory. The observation that in a much higher geological horizon, that of the Black Reef, the auriferous character of a conglomerate is repeated, also the absence of a constant horizon of ore deposits of this nature, is favourable to the theory.

The principal objection to this theory should certainly not be passed over: namely, that only single layers, not all conglomerate reefs, are found impregnated with gold, that therefore no equal impregnation zones run through all layers of the Witwatersrand along fault planes or dikes.

Against this the originally unequal porosity of such rock layers should be pointed out. Thus it appears that the cement of those beds known as poor, like the Main Reef, Elsburg, Bird, and Kimberley Reef, have become denser through a larger admixture of finest quartz sand than the other richer reefs. When apparently porous layers, such as the breccia-like 'Bastard Reef' below the Main Reef Leader in the New Goch mine (J. W. Gregory) is found poor in gold or with gold absent, this is certainly remarkable. Yet the question is whether this condition has not been occasioned by later removal of cement.

Objections to Infiltration.—A statement frequently brought forward against the infiltration theory is the view that channels are absent. J. Kuntz has meanwhile (1908) pointed out that south of Greys Mynpacht on the West Rand auriferous quartz veins were formerly worked, and that possibly such poor reefs have frequently occurred but been little observed. Also an auriferous diabase dike with up to 12 gm. to the ton is known at the Meyer and Charlton and the City and Surburban. I also noted an apophysis in the conglomerate which was projected from a diabase dike which intersected it. This offshoot was full of visible free gold, and even the South Reef was enormously rich at the immediate contact (over 6,000 gm. per ton). The diabase mass had in part replaced and filled the interstices between the pebbles. Much more numerous, according to J. Kuntz, are instances in which non-auriferous diabase dikes or faults have exerted a visible enriching influence on the conglomerates in contact on either side. Such an enrichment is known, for instance, in connection with a diabase dike at the Princess Estate and at the fault in the Bird Reef group of the Van Ryn and Chimes mine. In the latter case the enrichment extends for 200 metres on either side.

The most important argument for the infiltration theory is certainly the dependence of the value of the gold in the conglomerates on their dip, as has been lately demonstrated by J. Kuntz, who has practised on the Rand for many years. The steeper the dip of the conglomerates, the stronger have they been compressed and the more faulted they have become. This accounts for their greater gold values in comparison with the undisturbed reefs which are slightly or not all inclined. 'Compare,' wrote J. Kuntz (1908), 'the gold values in the reefs of the mines lying in relation to depth below one another, Ferreira, Ferreira Deep, Robinson Deep, and Turf mines, in the Central Rand; Champ d'Or, French Rand, and Tudor in the West Rand, or the several Knights in the East Rand, or the gold values of the Deep Level mines, with the results obtained in the deepest boreholes, and it will be found how clearly this appears to be the case.' The deeper the reefs, the flatter and more undisturbed are they known to occur. That this rule is not without exception is shown in the tables of F. W. Voit and Kessler."—Richard Beck, *Mining and Scientific Press*, Vol. II., May 10, 1913, p. 693. (A. McA. J.)

MISCELLANEOUS.

Subjects for Papers.—"The Council of The Institution of Mining Engineers invite original communications on the subjects enumerated in the following list, together with other questions which are of interest to mining and metallurgical engineers.

Assaying.
Boiler explosions.
Bore holes and prospecting, including methods of boring.
Brickmaking by machinery.
Brine-pumping.
Canals, inland navigation, and the canalisation of rivers.
Coal-getting by machinery.
Coal-washing machinery.
Coke manufacture and the recovery of by-products.
Colliery leases, and limited liability companies.
Compressed air as a motive power.
Corrosive action of mine water on pumps, etc.
Description of coal-fields and other mineral deposits.
Distillation of oil-shales.
Drift and placer mining.
Duration of coal-fields of the world.
Electric mining lamps.
Electricity and its applications in mines.
Engine-counters and speed-recorders.
Explosions in mines.
Explosives used in mines.
Faults and veins.
Fuels and fluxes.
Gas-producers and gaseous fuel and illuminants.
Gas, oil and petroleum engines.
Geology and mineralogy.
Gold recovery plant and processes.
Haulage in mines.
Industrial assurance.
Inspection of mines.
Laws of mining and other concessions.
Lead smelting.
Light railways.
Lubricating value of grease and oils.
Lubrication of trams and tubs.
Maintenance of canals in mining districts.
Manufacture of fuel briquettes.
Mechanical, magnetic and other methods of preparation of ores and minerals.
Mechanical ventilation of mines, and efficiency of the various classes of ventilators.
Mining of mineral deposits.
Natural gas: its conveyance and uses.
Occurrence of mineral ores, etc.
Ore sampling and sampling machines.
Petroleum deposits.
Preservation of timber.
Prevention of overwinding and accidents in shafts.
Pumping machinery.
Quarries and methods of quarrying.
Rescue work.
Rock drills.
Safety lamps.
Screening, sorting and cleaning of coal.
Shipping and discharge of coal cargoes.
Sinking, coffering and tubbing of shafts.
Spontaneous ignition of coal and coal seams.
Steam condensation arrangements.
Steam power plants.
Submarine coal mining.
Subsidences caused by mining operations.
Surface arrangements at mines.
Surveying and surveying instruments.
Transport on roads.
Tunnelling methods and appliances.
Utilisation of dust and refuse coal.
Underground conveyors.
Ventilation of coal cargoes.
Water as a motive power in mines.
Water tube boilers.
Watering coal dust.
Water incrustations in boilers, pumps, etc.
Winding arrangements at mines.
Winning and working of mines at great depths.

For selected papers, the Council may award prizes. In the conferment of awards, no distinction is made between communications received from members of the Institution and those received from others."—Transactions of the Institution of Mining Engineers, *The Institution of Mining Engineers*, Vol. xliv., Part 5, p. 20. (A. R.)

The Uranium and Radium Situation.—"Some months since rumours reached the U. S. Bureau of Mines of an increased demand for carnotite ores from Colorado and that these ores were being shipped abroad in some quantity. Further, it was reported that the methods of production involved large losses of material and that methods for concentrating low-grade material now being thrown on the dump were greatly needed. Accordingly, R. B. Moore and K. L. Kithil were assigned to the task of investigating the situation with headquarters at Denver, where the Bureau established a laboratory for the purpose of investigating the rarer metals occurring in the Western States and problems bearing upon the prevention of waste and increased efficiency in the mining industry. The surprising conclusion has been reached that while all the radium placed upon the market in the last few years has been produced in Europe, a large portion of this output has come from American ores.

Radium institutes have been established in Austria, France, Germany and England, a European science and industry have been developed from American radium ores, and even the uranium present with the radium has been manufactured into marketable condition only in foreign countries and returned in finished condition to our own. Purchasers have been forced to procure from abroad such radium as they could afford for experimental purposes and have been hampered by the almost prohibitive prices at which the finished material is held.

While the Austrian Government, realising the untold possibilities of the radium ores of St. Joachimstal, has purchased the mines, put their output under direct governmental supervision, and has entered into an arrangement whereby this ore is worked up in co-operation with the Vienna Academy of Sciences for experimental purposes in a carefully administered radium institute, America has allowed her large and much greater resources to be exploited on a basis which wastes perhaps irretrievably a large portion of the material mined, and has exported carefully selected ores at a price by no means commensurate with its radium value if worked up at home.

Even before carnotite was exported, pitchblende of the highest grade was sent out of the country at the time when the world's radium output was supposed to be coming from Austrian ores. At least 20 to 25 tons of high-grade pitchblende has been sent out of the country. Within the last two years, however, foreigners have realised the value of our carnotite resources and most of the radium that has been exported has gone abroad in this ore.

During the last year, carnotite carrying 28·8 tons of U_3O_8, from which 8·8 gm. of radium chloride, or 11·43 gm. of radium bromide could be obtained, were produced. Practically all of this ore was shipped abroad for the extraction of radium. The value of the radium salts extracted would be at the minimum market price, $528,000. The total supply of radium salts from all other sources including the Austrian mines was probably not more than 3·65 gm. of radium chloride, basing the production of the Austrian mines for 1912 upon that of 1911, which is known.

Pitchblende, the richest of all uranium minerals, is composed mainly of uranium oxide, but also carries lesser quantities of a large number of other substances. It has been found in small quantities in Connecticut and in the felspar quarries of North Carolina. Practically the total American output has come from the mines in Quartz hill, Gilpin county, Colorado. The mineral is a heavy black substance which can be readily identified by any one who will suspend a sample of the pitchblende above a photographic plate wrapped in black paper and kept in the dark for a few days with a key or other metal opaque to radium radiations placed between the sample of ore and the plate, so that when the plate is developed a shadowgraph of the object may identify the ore. Pitchblende may carry as high as 80% uranium oxide, although the average ore is not nearly as rich.

Carnotite is a yellow mineral consisting mainly of potassium uranyl vanadate, but containing also small amounts of barium and calcium compounds. Being a uranium mineral as is pitchblende, it of necessity carries radium, although it has not yet been definitely established that the uranium and radium are in equilibrium as they are in pitchblende. However, it is known that in our western carnotite the amount of radium is not far from the equilibrium ratio and in calculations given above an allowance of 10% has been made to cover this possible deficiency. While carnotite is known to occur in smaller quantities in other States, the more important deposits are scattered over a considerable area in Colorado and Utah, embracing Meeker and Skull creek, Colorado, Green river, Thompson's Moab, Richardson, Table mountain, Pahreah and other places in Utah. The largest proportion of the ore, however, has been produced in or around Paradox Valley in southwest Colorado, from which it has to stand long hauls by pack animal or wagon to the railroad. Carnotite always carries vanadium as well as uranium and radium, but is purchased almost wholly on its radium content, comparatively little being allowed for the vanadium present.

The ore consisting of a fine-grained sandstone, containg yellow, finely-pulverized carnotite, occurs in pockets and is easily mined. As ore below 2% uranium oxide cannot at the present time find a market, a considerable portion of the ore has been thrown on the dump and is now being wasted, as material of lower grade has to be discarded on account of the long haul and the fact that European buyers have set this standard as to quality. Ores of higher grade are sometimes obtained, but they occur only in small pockets, and it is generally advisable to mix these high-grade ores with ores of somewhat lower grade in order to increase the marketable output. Ore of 2% uranium oxide is now worth approximately $75 per ton f.o.b. New York.

In the mining of these carnotite ores it is probable that 5 tons of material capable of concentration are thrown upon the dump for every ton that finds its way to market. To develop methods for concentration of these ores and save the valuable material now wasted is one of the problems before the Bureau of Mines with fair prospect of a successful conclusion.

It is difficult to estimate the total amount of radium that has been produced up to the present time, but it is quite certain that if the ores which have been mined in this country and abroad and sold for radium production have been actually worked up into this material. there is now in existence something like 40 gm. (1¼ oz.) of radium. The price of radium salts varies somewhat. In large quantities it has been $60,000 per gram for both radium chloride and radium bromide, although the latter contains less metallic radium in proportion to its weight than the former. It should be remembered, therefore, that it is more advantageous to purchase radium chloride than radium bromide. In small quantities the average price has been $80,000 per gm., which represents about $2,250,000 an ounce.

The figures given show very plainly that the United States has taken the palm from Austria as the radium producing country of the world. Very few people have been cognisant of the fact that we have such deposits within our borders. Up to the present time very little interest has been taken in the matter, and only one firm has engaged in the extraction and refining of radium in this country—a condition which is deplorable. This firm has not yet entered the radium market. Practically every ton of ore mined in 1912 went abroad.

The applications of radium are still too little understood to admit of definite statement. Its discovery and marvellous properties have already changed our ideas regarding the constitution of matter, and scientific investigation will undoubtedly lead to valuable results which we cannot now even foresee. Altogether too many incorrect statements and vague speculations have been placed before the public as to its use in medicine. A recent report of the London Radium Institute and the many articles emanating from minor laboratories experimenting in the application of radium to therapeutics, all tend to show, however, that it has a real value, the certain application of which must await further experimentation. In the meantime no credence should be given to the many stories that are sure to be printed unless they are backed up by the highest medical authority which will always give publicity with caution.

The best medical authorities appear to agree that up to the present time, radium has not been proved to be specific for any disease, although it has been shown to be helpful in many cases, and the outlook for its future application to certain diseases not easily treated otherwise is decidedly encouraging.

Apparently no uranium is worked up in the United States, but according to statistics gathered by the division of mineral resources of the U. S. Geological Survey about $14,000 worth of its oxides and salts were imported into the United States in 1911. It is one of the few materials shipped abroad as ore and returned in manufactured form.

A preliminary report on uranium, radium and vanadium by R. B. Moore and K. L. Kithil will soon be issued by the Bureau of Mines. This bulletin describes the carnotite deposits of Colorado and Utah and the pitchblende deposits of the former State. It also contains details, of which the foregoing is simply a general summary, which cannot fail to be of value to all those interested in our mineral resources and their development."—CHARLES L. PARSONS, *Mining and Engineering World*, May 10, 1913, p. 909. (A. R.)

UTILISATION OF PINEAPPLE LEAVES.—"Dr. E. V. Wilcox, Director of the Hawaii Experimental Station, has recently sent to a firm of paper manufacturers in the United States a supply of pine-apple leaves, and this firm have made a favourable report on the fibre produced from them. It is found suitable for a kind of tough paper used for insulating purposes. It has been proved that the fibre can be produced with the machinery for making sisal fibre. Seventy pounds of fibre to the ton of pineapple leaves can be produced, which at 14 tons of leaves would be 1,000 lb. of fibre to the acre. At a profit of only one half-penny a pound this means £2 an acre for what is now waste. Dr. Wilcox is stated also to have received reports from manufacturers of varnish in various cities, upon samples of kukui-nut oil sent from the Hawaii experiment station. They are to the effect that it is far better than linseed oil for making varnish. Honolulu at present imports 50,000 gallons of linseed oil each year. Experiments are being made to produce a press to extract the oil."—*Journal of the Royal Society of Arts*, Vol. lxi., No. 3154, May 2, 1913, p. 615. (J. G.)

SOUTH AFRICAN SHEEP DIPS.—"The Bradford Chamber of Commerce has recently received a letter from the Chief of the Agricultural Department of the Union of South Africa at Pretoria asking for information regarding the grounds of the objections of wool-users to the use of caustic soda and sulphur, and lime and sulphur for sheep dips. The Chamber was requested to recommend an unobjectionable dip which would be likely to give equal satisfaction to the farmers.

On behalf of the Bradford Chamber of Commerce, Mr. Fawcett, president, replying to the letter, gave the following reasons for the objection of the Chamber to the use of caustic soda and sulphur, and lime and sulphur dips:—(1) Because any wool so treated will be damaged, and will fetch a lower price. Caustic soda and sulphur is used as sodium sulphide for fetching wool off the dead fleeces. (2) Wool so treated being partially dissolved, the resultant weight of the fleece is considerably less. (3) It makes the wool more difficult to scour. On solution in water the caustic soda is set free and acts on the wool, causing it to shrink. (4) The effect of alkaline soda salts destroys the spinning qualities of the wool, decreases its elasticity, and causes pieces made from such yarn to take a ruddy colour in the dyeing. Wool so treated will not spin to fine counts. (5) Wool treated with alkalis has a stronger affinity for colouring matters than wool not so treated. (6) The use of sulphur not only renders the wool difficult to wash, but while in the first instance it produces a good colour on the fibre later the colour deteriorates and becomes worse than before the sulphur was applied. (7) There is no necessity to use dips which are injurious to the fleece, as there are other dips upon the market, as recommended in the Report of the Departmental Committee of the Board of Agriculture and Fisheries, which are quite as efficacious in the curing of scab without having any injurious effect on the wool."—*Journal of the Society of Chemical Industry* (from *Chem. Trade J.*, May 10, 1913), May 31, 1913, p. 530. (F. W. W.)

Reviews.

EVERYDAY USES OF PORTLAND CEMENT. Cloth, octavo, 360 pages, illustrated. London: The Associated Portland Cement Manufacturers, Limited. Price 2s. 6d.

"This is the third edition of a book that has had a wide circulation. It gives valuable practical information relating to the application of cement to concrete construction, and it is therefore of use to the mining engineer, who is now required to know all about concrete shafts, supports, etc."—*Mining Magazine*, May, 1913. (A. R.)

A MANUAL OF PETROLOGY. By F. P. MENNELL. Cloth, octavo, 260 pages, with many illustrations. London: Chapman & Hall. Price 7s. 6d.

"This is a new edition of the author's 'Introduction to Petrology,' with many re-arrangements and additions. Mr Mennell is an authority on Rhodesian geology, and he has contributed many articles to the geological journals and transactions. The book contains many references to Rhodesian and other South African occurrences of rocks, minerals, and ores, so much so as to give it a distinctive character. Moreover, the information and views are founded largely on his own observations."—*Mining Magazine*, May, 1913. (A. R.)

METHODS OF CHEMICAL ANALYSIS. By F. A. GOOCH. Chapman & Hall, London. Pages 536+XII, including Indexes. Chapters XII, with 29 illustrations. Price 17s.

"This volume is apparently being used in the Yale University laboratories, and contains useful information as to the general methods of chemical analysis. The general scheme includes the estimation of beryllium, boron, lanthanum, cerium, thallium, vanadium, and other metals. It is a useful volume for the laboratory bookshelf, and contains information which is not generally met with in the more general text-books on the subject."—*The Chemical World*, June, 1913. (J. W.)

TREATMENT OF CONCENTRATE AT THE GOLDFIELD CONSOLIDATED MILL. By J. W. HUTCHINSON. Pamphlet, 16 pages, illustrated. San Francisco: *Mining and Scientific Press*; London: *The Mining Magazine*. Price 1s.

"In the summer of 1911, a series of articles by J. W. Hutchinson was published in the *Mining and Scientific Press*, describing the metallurgical practice at the Goldfield Consolidated, Nevada. Since then, important modifications have been made in the method of treating the concentrate, and particulars were given by Mr. Hutchinson in an article that appeared in the issues of January 25 and February 1 of this year. The new method consists of a combination of cyaniding raw and roasted concentrate. The rich concentrate coming from the Deister tables is cyanided raw, whereby an extraction of 85% of the gold is obtained, leaving gold worth $25 in each ton of tailing. The tailing is then dried, roasted, washed for copper, and cyanided. Mr. Hutchinson gives the reasons for the adoption of this method, and full details of the process. These articles are now reprinted in pamphlet form and constitute a valuable addition to the literature of gold metallurgy."—*Mining Magazine*, May, 1913. (A. R.)

Abstracts of Patent Applications.

(C.) 375/12. Albert Henry Guy. An improvement in bearings for shaft rollers, or other mechanical haulages and the like with the necessary fixings thereof. 17.7.12.

This application relates to rollers used in incline shafts of mines and on the tracks of mechanical haulages for carrying the hoisting or haulage rope. The particular feature of the invention consists in providing the roller with resilient bearings. This is accomplished by the use of springs which are placed underneath the bottom half of the bearing bush of each end of the roller spindle.

(C.) 459/12. Alfred George Newkey Burden. Improvements in ore feeders for stamp mills. 26.8.12.

This application refers to an ore feeder comprising rollers carrying a short conveyor belt, and refers principally to provision of means for preventing excessive sag of the belt by automatically maintaining a constant tension of sufficient magnitude to admit of the belt being carried by the end rollers only, and to ensure the requisite amount of friction of the belt and periphery of the driving roller. This object is accomplished by the provision of either springs with tension adjustable by means of screws and nuts, or by means of weights acting upon levers.

(C.) 464/12. Hans Nordrok (1), Gentil Prelle (2). Improvements in ore feeders. 28.8.12.

This application has reference to means for actuating ore feeders of the revolving drum type and comprises a flexible steel band less in length than the periphery of the drum and acting in the same manner as an ordinary band brake. The drum is revolved by means of a cord connected to a tappet lever so that when the tappet descends this cord tightens the steel band which then grips the drum and moves it forward. The steel band is brought into its original position again by means of springs. The application also includes provision of ball bearings for supporting the drum.

(C.) 465/12. Robert Rodger. Improvements in mills or apparatus for reducing ore or other materials. 28.8.12.

This application covers improvements in apparatus for reducing ore or other material, comprising superimposed sets of crushing rolls with corresponding plates, acting successively on the material to be reduced, the material falling from one set of rolls to those underneath.

(C.) 474/12. Wilhelm Mauss. Improvements in mountings for percussive coal cutters and the like. 2.9.12.

This application covers a mounting for percussive coal cutters in which the percussive machine is arranged to rotate about a vertical axis and is mounted radially to said axis. It consists of a frame carrying a rotatable member constructed to carry the percussive machine, said frame having front and rear feet at such distance from the axis of rotation as is necessary to enable the rear end of the percussive machine to clear it.

(C.) 3/13. Giuseppe Antonie Pietro Provay. Improvements in truck tipplers or means for discharging ore contents of trucks and the like vehicles. 2.1.13.

This application has reference to improvements in tipplers for the purpose of discharging the contents of trucks.

(C.) 106/13. John Ashford. Improvements in and relating to tube wells and the like. 28.2.13.

This application refers to specific construction of a filtration screen placed in bore-holes for water, to prevent sand and other sediment from being drawn into the bore hole by pump suction. The application embodies 19 claims, giving various modifications of construction. In general, however, the construction claimed embodied a longitudinally slotted pipe construction, over which is wound in helical fashion a wire so placed as to give any desired width of filtration opening.

(C.) 208/13. Matthew Arnold. Improvements in ore pulverising machines. 8.5.13.

This patent refers to a pulverising machine formed of sufficiently strong material and parts so hardened as to decrease the wear of pulverising zone to a very small per centage, and also producing pulverised ore of any size through proper adjustment of machine, saving more gold, and doing twice the work with less power and less water than in the old way with stamp mills.

(C.) 258/13. Alfred Edward Barker. Improvements in bearings for conveyor belt idlers and the like. 5 6.13.

This application refers to ball-bearing idlers for conveyor belts and is for the provision of an enclosed space for the lubricant, so that the whole body of the idler is not used for this purpose.

(C.) 284/13. Nicholas Henrie Marie Dekker. Improvements in or relating to electrolytes for use in electro-metallurgy. 19.6.13.

This patent refers to improvements in electrolytic processes whereby either metals or metallic oxides are obtained.

The governing feature of this invention consists in the use as electrolyte of baths containing a salt of the metal to be obtained and containing no other water than the water of crystallisation of the salts forming the bath; thus avoiding the formation of oxygen and hydrogen resulting from the decomposition of water in the ordinary processes, which factor causes secondary reactions affecting unfavourably the operation.

The process can be applied to the recovery of metals from their salts, the recovery of metals by refining mattes of impure metals and for the formation of metallic oxides.

The patent specification deals in detail with the different ways employed to obtain the results above mentioned.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 317/13. John William Cobb. A process for the extraction of ammonia and sulphur compounds from gas. 3.7.13.

(C.) 318/13. Ray C. Newhouse. Improvements in roll crushers. 4.7.13.

(P.) 319 13. Johannes Frederick Janse van Rensburg. Improved respirating apparatus with pulsating reservoir. 4.7.13.

(P.) 320/13. John Roger Walters. Steel rope slip roller. 4.7.13.

(P.) 321/13. Erik Gustaf Hedman. Improvements in knife edge bearings for weighing apparatus. 4.7.13.

(C.) 322/13. F. W. Offermeier (1), J. A. J. Stuy (2). Improved nut with spring washer. 5.7.13.

(P.) 323/13. Thomas S. McLaren. Electrothermic hot air incubator. 9.7.13.

(C.) 326/13. Francis Edward Matthews (1), Henry James Wheeler Bliss (2), Harry Montagu Elder (3). Improvements in the manufacture of unsaturated hydrocarbons. 10.7.13.

(P.) 327/13. Alfred William Torkington. Improvements in or relating to tyres and in the manufacture of the same. 10.7.13.

(P.) 329/13. George Alfred James (1), Electric Amalgamator Company (2). Method and apparatus for recovering metals. 10.7.13.

(C.) 330/13. Frederic Lewis Nathan (1), Wm Rintoul (2), Frank Baker (3). Improvements relating to explosives. 10.7.13.

(P.) 331/13 John Pountney Udal. Improvements in the manufacture of nitro-glycerine 11.7.13.

(C.) 333/13. Ernest Kilburn Scott. An improved furnace for the fixation of nitrogen. 11.7.13.

(C.) 334/13. Edward May Munro (1), The R.E.T. Construction Co., Ltd. (2). Improvements in or relating to trolleys for electric traction systems. 11.7.13.

(C.) 335/13. Sandycroft, Ltd. (1), Thomas Murthwaite Dutton (2). Improvements in and connected with electric controllers. 11.7.13.

(P.) 336/13. Josef Carl Steinebach. Improvements in devices for automatically stopping electric hoists. 12.7.13.

(P.) 338/13. Wm. Blane Dodds (1), David Charles Bowen (2). Improvements in linings for the ends of tube mills and the like. 16.7.13.

(P.) 339/13. Gaston Jacquier. Improvements in the precipitation of metals from solutions of their salts, and apparatus therefor. 16.7.13.

(P.) 340/13. Evan Williams. Improvements in means for collecting detritus and dust produced in the operation of rock drilling machines and the like. 16.7.13.

(P.) 341 13. Urlyn Clifton Tainton (1), Malcolm Foerster Lambe Ayme Aymard (2). Improvements in the electrolytic recovery of metals from their solutions and in apparatus therefor. 16.7 13.

(P.) 342/13. James Dunlop Wilson (1), Maurice Solvie Aaron (2). Improvement in safety catches for mine skips and cages, hoists, and the like. 17.7.13.

(C.) 343/13. Thomas Hawkins. Improvements in the manufacture of explosives of the sprengel class. 17.7.13.

(P.) 345/13. Alexander McNamara. Improvements in and in connection with rotary drills. 17.7.13.

(P.) 346/13. Percy Herbert Benjamin. An improved adjustable rim or emergency wheel for use with motor driven vehicles and the like. 17.7.13.

(P.) 347/13. Ingo Emmanuel Kohlmeyer. Improvements in and relating to rotary mills for reducing ore and other materials. 18 7.13.

(P.) 349/13. A. Macgregor Ritchie. A new combination forming a material for use in forming ceilings, covering partitions, and the like. 18 7 13.

(C.) 350/13. Alexander Purser. Improvements in valves. 19.7.13.

(P.) 352/13 George Harold Leeney. Improvements relating to ventilators for buildings. 21 7.13.

(C.) 353/13. Otto Bernard Blackwell (1), Gustavus Adolphus Anderegg (2). Improvements in methods of reducing telephonic disturbances. 21 7.13.

(P.) 354/13. Charles Lester. An improved valve for pumps and the like. 22.7.13.

(P.) 355/13. Walter Edward Letty. Safety shield to high tension overhead transmission towers. 22.7.13.

(P.) 358/13. George Stott. Improvements in rollers for mine shafts, haulage ways, and the like. 23.7.13.

(P.) 359/13. John Harbottle. Improvements in rope sheaves, rope pulleys, and the like. 24.7 13.

(P.) 361/13. Wm. George. Improved explosive cartridge. 25.7.13.

(C.) 362/13. Frederick Lamplough. Improvements in or relating to a process and apparatus for the conversion of heavy hydrocarbons into lighter hydrocarbons. 25.7.13.

(C.) 363/13. The British Cyanides Co., Ltd. (1), Edmund Charles Rossiter (2). Improvements in the manufacture of sodium alloys. 25.7.13.

(C.) 364/13. Société Générale des Nitrures (1), Paul Bunet (2). Improvements in the manufacture of aluminium nitride and other operations in electrical furnaces. 25.7.13.

(P.) 365/13. Phillip Arthur Whear. Improvements in work drills with detachable bits. 29 7.13.

(P.) 367/13. Edward Oscar Ord Christopherson. Improvements in bearings of truck wheel axles. 30.7.13.

(P.) 368/13. Lancelot Ussher. Improvements in cocks for water sprays and the like. 30.7 13.

(C.) 369.13. John Evans. An improved process for converting fine coal into a marketable fuel. 30.7.13.

(C.) 370/13. Neston Gabriel Auguste Marie Joseph Champy (1), Leon Paul Alexandre Germain Champy (2). Process of and relating to the preparation of porous material for filling receptacles intended for storing explosive gases. 31.7.13.

(C.) 371/13. Oscar Schluter. A process and apparatus for melting in the holder wires for the supporting frames of metal filaments or metal wire lamps. 31.7.13.

(C.) 372/13. Richard Friedrich Hermann Tröger (1), Dr. Ing. Erich Philippi. A switch gear for several conductors, machines, or apparatus to be connected to one and the same busbar, the said conductors or machines being switched in and out with the use of one and the same resistance. 31.7.13.

(C.) 373/13. Emmet Frank Galligan (1), Wm. Henry Galligan (2), Henay Enson Woodward (3). Improvements in dust collectors for rock drills. 2 8.13.

(C.) 374/13. Charles Wesley Jean. Improvements in presses. 2 8.13.

(P.) 375/13. Anthony William Richard Schieder. Improvements in apparatus for "lashing" or moving broken rock or other material in underground working of mines or the like. 2.8.13.

(P.) 376/13. Henry Lethbridge Templer. Improvements in rock drills with detachable bits. 5.8.13.

(P.) 377/13. Henry Wm. Rogers. A new water drill to prevent all dust from the drill. 5.8.13.

(C.) 378/13. Irving Langmuir. Improvements in incandescent electric lamps. 6.8.13.

(C.) 379/13. Wm. Augustus Hall. Process of desulphurizing ore. 8 8.13.

(C.) 380/13. Wm. Augustus Hall. Improvements in the art of desulphurizing ores. 8.8 13.

(C.) 381/13. Wm. Augustus Hall. Improvements in furnaces for decomposing pyrites and other metallic sulphides. 8.8.13.

(P.) 383/13. Wm. Miller. Safety brake. 8 8.13.

(P.) 384/13. Stephanus Venter. Improvements in apparatus for recovering precious stones and metals from alluvial and other matter. 9.8.13.

(P.) 386/13. J. Christian Reul. Chlor-petrol. 9.8.13.

(P.) 387/13. Lewis James Rossiter. Improvements in valves. 12.8.13.

(P.) 388/13. Wm. Holman James. Electrochemical treatment of metal bearing material.

(P.) 389/13. John Head. Improved hydraulic press for skins, forage and the like. 15 8 13.

(C.) 390/13. Dr. Graf Botho Schwerin. Improvements in the electro osmotic extraction of water from animal, vegetable or mineral substances, and in apparatus therefor. 16.8.13.

(C.) 391/13. Auguste Fernbaeh (1), Edward Halford Strange (2). Improvements in connection with fermentation processes for the production of acetone, and higher alcohols, from starch, sugars, and other carbohydrate materials. 16.8.13.

(P.) 392/13. James Dunlop Wilson (1), Maurice Solvie Aaron (2), Lewis Levy (3). Improvements in drills and detachable cutters therefore. 18.8.13.

(P.) 393/13. James Dunlop Wilson (1), Maurice Solvie Aaron (2). Improvements in means for cleaning the screws of rock drilling machine bars. 18.8.13.

(C.) 395/13. Louis Wm. Campbell. Improvements in electrical heating units and resistance coils and method of manufacturing the same. 21.8.13.

(C.) 397/13. James Dowson Jackson. Improved apparatus for separating solid matter from smoke. Improvement in and connected with apparatus for clearing smoke. 21.8.13.

(C.) 398/13. Harry Pauling. Improvements in the process of concentrating dilute nitric acid. 21.8.13.

(C.) 399/13. Sterling Telephone Electric Co., Ltd. (1), Francis George Bell (2). An improved contact breaker for electric bells and the like. 21.8.13.

(C.) 401/13. Constantin de Sedneff. Improvements in electrical accumulators. 21.8.13.

(C.) 403/13. Alphonse Emile Verge. Improved gelatine explosive. 22.8.13.

(C.) 404/13. Alphonse Emile Verge. Manufacture of derivates of toluene. 23.8.13.

(P.) 405/13. James Esson. An improved carburettor for internal combustion engines. 23 8.13.

(P.) 407/13. Gaston Jaequier. A new process of detinning. 25.8.13.

(P.) 408/13. Harold Austin Lewis. Improvements in spray devices for rock drilling machines. 25.8.13.

(P.) 409/13. Joseph Foulkes Williams. An improved electrical fuse lighter. 25.8 13.

(P.) 410/13. Sigismund Banman Schlam. Safety link for preventing overwinding of mining cages or skips. 26.8.13.

(C.) 411/13. Thomas George Johns (1), Albert Richard Bannister (2). Improved variable power transmission gear. 26.8.13.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

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Vol. XIV. SEPTEMBER, 1913. No. 3

Proceedings
AT
Ordinary General Meeting, September 20th, 1913.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, September 20th, 1913, Mr. Alex. Richardson (President) in the chair. There were also present:

37 Members: Messrs. H. A. White, T. Donaldson, E. Pam, C. Toombs, F. W. Watson, John Watson, Prof. J. A. Wilkinson, R. G. Bevington, Dr. W. A. Caldecott, A. F. Crosse, W. Cullen, W. R. Dowling, A. McA. Johnston, Dr. J. Moir, J. R. Williams (Members of Council), A. S. Allam, J. Chilton, W. M. Coulter, R. Dures, L. L. Ellis, R. Gascoyne, H. Hill, W. E. John, G. A. Lawson, W. A. Ledingham, A. Macdonald, G. Melvill, J. M. Neill, F. D. Phillips, J. C. Phillips, A. Schwarz, W. B. Semple, A. L. Spoor, A. Thomas, W. E. Thorpe, J. H. Veasey, and A. J. Walton.

16 Associates and Students: Messrs. N. S. Arnot, E. G. Baskett, G. J. V. Clarence, O. A. Gerber, W. Human, R. W. Irwin, S. E. Willows-Munro, D. Nicholas, O. P. Powell, R. Ramsay, H. Rusden, B C. Smith, H. Stadler, I. R. Trollip, R. R. Vail, and H. Ward.

9 Visitors, including Dr. R. A. Lehfeldt, and Fred. Rowland (Secretary).

NEW MEMBERS.

Dr. W. A. Caldecott and Dr. J. Moir were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected, as follows :—

Martin, John Stanley, New Modderfontein G.M. Co., Ltd., P. O. Box 25, Benoni. Assayer.

Gross, Leon Ansell, B.Sc., Knights Deep, Ltd., P. O. Box 143, Germiston. Metallurgical Chemist.

The Secretary announced that since the last meeting of the Society the following Associates and Student had been admitted by the Council :

As Associates—

Blenkinsop, George Henry, M.Inst.M.M., P. O. Box 121, Johannesburg. Consulting Mining Engineer and Geologist.

Gifford, Arthur, Mysore G. M. Co., Ltd., Marikuppam, S. India. Mining Engineer.

Jensen, Eiler, Great Fingall Consolidated. Ltd., Day Dawn, Western Australia. Metallurgist.

Keough, Joseph. Messina (Transvaal) Development Co., Ltd., Grenfell Camp, Messina. Mine Overseer.

Scott, Bertram John. Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Mill Manager.

As Student—

Smith, Benjamin Charles. Roodepoort United Main Reef G M. Co., Ltd., P. O. Box 152, Roodepoort. Plate House Assistant.

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

General Business.

The President : I have pleasure in drawing your attention to the fact that there are no less than eight Past-Presidents with us to-night. The continued and active interest which those who have occupied the chair show in the work of the Society is a good augury for its future.

AMBULANCE COMPETITIONS AND THE RAND MUTUAL ASSURANCE CO., LTD.

The Secretary : I have received a letter from Mr. J. G. Bilbrough, the Managing Secretary of the Rand Mutual Assurance Company, Ltd., which he wishes to have brought to the notice of those who may be taking part in ambulance competitions :—

"I am directed to advise Members of the Rand Mutual Assurance Co., Ltd., that this Company is prepared to hold covered, in terms of its Policies, those employes of Members who may take part in Ambulance Competitions, and should any such employe be injured by an accident arising out of and during the course of such a competition then this Company will pay such employes claim, as though the accident had arisen out of and in the course of such employes work for his employer, provided always that the following conditions are observed:—

1. The competition shall be under the auspices of an Association duly recognised by the Minister of Mines in terms of Art. 277 (3) of the Mines and Works Regulations; such as the St. John Ambulance Association or the South African Red Cross Society.
2. This Company must be duly advised of particulars of each competition, viz.:—
    - (a) Where the competition is to take place.
    - (b) Whether on surface or underground or both.
    - (c) The date of such competition.
    - (d) The names, occupations and wages of the employes of your Company who are taking part in the competition.
3. This Company to confirm, in writing, to the member concerned that it accepts the risk in regard to the particulars given in terms of the above (Condition 2).
4. No risk will be accepted by this Company in regard to injuries sustained by officials, judges or any persons other than members' employes actually taking part in such competitions.

In view of the fact that a good knowledge of ambulance work amongst members' employes will tend in the direction of reducing claims on this Company, and in order to encourage members' employes to take part in such useful work, my Board have decided to cover the aforesaid risks free of charge."

IMPROVED TAPS FOR BURETTES.

Prof. J. A. Wilkinson (*Member of Council*): I should like to bring to the notice of members a slight modification in the tap of an ordinary burette, which I have found in practice to obviate some of the difficulties formerly experienced with the ordinary kind. If in titrations one holds the handle of the tap round the burette with the left hand, whilst stirring with the right hand, the tap is very liable to leak. Also in carrying the burette about a laboratory, it is generally held horizontally and more often than not with the handle of the tap pointing downwards, with the result that the tap falls out and is broken. The modification designed to overcome these unwelcome experiences is to prolong the tap through the hole and drill a small hole through it at right angles to the axis of the tap. Through this hole insert a small piece of wood—a match splint serves the purpose—so that the wood touches the burette and so prevents displacement of the tap. Another device is to groove the prolongation and insert a rubber band in the groove. This method, I believe was used first in America. It exists on the taps of the Levy air-analysis apparatus.

Mr. H. A. White (*Vice-President*): I would like to ask Prof. Wilkinson how he drills that hole?

Prof. J. A. Wilkinson: In reply, I may say that ordinary burette taps are not made long enough to permit of a hole being drilled, and so I have not attempted drilling. The burettes handed round were supplied to my order with the special taps ready for use by Messrs. Baird & Tatlock, of Hatton Garden, London, but I am sure any chemical supply firm would make them. It should be borne in mind that here in South Africa there are no manufacturing or supply firms of this character and if a tap gets broken it is practically speaking an impossibility to procure another, so that the burette is almost useless.

THE DETERMINATION OF THE ACIDITY OR ALKALINITY OF WATERS—A STUDY IN INDICATORS.

By Jas. Moir, M.A., D.Sc. (Past-President).

The object of this small paper is to call attention to the fact that very misleading conclusions may be drawn as to the acidity of mine water or the alkalinity of town water, if the titrations are done with the wrong class of indicator. The table, which is the essence of this paper, shows that the town water can be shown to be either acid or alkaline according to the indicator used, and that the mine water, which really contains about 65 to 70 parts of free sulphuric acid per 100,000, can be made to show anything from *nil* to over 100 parts of H_2SO_4 according as the indicator is an arylamino-dye (*e.g.*, tropaeoline OO) or a phenol (*e.g.*, thymolphthalein).

The table explains itself, as the indicators are arranged roughly in order of strength. In both cases large samples of the water were taken so as to ensure uniformity of composition; and in the case of the town water great care was taken to collect it slowly so that dissolved CO_2 (which is titrated as an acid by half of the indicators) should not be lost by agitation.

Another mine-water of different type (Crown Mines, Shaft 3) was tried. It contained practically no ferrous iron, but plenty of ferric sulphate. 100 cc. used 17 to 17½ cc. $N/5$ soda

Indicator.	NEW GOCH MINE WATER, ACID (Untreated) Contains ferric, ferrous, aluminium, calcium and magnesium sulphates with free sulphuric acid.	JOHANNESBURG "ALKALINE" TAP WATER (Plein Street Laboratory). Contains calcium and magnesium bicarbonate with a little sulphate and some free carbonic acid
	Cc. *N*/5 Sodium Hydrate per 100 cc. Water.	
		Water acid to first 12 indicators
1 Alizarine-yellow R	abt. 15 to pink : intermediate orange at 12	abt. 4 to pink : orange at 1
2 Tropaeoline O	abt. 13 ,, ,, ,, at 11	3·5 to 4 to pink ; fairly sharp
3 Thymolphthalein (*a*) little	10·8 } to permanent green due to blue mixed with $Fe(OH)_3$	1·0 } to faint permanent blue
(*b*) much	10·1	0·5
4 Benzaurine (phenolbenzeïne)	9·5 to 10·5 (unsharp)	3·5 to 4 for violet : some deepening of orange at 1·5
5 Tetrabromphenolphthalein	6 to 9 (unsharp)	Trace to 0·1 (sharp to faint violet)
6 Phenolphthalein	8·0 permanent : temporary pink much earlier	0·2 to permanent pink : rather less with excess
7 4-nitrobenzeneazoguaiacol	6·9 (sharp)	0·3 to 0·5 (fairly sharp)
8 Litmus (to blue)*	6 to 7 (uncertain: precipitate at 5·5)	Indefinite trace
9 Lacmoid (to blue)*	abt. 5·9 (fairly sharp)	Quite indefinite up to 2 cc.
10 4′-acetamino-4-oxyazobenzene	6·4 (sharp)	Trace to 0·1, from green-yellow to orange
11 4′ oxyazobenzene	6·2 (fairly sharp)	0·2 to 0·3
12 4′-amino-4 oxyazobenzene*	Indefinite excess to alkaline colour	abt. 0·5 to alkaline orange (light green in neutral stage)
13 Orange I	Not tried	Practically neutral
		Water alkaline to remaining indicators cc. *N*/5 HCl per 100 cc. water
14 *α*-naphtholphthalein	6·75 (sharp)	Trace to 0·05
15 4′-nitro-4 oxyazobenzene	6·4 (at point of precipitation)	0·1 (sharp)
16 Rosolic acid (aurine)	6·6 (sharp, but takes time to finish)	0·6 to 0·9 (vague)
17 Alizarine	7 to 8 (vague : spoilt by precipitation)	abt. 0·8 (to yellow, but unsharp)
18 Litmus (to red)*	——	abt. 1·3 (useless)
19 Lacmoid (to red)*	——	1·5 (sharp)
10 Methylred	5·9 (to yellow, sharp)	1·6 (to red, sharp) (1·3 to salmon)
21 Phenacetoline	abt. 7·5 (to violet, fairly sharp)	1·3 to 1·6 (to yellow, unsharp)
22 Paranitrophenol	abt. 2 (useless)	abt. 3 (useless)
23 Dianisidine-azo-dimethylaniline	3·5 to orange (fairly sharp)	1·9 to violet (1·5 intermediate)
24 Dimethylamino-azobenzene	2·1 (sharp)	1·95 (to pink, sharp): salmon at 1·7
25 Methyl orange	Indefinite alkaline point, 2·5 salmon	2·4 to pink, sharp : salmon at 1·6
26 Congo red	abt. 7 to red : 1·5 to brown (poor)	abt. 10 to blue : brown between 1·5 and 3
27 4′-amino 4 oxyazobenzene*	0·5 from acid to neutral	3 to excess to pink (useless)
28 Tropaeoline OO.	Trace	abt. 10 to violet : deepening at 4
29 Methylviolet	Water not acid to this	abt. 10 to green blue

*These indicators are markedly amphoteric and have a wide range of neutral shade, with separate end points for both acid and alkali.

with thymolphthalein ; 14·4 with phenolphthalein ; 12·5 with nitrobenzeneazoguaiacol ; the same with *α*-naphtholphthalein ; with methylred, 9·5 were required to the salmon shade, and 11·4 to the yellow ; whilst with methylorange, only 3 to 3·5 cc. were required to the intermediate tint. The true acidity of this water is about 115 parts per 100,000, which is the greatest I have met with.

It is obvious that no hard and fast rule can be laid down as to the proper indicator for giving the true acidity or alkalinity : all depends on the purpose for which the result is wanted. Thus, if the New Goch mine water is to be neutralised sufficiently to kill the free acid and leave the iron salts (say for the purpose of minimizing corrosion of iron and steel), the low values given by methylorange and the other strong indicators are correct, but if we wish to precipitate the iron and alumina, the intermediate values (5 to 7 cc. *N*/5 caustic soda per 100 cc. water) are the correct ones, and lacmoid is a suitable indicator : but if this water is to be neutralised for use in cyaniding, it must be made alkaline enough to decompose so-called neutral salts (*e.g.*, $MgSO_4$), and an indicator higher up on the list, *e.g.* phenolphthalein, must be used in titrating the water (this is also the proper kind of indicator for use in calculating the amount of soda required to prevent scale from boiler waters) : and finally, where *definite* alkalinity is required, as when the water is to be used in the sand-filling of stopes (the definite alkalinity being required to *ensure* that no cyanide is hydrolysed into free prussic acid), then the weakest class of indicator, *e.g.*, thymolphthalein, should be used. The same applies to Rand Water-Board water: although practically neutral [in the sense of having its hydrion concentration about the same as that of distilled water,*] yet owing to its CO_2 and dissolved salts, it counts as an acid towards cyanide solutions, as any of you can verify by adding a small quantity of pure KCN to say 500 cc. of tap water, when on titrating with $AgNO_3$, cyanide (as cyanidion) will be found to be absent, having been converted into HCN. This experiment will not succeed with *low-grade* sodium cyanide, since it contains free soda.

This difference between the indicators is of course well-known in other cases, but it becomes much accentuated in the case of these waters owing to the presence of quasi-neutral salts and CO_2. For comparison I append the result of titrating my own $N/5$ hydrochloric acid with $N/5$ soda (the latter made from sodium, and the former from pure acid and *fresh* distilled water): 100 cc. acid required:—

100·0 with methylorange (salmon, not red).
100·3 with methylred ,, ,,
100·8 with α-naphtholphthalein (yellow-green, not blue-green),
101·15 with phenolphthalein (pale violet).

Time was allowed in all cases for the reactions, which are not instantaneous, to be completed: 15 sec. suffices. Previous work on this point is given by M. Scholtz (*Journal Chemical Society*, abs. ii., 1905, 57), who gives however a difference of 8% between the results for methylorange and phenolphthalein: this might be explained by his having probably taken methylorange to its first change from red only, which certainly occurs while the solution is quite acid. This shows in any case how careful the operator has to be, both in standardising and in using standard solutions for acidimetry.

The true neutral-point, as you will have gathered, is rather difficult to fix. Common distilled water, which has been exposed to the air, is not neutral, for example, but slightly acid (owing to carbonic acid) as can be demonstrated with phenolphthalein previously made just alkaline. An artificial solution which is truly neutral can be made by mixing $M/10$ solutions (freshly boiled) of Na_2HPO_4 and NaH_2PO_4 in the proportion of 1 of the former to 3 or 4 of the latter: (between these limits the quantity of NaH_2PO_4 added makes scarcely any difference to the hydrogen-ion-concentration of the final mixture, which is almost exactly 10^{-7} as in 'conductivity-water'—the latter being a substance practically unmakeable in South Africa). Such a 'neutral' solution can be used for finding the true neutral colour of any indicator in any strength; and the results are rather surprising. For example neutral methylorange is quite yellow, whereas all analysts use the salmon tint in titration and therefore get results slightly on the acid side. It may be noted that the middle tint of difficult indicators such as litmus may be got by a device I invented about 1899 but have never exhibited. It consists of two parallel-walled glass cells of equal distance between the walls. These are filled with the litmus in common water previously made to about half the usual strength in another vessel and then equally divided between the cells. One cell is made strongly acid with 1 cc. strong HCl and the other treated with 1 cc. strong soda. They are then placed side by side, and the tint obtained by looking through both is obviously the same as that of a single solution exactly at its intermediate tint (with exactly 50% of each kind of ion). The solution to be titrated is then compared with the colour of the double cell, and either acid or alkali added until the colour matches exactly. Using litmus for example in this way one can get about as fair an approach to true neutrality as is possible in technical work. Naturally it does not work with methylorange or phenolphthalein, since neither of these passes through the true neutral-point ($[H^\cdot] = 10^{-7}$) during their change of colour. The method can be applied with litmus, lacmoid, methylred, rosolic acid, neutralred, alizarine and some others given in the middle of the last Table.

A rather pretty example of the different end-points given by different indicators may be demonstrated by means of a mixture of thymolphthalein, phenolphthalein, naphtholphthalein and *p*-nitrophenol with a little alkali. This is dark-blue and on blowing one's breath through it with a glass tube the CO_2 gradually decolorises these indicators in the order given and the solution becomes first inky-pink, then red, then dirty green, then greenish yellow, then straw coloured and finally almost colourless.

Last time I addressed the Society, Mr. F. W. Watson suggested that I should tabulate the indicators in such a way that their properties would be shown at a glance. This I have pleasure in doing, but I have to utter a warning against accepting as completely accurate the order in which I have placed them. The literature available here, even in our excellent Seymour Library, is scarcely equal to the strain of such an enquiry. Besides this, the indicators with an unsharp change cannot be accurately placed in the order. In this second table the strongest indicators are at the top, which is the reverse order to that of Table I.

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*The indicator α-naphtholphthalein, used in some excess, gives this true neutral point fairly closely.

TABLE GIVING THE APPROXIMATE ORDER OF 'STRENGTH' OF THE INDICATORS.

The first class requires mineral acid to effect the colour change, and the earlier members of this class are used for establishing definite (mineral) acidity : they are quite insensitive to organic acids (even to oxalic), and the first two require quite strong acid to change them :—

	Common Name.	*Chemical Constitution.*
1	Methyl Violet	Pentamethylrosaniline chloride.
2	Mauveine..	2-phenyl-3-tolusafranine chloride.
3	'Phenylamidoazobenzol'	4-anilino-azobenzene.
4	Tropaeolin OO (Orange IV.)	Sodium 4′-sulphonate of No. 3.
5	Metanil-yellow	Sodium 3′-sulphonate of No. 3.
6		4′ amino-4-oxyazobenzene (from neutral to acid stage).
7		Sodium 4-benzylamino-azobenzene-4′ sulphonate.
8		4-benzylaminoazobenzene.
9		Sodium 2-chlor-4-diethylamino-azobenzene-4′ sulphonate.
10	Congo-red (blue to brown)	Diphenyltetrazo-*bis*-naphthionic acid.
11	Butter yellow (Dimethylaminoazobenzene)	4-dimethylamino-azobenzene.
12	Methyl Orange (Helianthin, Orange III., Tropaeolin D)	Sodium 4′ sulphonate of foregoing.
13		Dianisidine-azo-dimethylaniline.
14	Chrysoïdine...	2-4-diamino-azobenzene.
15		Benzeneazonaphthylamine (1 : 4).
16		Sodium sulphonate of above

Those near the end of this list behave like litmus at its acid stage, and change colour at a hydrion-concentration of about 10^{-5} : we have (17) litmus (red) : also (18) benzopurpurine B (the 3-3′ dimethyl-derivative of Congo-red) : (19) fluoresceine (2-2′-oxo-derivative of phenolphthalein) : (20) iodeosine (tetraiodofluorescein), and (21) alizarine (yellow to orange) : all rather indefinite in behaviour.

The second class changes between $[H^{\cdot}] = 10^{-5}$ and $[H^{\cdot}] = 10^{-7}$. They thus practically indicate neutrality with a trifling error on the acid side.

	Common Name.	*Chemical Constitution.*
22	Methyl Red...	Sodium 2′-carboxylate of No. 11.
23	Alizarin sulphonic acid	Sodium 1 : 2-dioxyanthraquinone-4-sulphonate.
24	Lacmoid (red to violet)	(?) Dioxyphenoxazone.
25	Paranitrophenol (to faint yellow)	4-nitrophenol.
26	Cyanine (to decolorization)	(?) Amyl-diquinolylcarbinol iodide.
27	Luteol	3-chlor-2-oxy-αβ diphenylquinoxaline.
28	Gallein (first change)	Pyrogallol-phthalein.

The middle violet of litmus comes next, almost exactly at true neutrality, if the operator can only catch it : so with the first deepening of Congo-red : similar indefinite indicators of this group are added here for completeness : (29) phenacetoline, (30) cochineal, (31) hematoxyline.

The third class comprises those which go just over the neutral stage and are changed by alkali of minute strength, viz., $[OH']$ between 10^{-7} and 10^{-5}. These also indicate practical neutrality with a trifling error on the alkaline side :—

	Common Name.	*Chemical Constitution.*
32	Neutral Red	*N*-4′-dimethyl-4 : 4′-diamino-3-methylphenazine chloride.
33	Rosolic acid (Aurine, Coralline)	*bis*-4-oxyphenyl methylenequinone.
34		4′-nitro-4-oxyazobenzene.
35	α-naphtholphthalein...	*bis*-4-oxynaphthyl-phthalide.
36		4′-acetamino-4-oxyazobenzene.
37	Benzeneazophenol	4-oxyazobenzene.
38	Tropaeolin 0002 (Orange I.)	Sodium α-naphtholazosulphanilate.
39	Turmeric (Curcuma)...	

At this stage is reached the alkaline colour of litmus, lacmoid, alizarine, gallein (2nd stage) iodeosine and amino-oxyazobenzene, all rather indefinite and too alkaline for true neutrality ($[H^{\cdot}]$ between 10^{-8} and 10^{-9}).

The remaining indicators should only be used for establishing definite alkalinity : the final ones require centinormal or even decinormal alkali to change them :—

	Common Name.	*Chemical Constitution.*
40	Diazonitranilineguaiacol	4-nitrobenzeneazoguaiacol.
41	Phenolphthalein	*bis*-4-oxyphenyl-phthalide.
42	Curcumine W.	Ammonium azoxystilbene-disulphonate.
43	Benzaurine (phenolbenzeïne)...	Phenyl-4-oxyphenyl-methylenequinone.
44	Naphtholbenzeïne	Phenyl-4-oxynaphthyl-methylene-α-naphthaquinone.

Table II.—*(Continued).*

45	Tetrabromophenolphthalein.	
46	Thymolphthalein	Diisopropyldimethyl-phenolphthalein.
47	Poirrier-blue C4B	(?) Triphenylrosaniline-monosulphonic acid*.
48	Chrysamine G	Sodium diphenyltetrazodisalicylate.
49	Alizarin-yellow GG Prager	Sodium nitrobenzene-azo-β-resorcylate.
50	Ponceau 4GB (Brilliant orange, Crocein orange)	Sodium benzeneazo-β-naphthol-6-sulphonate.
51	Tropaeoline 0001 (Orange II.)	Sodium β-naphtholazo-sulphanilate.
52	Fast Red	Sodium ,, -naphthionate.
53	Alizarin-yellow R	Sodium 4-nitrobenzenazo-salicylate.
54	Tropaeoline O (Chrysoin, resorcin-yellow) ...	Sodium resorcinolazo-sulphanilate.

Finally comes the last change of gallein, alizarine and methyl violet in strong alkali.

References I have used either direct or in abstract are: Handa (*Berichte, 1909*, 3179); Salm (*Z. Elekch, 1904*, 341): also Scholtz (l.c. 549): Salessky (l.c. 204): Friedenthal (l.c. 113): Schoorl (*J. C. S.*, abs. ii., *1907*, 389) and Sörensen (l.c.) above all.

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* According to Gohn: but this is "soluble aniline-blue," which is different from Poirrier's.

Mr. A. McArthur Johnston (*Past-President*): It is with great pleasure that I propose a hearty vote of thanks to Dr. Moir for his estimable paper. None of us would wish to criticise his paper to-night, but I am certain you will all agree with me that our thanks are due to Dr. Moir for writing these notes and presenting them to us in such an attractive form.

Mr. A. F. Crosse (*Past-President*): I would like to second the proposition. There is a great deal more behind this question of the examination of waters in South Africa in ways other than those generally considered. It will be the opening of an interesting question which Dr. Moir has raised to-night; but I think he has rather complicated it by using so many indicators.

Prof. J. A. Wilkinson (*Member of Council*): I should like to ask Dr. Moir what he means by "litmus." Is it the substance obtained by adding water to the commercial article and either decanting or filtering? or has he specially purified this according to Mohr's method? As is well known, the latter indicator is much more sensitive than the former.

Dr. J. Moir (*Past-President*): The answer is that, in my own experiments, common litmus is meant, and in the table the other substance azolitmin is often meant, because certain authorities give the hydrogen-concentration of its changes.

NOTE ON THE QUANTITATIVE DETERMINATION OF NITROUS FUMES IN FIRING (CHEESA) STICKS.

(Read at April Meeting, 1913.)

By Dr. L. Heymann (Associate).

REPLY TO DISCUSSION.

Dr. L. Heymann *(Associate)*: I am afraid Mr. Donaldson has quite missed the point of my reference to the standard of NO_2 proposed by the Government Mining Engineer. The regulations which prohibited the use of 'cheesa' or firing-sticks which give off any NO_2 on burning have been altered departmentally to a maximum of 0·05 gm. NO_2 per gm. of substance. In view of the very poisonous nature of the nitrous fumes, the presence of such a maximum percentage of NO_2 must be regarded with the gravest apprehension.

The standard would seem to have been based upon the results of defective methods of determination, as my results show that 'cheesa' sticks liberating from one-fifth to one-tenth of this maximum are readily obtainable, and the present standard is therefore unnecessarily careless of the health of the workers. Under the conditions of my recorded determinations the samples burnt slowly and quietly without any spirting, the ash remaining in the crucible.

Since making the experiments described, I have come across some samples of 'cheesa' sticks containing chlorates, and others containing iron filings from which particles of ash were projected during burning. It is easy in such cases to prevent loss of ash by covering the crucible with a piece of platinum gauze or by other suitable means which will readily suggest themselves to the chemist. Dr. Moir may even be able to assist in such a contingency by describing his own method of securing the complete exclusion of floating particles of ash.

I have to characterise as unfounded the statement made by Dr. Moir that "Dr. Heymann's method has been already described as being in use by Mr. Mann." I have challenged Dr. Moir to produce his evidence for that statement, and I shall be glad if he will do so at an early date.

THE DETERMINATION OF GOLD IN THE PRESENCE OF IRIDIUM AND ALLIED METALS IN MATERIALS SUCH AS BLACK SAND.

Prize Papers by Mr. Jas. Gray, F.I.C., and Mr. Chris. Toombs (Members of Council).

(Published in July Journal, 1913.)

DISCUSSION.

The President: Since the last meeting the Council has received several requests from members, including Messrs. Toombs and Gray, to throw these papers open to discussion. In response to these requests the Council has pleasure in intimating that the papers are now open to discussion.

THE SAND-FILLING OF MINES.

By W. A. Caldecott (Past-President) and O. P. Powell (Member).

As it is now some years since the methods of sand filling of mines were discussed before this Society, and as considerable progress has been made in the interim, the present time appears appropriate for reviving the subject. Experience has shown the advantages of sand-filling to be so considerable on the Rand that little space will now be given to detailing them in view of their previous enumeration[*] by Mr. Edgar Pam and one of the authors (O. P. Powell).

The authors propose in the first place to briefly discuss certain general considerations relating to sand-filling, and then to describe the methods actually employed at the present time on certain mines and their cost. The main features involved in the process of sand-filling are the destruction of cyanide in the solution moistening the sand residue; the transport of material after cyanide treatment from the sand vats to the point of lowering into the mine; the method of lowering; and the deposition and drainage of the sand in the worked-out stopes.

Although in the early days of these fields untreated "tailings" were reported to be run into abandoned mine workings as a ready means of disposal, it is to the "Corner House"[†] that the credit must be given of first realising the advantages of systematically practising sand filling on the Rand, and the method introduced on certain mines, which is essentially the same as that long practised in Silesia, was fully described by Mr. Edgar Pam in a paper read before this Society on 18th June, 1910. By this method Mr. H. F. Marriott states in the annual report of the Central Mining and Investment Company that in 1912, on seven mines of this group, 984,650 tons of sand were lowered into worked-out stopes at a total cost of 1s. 3·36d. per ton. Briefly, this system consists in transporting by trucks or belts several hundred tons of sand residue to a large cemented bin or chamber at the top of a shaft, and when the chamber is filled sluicing the whole mass as a liquid pulp through a pipe into the mine. For a considerable time only sand residue from dumps was thus lowered, owing to fear of accident underground from poisonous fumes and drainage from fresh current residue, but experience has since shown that with due precautions current sand residue can be employed without inconvenience or danger, and is in fact far preferable. The utilization of dump residue usually involves the use of a considerable amount of lime to neutralize the sulphuric acid formed by the weathering of the pyrite, as well as the cost of loading and transporting sand from the dump to the point of lowering, whilst the usual expenditure on sending current sand residue to the dump is simultaneously incurred. The older dumps likewise contain a considerable amount of colloidal slime the presence of which is undesirable in the sand deposited underground. Another system than the above of transport and lowering of sand residue for mine filling has been installed at the Cinderella Consolidated Mine,[*] and can be operated in fine weather.

The system to be described in this paper was recommended[†] some time ago for installation at the Simmer & Jack by one of the authors (W. A. Caldecott) as being well suited to local conditions. and also the modified form subsequently installed at the Robinson Deep, on both of which mines it is in regular and economical use, as well as on various other mines.[‡] The essential features of the system as installed at the Simmer & Jack consist in mixing the sand residue, immediately after truck discharge from the vats, with water and a small amount of permanganate of potash solution; and pumping the mixture through pipes to de-watering diaphragm cone classifiers placed immediately above the borehole or other point of lowering, down which the thick sandy underflow continuously descends. The fluid cone overflow gravitates or is pumped back to the mixing box beside the pump into which the residue is dumped, thus completing its circuit and serving to transport more residue. The underflow falling down the borehole into the mine is then conveyed by launder to the stope to be filled where it speedily drains, leaving a solid mass of sand behind. It is a fortunate circumstance that in lowering sand pulp by means of a borehole any accidental filling of the latter with sand is of temporary inconvenience only, as the turning into the top of the borehole of a small stream of water in the evening will result in a clearance by the following morning.

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[*] See this *Journal* p. 431, June, 1910, Vol. X., and p. 78, August, 1910, Vol. XI.

[†] The mines controlled by the Central Mining and Investment Corporation, Ltd. and the Rand Mines, Ltd., have their head offices in the Corner House, Johannesburg.

[*] See *Bulletin 97 of Inst. of Min. and Met.*, 10th October 1912.

[†] See article by S. H. Pearce in the *The Cocopan*, p. 30, January, 1911.

[‡] See description of sand-filling installation at Wit. Deep in *South African Mining Journal*, p. 707, 1st February, 1913; also Simmer & Jack Annual Report of 30th June, 1912, and Robinson Deep Quarterly Report of 30th June, 1913.

In the free fall of thick pulp down a borehole the cross-sectional area of the falling stream rapidly decreases as its velocity rises under the influence of gravity, with the result that in, say, a 7 in. borehole the pulp only comes in contact with the side at intervals where the borehole is deflected from the vertical. The pulp hence reaches the bottom at a very high velocity, and draws down a distinct current of air with it. In a borehole of small diameter the free fall is liable to be interrupted by frictional contact with the side, and the likelihood of the borehole filling up with sand is greatly increased.

In brief it will be seen that this system is applied to current residue; that the residue is transported from the sand plant to the lowering point as a flowing pulp and not by trucks, the water performing a circuit; that the lowering proceeds continuously instead of intermittently, and is usually performed by passing a thick pulp through a borehole instead of a more fluid pulp through pipes, thus avoiding wear of the latter; and that distribution of the sand underground is carried out in open launders instead of in pipes under pressure. One portion of the Simmer & Jack mine requiring filling happens to be nearly below the sand plant, so an 8 in. borehole (500 ft. deep) has been put down near the residue car track, and a short steep tunnel from the mixing box to the borehole allows the residue there dumped to be carried by a small stream of water as a thick pulp into the mine, without the need for pumping or dewatering. This modification, however, has been more fully developed on the Robinson Deep and will be discussed later in greater detail.

The system of utilizing current sand residue as mine filling enables ore to be returned underground, minus nearly all its gold content, within a fortnight of hoisting, and it is a coincidence that two tons of ore occupying about 24 cu. ft. of space *in situ* in the mine yield about the same volume of sand for return underground, if the ore has been well crushed and classified into 50% by weight each of sand and slime.

To obtain full benefit from sand-filling it is essential that this operation should not be periodically suspended for days or weeks at any time [illegible] such conditions the surface dump equipment and [illegible] organisation must also be maintained at considerable cost to save hanging up the mill. To avoid this extra expenditure on dual systems of residue disposal, preparations should be made in any mine, where sand-filling is intended, for several depositing sites to be ready in advance and to be always available, so that accident or the necessity for removal of pillars or residual ore in one or more stopes, will not put even a temporary stoppage to the sending of sand underground.

The most desirable sand-pulp for mine filling is that which is lowered underground with a minimum of slime and water, and such a sand can be readily obtained by good classification prior to current cyanide treatment, for which it is in any case essential for the most profitable extraction of gold.* Sand sent underground with much slime tends to undergo a similar classification there as in a tailing dam, with the result that the slime is carried furthest from the point of entry into the stope and by lodging against the barrier interferes with free drainage. The higher the ratio of water to solids in the pulp entering the stope the more this effect is accentuated, with the result that the deposited sand is liable to form a water logged mass. The behaviour of water as an incompressible fluid under such conditions causes any subsequent vertical rock pressure, which it is the object of sand-filling to withstand, to be laterally transmitted against the barriers, frequently resulting in their giving way.

On the other hand, clean sand deposited without material classification or segregation behaves like a desirable uniform and permeable charge in a leaching vat, and not only allows the water conveying it as a pulp to readily escape, but also mine drainage water which may subsequently enter the stope. Well drained sand shows little tendency to transmit vertical into lateral pressure against the barrier, owing to the enormous friction between the sharp-edged particles, and as shown by the following table of tests carried out by Mr. J. A. Vaughan and published in p. 59 of the Government Mining Engineer's Annual Report for 1911, its compressibility is so slight that even at 6,500 ft. depth the contraction of a deposit 6 ft. deep cannot amount to more than 1 ft.† :—

"*Sand-filling Tests.—Compression Tests of Tailings.*

"A number of tests were made with the idea of arriving at the reduction of volume produced by compressing water-filled tailings. These were water borne into a cylinder of 8 in. dia., and allowed to dry for seven days. A closely fitting plunger was then forced into the cylinder very slowly by the testing machine, the bottom

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*See *S. A. Mining Journal*, 4th May, 1912.

†See also *S. A. Mining Journal*, p. 143, 5th October, 1912.

of the test cylinder being perforated with small holes to allow any surplus water to run off. The reduction of volume produced by various loads is tabulated below, viz.:—

Percentage Decrease of Original Volume.	Compressing Stress in Tons (2,000 lb.) p. sq. ft.
5 per cent.	30
10 ,,	120
15 ,,	350
17 ,,	540

"In these tables the frictional resistance of the plunger in the cylinder is neglected, it being too small to affect the results for any practical purpose."

"Note.—It was found that the tailings did not 'flow' out of a hole ⅜ in. dia. near the bottom of the cylinder during the whole period of test."

In an earlier discussion* of sand-filling by one of the authors (O. P. Powell) there is illustrated the transfer of sand residue to the pump by shovelling from the vats through the discharge doors into a launder down which water flowed. Whilst trial has shown this method to be practicable. yet the limited grade available for launders in most existing sand plants requires a much higher ratio of water to solid for the pulp to flow by gravity than is needed for subsequent pumping, and of course still more than is desirable for lowering underground. The same consideration applies to hydraulicking sand residue out of the vat itself by water-jets under pressure, in addition to the liability in this case to drive slime through the filter-cloth, and thus interfere with the cyanide treatment of subsequent charges. If discharge of the residue to the dump is necessary at times owing to underground conditions, it is of course essential that trucks or belts under the sand vats should not have their operation at any time interfered with by a launder installation.

The undesirability of excess of water in causing a non-uniform deposit of sand underground has already been referred to, but in addition when surface transport by pumping of sand as a pulp is practised the power consumption is increased by superfluous water, and both in pipes and in launders a more fluid pulp increases the velocity and the wear by abrasion.† A sand-pulp delivery pipe for sand-filling has been in use with thick pulp at the Simmer & Jack for upwards of three years without appreciable signs of wear. The rapid increase of the amount of drainage water underground with fluid pulp is clearly illustrated in a table by the late Mr. G. O. Smart, published on page 217 of our November (1910) *Journal*, where it is shown that nearly three times as much drainage water has to be dealt with when a pulp containing 50 % of moisture is lowered into the mine as with a pulp containing 30 % moisture.

In the transport of sand residue as pulp both on the surface and underground the water serves essentially as a transporting medium, and its ratio to solids should be kept as low as compatible with serving this purpose. A sand-pulp may be pumped by a centrifugal pump up a short vertical pipe with as low a ratio by weight as 1 of water to 1 of sand, but for pumping any distance on an incline it is usually necessary to employ a 2 to 1 ratio. Where the velocity of the sand-pulp in a pipe is insufficient to keep the heavier particles in suspension the pyritic particles are liable to settle, and periodical pumping of water through the pipe is necesssary to prevent choking from the gradual accumulation by classification of such heavy pyrite. As with the sand-solution pulp from sand filter tables, that portion of the delivery pipe on the down grade works well with a fall of 3·5 %, so that acting as a launder it may clear itself on stoppage of pumping. In a rising delivery pipe the vertical component of the pulp velocity should be greater than the rate of fall of the coarse pyritic particles so that they cannot settle. A horizontal delivery pipe is very undesirable, as a very high velocity of pulp is required to increase the friction between the water and solids sufficiently to transport the latter and prevent their settlement. As with launders, pipe grade and water ratio in pulp are within limits inversely proportional, and hence the steeper the grades practicable the less water need be used. In any case the coarseness of the sand residue is also a factor, since the finer the particles the lower the velocity or pipe grade required. When possible a vertical rising pipe from the pump, with a uniform down-grade from its highest point constitutes a satisfactory arrangement, as does also a steep up-grade to the point of delivery. It is of course essential that there should be no pockets or flats in a delivery pipe-line, as any unfavourable grades effect the whole pulp transport operation.

The launder grade required for the transport of sand pulp is dependent not merely upon the size of the sand particles and the ratio of water to solids, but also upon the volume of the pulp and the internal shape and roughness or otherwise of the launder, whilst the head of a launder and any curves require a higher grade than the average. In general, however, as regards launders for thick pulp, an ordinary sand pulp containing

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*See this *Journal*, p. 77, August, 1910, Vol. XI.

†See paper by Mr. Victor Viannay in *Bull.* May, 1913, of Societe de l' Industrie Minerale, extracted in *S. A. Mining Journal*, 6th Sept., 1913.

30% moisture will flow slowly though without difficulty on a 30% grade (about 17°) launder, with 40% moisture on a 20% grade (about 11°), and with 60% moisture on a 10% grade (about 6°).

The rate of fall of particles in a tube or pipe is a somewhat complicated problem,[*] but assuming the rate of fall in water of a particle of pyrite 0·01 in. in dia. to be 0·3 ft. per sec., an upward grade of pipe of more than 5% with a pulp-velocity of 6 ft. per sec. would be required to transport it in suspension.[†] Further researches on this subject would be desirable, however, as the rate of fall of particles in a pipe appears to differ from the rate in a large vessel, and the higher specific gravity of pulp containing particles in suspension may also have an influence on the rate of settlement, as shown by the behaviour of a hydrometer in slime pulp.[‡]

The water ratio in pulp, besides determining the size of pumps and piping and power consumption, also affects the number of dewatering cone classifiers employed, since the capacity of these is limited to a certain volume of pulp entering them, as overflow of sand should not occur. With an inflow pulp having a 2 to 1 water ratio, a diaphragm cone (8 ft. in dia. × 10 ft. deep)[§] will deliver 10 tons to 12 tons of sand as a thick (30% moisture) pulp underflow per hour, if slime has been adequately removed from the sand before its cyanide treatment. If, however, the sand residue contains much colloidal slime so as to form a viscous deposit in the cone, the capacity of the above classifier may be reduced by one-third or more. It will depend upon the relative positions of the point of lowering and the pumping station whether the overflow from the cones at the former will gravitate back to the latter thus completing its circuit, or whether a return of the overflow by pumping is necessary. In the latter case any additional water necessitated by unfavourable pipe-line grades constitutes a constant source of expense. If conditions are favourable the dewatering cones may be placed in a small excavation above the borehole, thus saving pumping head, but if the return water requires re-elevation in any case, there is not so much gained by installing cones having their inflow at ground level. When only a limited area underground is to be filled from one lowering point the cones may be supported by a substructure bolted together for facilitating transfer to another point later. In addition to removing from the pulp for re-use water which was necessary for the surface transport of the sand but undesirable to send underground, the dewatering cones in the system described, operated nearly full of sand, serve the purpose of giving time for the permanganate to act before the sand enters the mine, and also neutralize the effect of any temporary fluctuations in the addition of permanganate or operation of pumps.

The destruction of cyanide in residual sand has been discussed at length before this Society, but the method now generally employed[*] is that suggested by Mr. H. A. White of converting it into a cyanate by means of potassium permanganate, and in this regard the necessity may be emphasised of keeping the residue pulp alkaline so as to facilitate the oxidising effect of the permanganate. The following is the equation formulated by Mr. White:—

$$2\ KMnO_4 + 3\ KCN = 2\ MnO_2 + K_2O + 3\ KCNO\cdot$$

Assuming the sand residue to contain 12% of moisture and the cone underflow entering the mine to contain 30% moisture, 0·292 tons of water will be withdrawn from the surface transport circuit for each ton of sand lowered. It is hence necessary to continuously add this amount of "make-up water" to the mixing box, and portion of this water is conveniently allowed to enter and overflow a barrel containing permanganate of potash which it dissolves and carries into the pulp. The permanganate crystals are best added to the barrel in small amounts at intervals for a given number of truck-loads of residue. The consumption of permanganate may vary from 0·04 lb. to 0·08 lb., equivalent to about one farthing to one halfpenny per ton of sand residue lowered.

Whilst the free alkaline cyanide as such ceases to exist in presence of free acid, hydrocyanic acid being formed, this latter compound is of course more dangerous than alkaline cyanide owing to its volatility, and a more radical transformation of the cyanogen radical is needed than acidulation only.

According to the Government mining regulations no solution containing more than 0·005% total cyanide may be sent underground, but with due care one-fifth of this amount, or 0·001% can be employed as a standard. In this connection the amount and strength of cyanide solution moistening the residue when discharged is of course an important factor; clean freely leachable sand charges are essential for the best results with a treatment finishing with a thorough wash of precipitated slime solution, so that the last draining may be very low in cyanide content and the residue moisture likewise a minimum.

The following recent average results of tests during July and August (1913) at the Simmer & Jack, which are regularly made at the Consolidated Gold Fields Laboratory upon samples taken not less than twice weekly, illustrate the efficiency of cyanide destruction. The amount of permanganate used is regulated in accordance with these routine tests, being increased when the cyanide contents of the solution rises, and *vice versa* :—

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[*] See Louis' "Dressing of Minerals," Chapter V., 1909.

[†] See Richards "Ore Dressing," Chapter XII., Vol. I., 1909.

[‡] See this *Journal*, Vol. II., p. 103, July, 1897; also G. Caetani, *Mining and Scientific Press*, p. 438, March 1913.

[§] See "Rand Metallurgical Practice," Vol. I., p. 154.

[*] See *Journal* of this Society, p. 15, Vol. XI., July 1910.

Source of Samples.	Total Cyanide (as KCN) %		Sulpho Cyanide (as KCNS) %		Alkalinity (as NaOH) %	
	July.	August	July	August.	July.	August.
De-watering cones overflow	0·0009	0·0008	0·0005	0·0006	0·0121	0·0134
De-watering ,, underflow	0·0008	0·0008	0·0006	0·0006	0·0111	0·0130
Sand-filled Stope drainings	Nil.	Nil to 0·0002	0·0009	0·0006	0·0052 alk. as NaOH to 0·0024 acid as H_2SO_4.	0·0039 alk. as NaOH to 0.0078 acid as H_2SO_4.

Although the sand pulp sent underground is alkaline, the weathering of the pyrite in the moist drained sand soon induces an acid reaction from the formation of sulphuric acid and acid sulphates. The traces of free cyanide left on lowering are converted into iron-cyanogen compounds, though the sulphocyanides present persist as such in the drainage water for a considerable period; the reactions involved have been discussed elsewhere.[*] It appears likely that one effect of sand-filling will be a steady increase in the acidity of mine water, as is the case with the drainage from sand residue dumps on the surface as weathering proceeds. In this regard a supply of cheap crude carbonate of soda would be very desirable as being more readily dissolved and utilised for neutralization purposes than ordinary burnt lime, and it is to be hoped that such material may in time become available.

The determination of the total cyanide present in solutions from sand-filling operations is a more delicate operation than the usual works tests for cyanide solution, on account of the necessity of accurately determining the usually very small amount of total cyanide present. The customary works method even when employing $N/20$ silver nitrate solution, can only be considered roughly approximate.

The total cyanide present in sand-filling or drainage solutions may exist as alkaline or sodium cyanide, double zinc alkali cyanide, and hydrocyanic acid.

The following methods[†] are employed by Mr. A. McA. Johnston for the above determination and others connected with sand-filling :—

Total Cyanide.—To 250 cc. of the solution from the sand about 5 cc. of a 10% caustic soda solution is added. When the solution collected as seepage from the sand underground is acid, it may be necessary to add a larger quantity than this, for an excess of alkali should be present.

To this alkaline solution, two drops of a dilute solution of ammonium sulphide are added, and the whole evaporated over a water bath. The sodium cyanide is thus converted into sulphocyanide. The residue is then digested with hot water and filtered. Acidify the filtered solution with sulphuric acid and add 15 cc. of a 5% ferric sulphate solution, when a red coloration is produced if sulphocyanide is present. Bring the volume to 100 cc. and colorimetrically test this in Nessler glasses against a standard solution of potassium sulphocyanide.

A standard solution of potassium sulphocyanide is made by dissolving about one gram of the salt in a litre of water and estimating the strength by titration with silver nitrate.

The amount found, less the potassium sulphocyanide originally present in the solution, gives the quantity of "total" cyanide.

No. of cc. $KCNS \times 65/97 \times 2/5 = KCN$ %.

Potassium Sulphocyanide.—To 50 cc. of the original solution is added sulphuric acid till acidity be obtained, and then 5 cc. of a 5% ferric sulphate solution. This is tested colorimetrically as above or in the usual way by titration against a decinormal silver nitrate solution.

Protective Alkali.—To 100 cc. of the solution add sufficient silver nitrate solution to give a slight turbidity, and then 5 cc. of a 5% solution of potassium ferrocyanide, and titrate against decinormal acid using phenolpthalein as indicator.

Hydrocyanic Acid in Air.—It is frequently advisable to determine whether any hydrocyanic acid gas is present in the air underground. This may be due to the decomposition of the cyanide salt by contact with acid mine water or with the carbonic acid of the air.

For this purpose the following test[*] evolved by Weehuizen and applied by Thiery may be used.

"An alkaline solution of phthalophenone "(phenolphthal*ine*)[†] is prepared by dissolving "½ gm. of phenolphthalein in 30 cc. of

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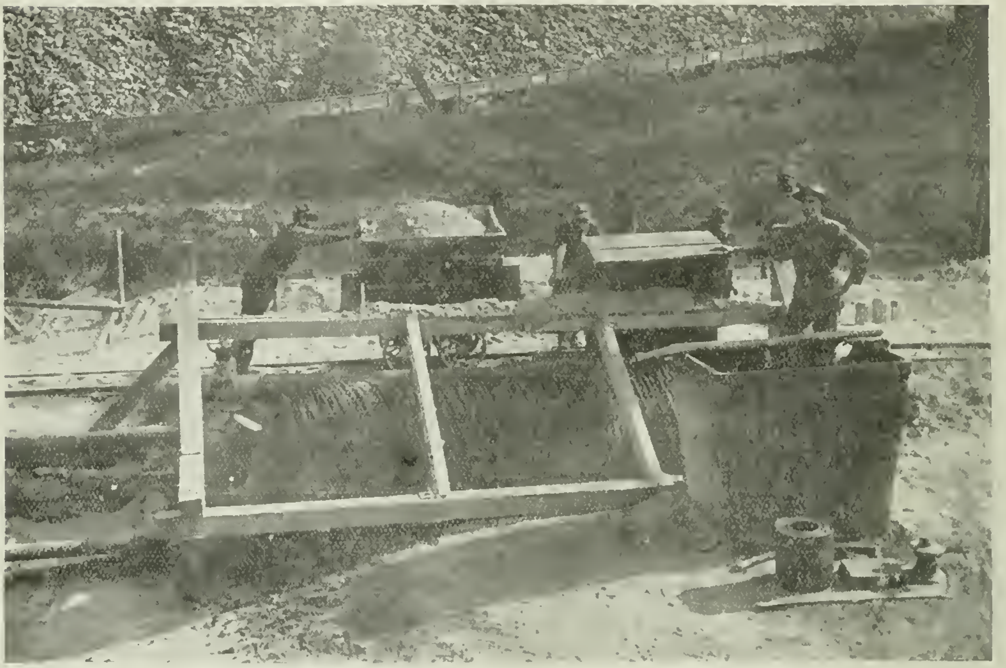
[*] See "Rand Metallurgical Practice," Vol. I. Chapter XI.

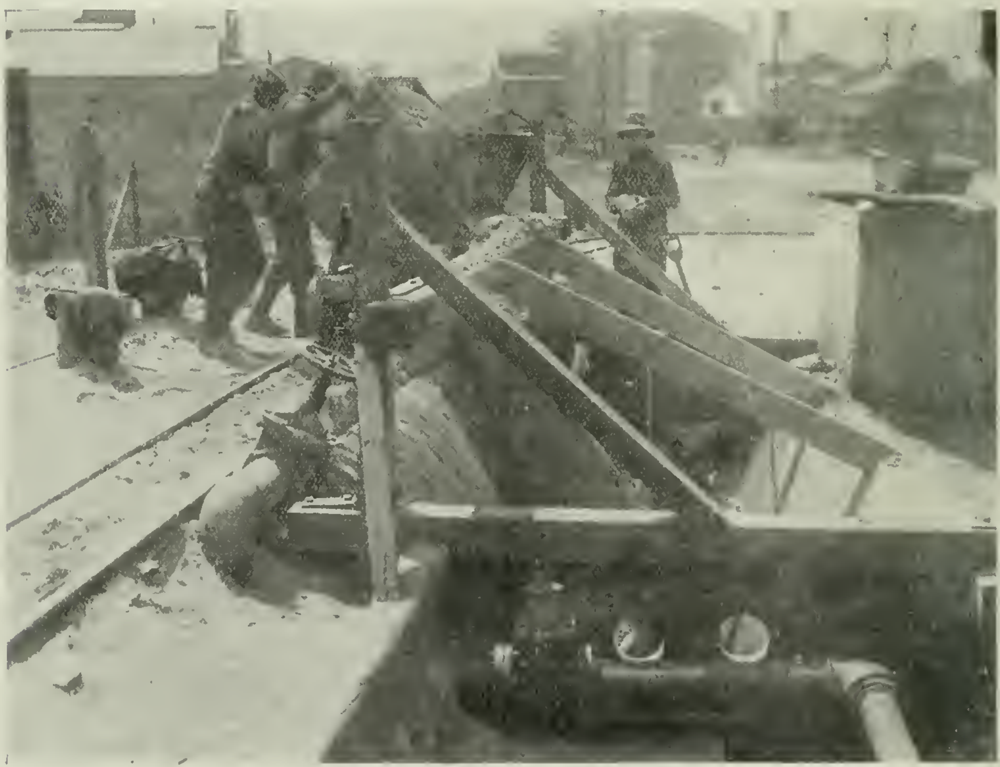
[†] See "Rand Metallurgical Practice," Vol. I., p. 324.

[*] See *Journal Soc. of Chem. Ind.*, p. 168, Feb. 28th, 1907.

[†] See also this *Journal*, Vol. VII., May 1907, p. 375.



I.—Truck of Sand Residue about to tip into Mixing Box at E, showing Return Water and the Permanganate Tank. Simmer & Jack Proprietary Mines, Ltd.



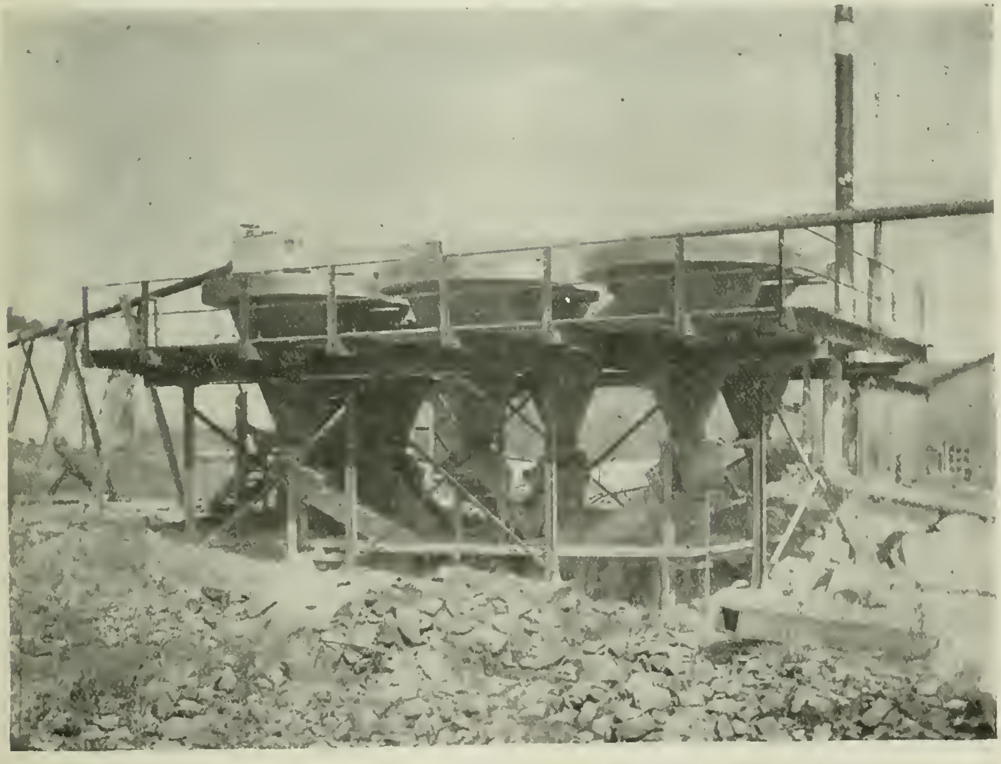
II.—Truck of Sand Residue Tipping and Mixing at Box E, showing Return Water and the Permanganate Tank. Simmer & Jack Proprietary Mines, Ltd.

"absolute alcohol; sufficient distilled water "is then added to produce a faint turbidity, "when 20 gm. of sodium hydroxide are "added. Aluminium dust is then added to "the red alkaline solution, a little at a time, till "the colour is discharged. The liquid is next "diluted to 150 cc. with distilled water, which "has been boiled and cooled, excluded from "contact with the air. The reagent is then "filtered."

A copper sulphate solution is made by dissolving 1 gm. of copper sulphate in two litres of distilled water.

Filter paper is cut into strips, and after moistening with the copper sulphate solution, dried, and placed in an air tight bottle. To test the air, one of these strips is wetted with the phthalophenone solution and waved about gently in the suspected atmosphere. Should hydrocyanic acid be present, the paper will be tinged red. This test is extremely delicate, and will detect 1 part of HCN in 2,000,000 parts of air.

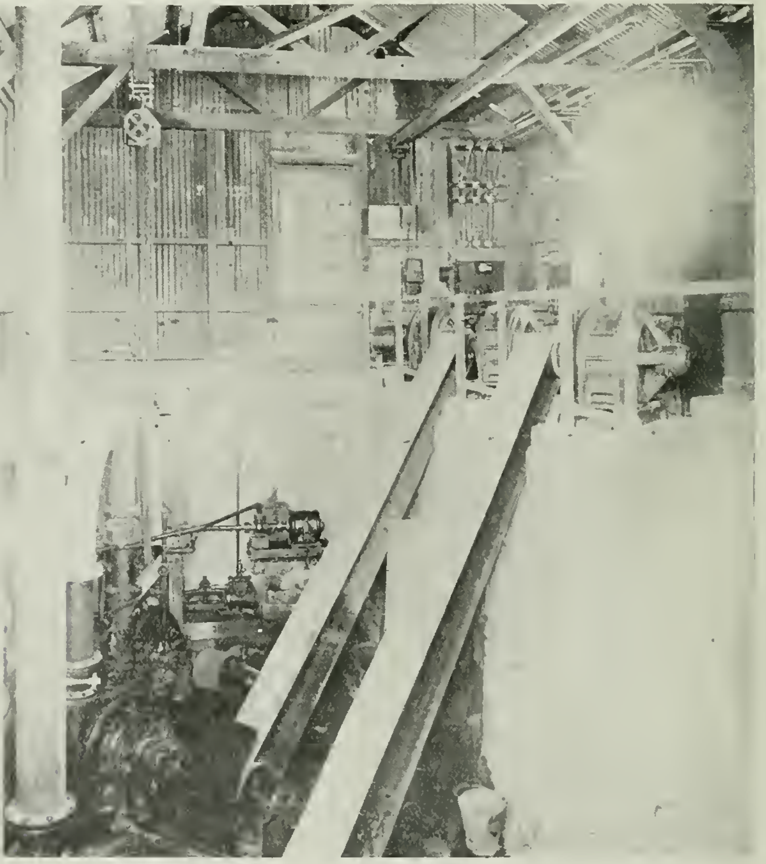
The main features of the arrangements for surface transport of sand residue at the Simmer and Jack are clearly shown in the diagram VIII. This outcrop mine extends over a large area and has been worked for many years. It was hence desirable and economically practicable to lower sand residue at several points, and the contingency of employing other points of lowering in the future had also to be borne in mind. Under these conditions portion of the sand from the residue trucks is periodically gravitated as a pulp from the mixing box F down the tunnel and borehole D, put down for this purpose. The remaining sand is pumped as a pulp to points A, B or C. (See illustration IV.) From A the cone overflow gravitates back to the mixing box E (15 ft. long, 8 ft. wide and 6 ft. deep with steeply inclined bottom) beside the residue track, although the tops of the cones at A are at ground level to saving pumping head. The cones at B and C are above ground as their location necessitates return by pumping of the overflow to the mixing box. (See illustration III.) The graded launder about 20 ft. long between the mixing box at E and the pumps serves as an automatic regulator of the consistency of pulp entering the pumps and prevents the latter being



III.—Dewatering Cones at B for delivering Thick Sand Pulp Underflow to Incline Shaft and Fluid Overflow for Return to E. Simmer & Jack Proprietary Mines, Ltd.

choked, as very thick pulp will not flow down this launder. (See illustrations I. and II.) The empty trucks in every case return to the sand plant. From the foregoing it will be seen that the arrangements for the surface transport of sand residue to each lowering point is determined by the conditions prevailing in each case, and that the circuit system so characteristic of modern ore treatment practice is fully utilized.

As the local conditions at the Robinson Deep differed somewhat from those at the Simmer and Jack it was decided after consideration to modify somewhat the procedure in regard to sand transport for sand-filling. The greater depth and lesser area of the mine rendered it both advisable and practicable to permanently restrict the lowering to one point at the north of the property. (See diagram VII.) A bore hole, decreasing from 10 in. diameter at the top to 7 in. diameter at the bottom and 1,729 ft. deep, was accordingly put down so that all worked-out stopes could be served from it by pulp gravitation underground, and as the upper end of the bore hole was 18 ft. higher than the track under the sand vats and a considerable distance away, it was decided to drive a tunnel ($4\frac{1}{2}$ ft. × 6 ft.) from the sand plant at a dip of 20° (36·4% grade)*. This tunnel was 1,125 ft. long and intersected the bore hole at 390 ft. from the surface, so that a thick pulp could be gravitated direct from the sand plant through the tunnel and bore hole into the mine without the need of pumping or dewatering the pulp. The open end of the tunnel is directly under the residue track and just clear of the vats so that the residue after dumping from trucks, and addition of a small stream of water and permanganate solution, descends as a 28% moisture pulp till it comes to rest in the desired stope underground. This installation costs a good deal more in capital expenditure upon the tunnel than the Simmer and Jack installation of surface pumps, pipes and cones for a lowering



IV.—Interior of Pump Station at E for Sand Residue Pulp, showing delivery pipe on left, Simmer and Jack Proprietary Mines, Ltd.

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*The percentage grade is conveniently calculated by multiplying the tangent of the angle by 100.

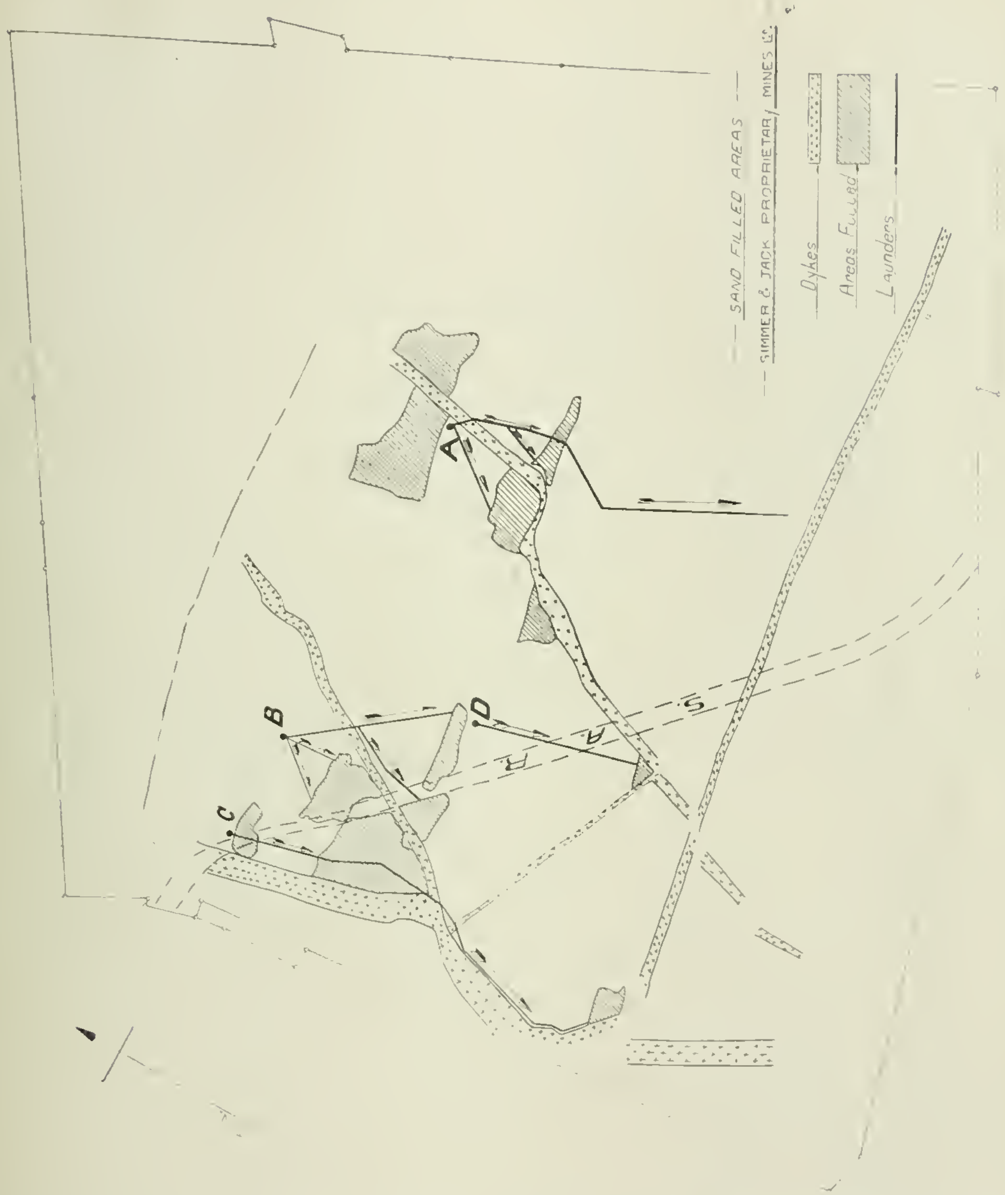

Diagram V.

point, but is preferable under the present conditions indicated and has the advantages of low surface operating costs, these being merely the short truck transport of residue from vats, tipping and returning the trucks, and a little permanganate. A screen of vertical grizzly bars across the tunnel prevents the liability of the bore being choked by foreign substances. The amount of drainage water to be pumped to the surface from underground involved by sand-filling with such a thick pulp is small, amounting only to about a quarter of a fluid ton per ton of dry sand deposited, or 250 fluid tons (50,000 gallons) of water daily with a monthly ore tonnage milled of 60,000 tons. Prior to the starting of sand-filling a system of diaphragm cone classification of tailing pulp was installed for the purpose of obtaining a clean sand residue for mine filling, and likewise for the advantages derivable during cyanide treatment.

With regard to the underground operations of sand-filling in general, when sand is filled with

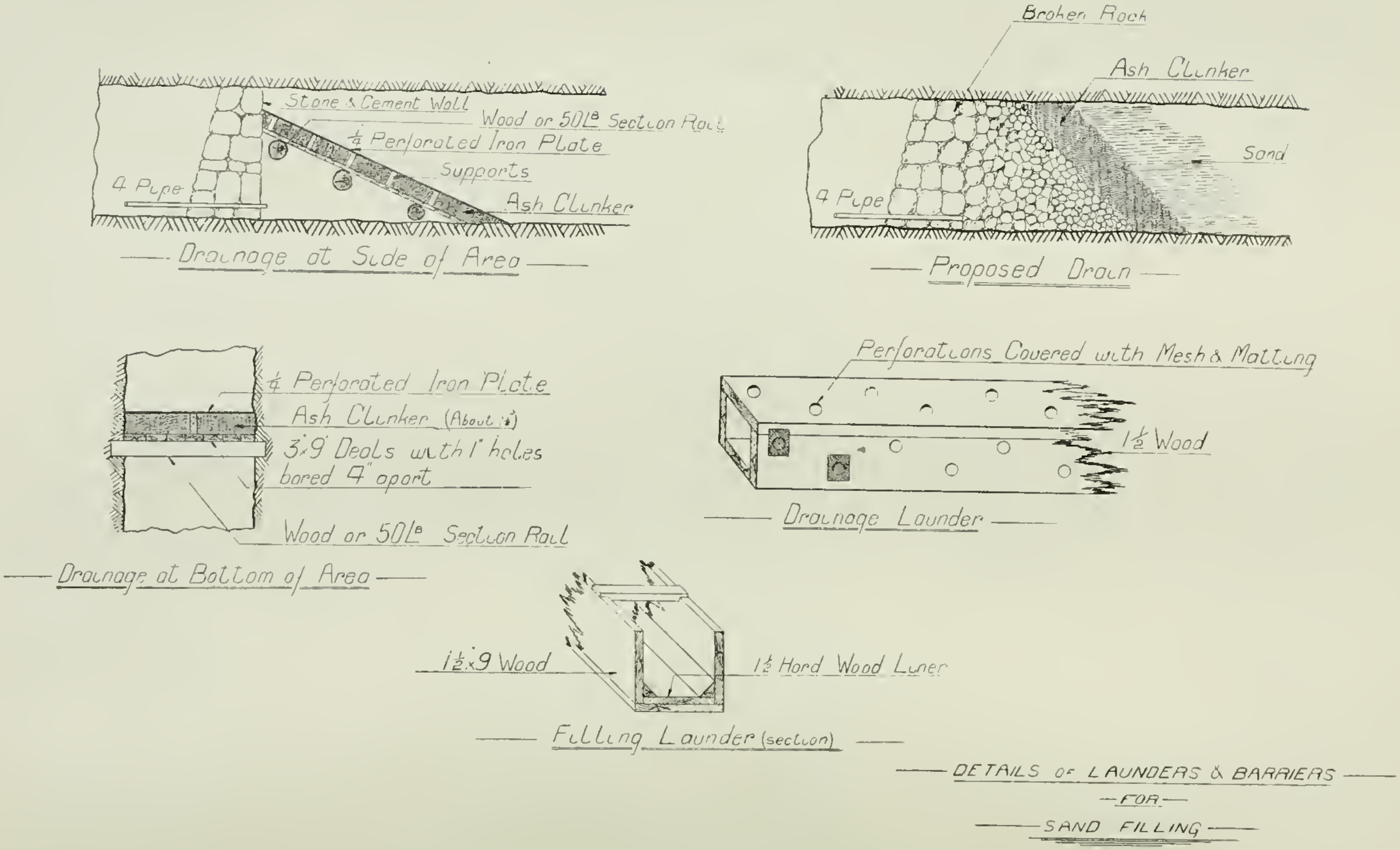
Stone & Cement Wall
Wood or 50lb Section Rail
¼ Perforated Iron Plate
Supports
4 Pipe
Ash Clinker
Drainage at Side of Area
Broken Rock
Ash Clinker
Sand
4 Pipe
Proposed Drain
¼ Perforated Iron Plate
Ash Clinker (About ¼)
3×9 Deals with 1" holes bored 4" apart
Wood or 50lb Section Rail
Drainage at Bottom of Area
Perforations Covered with Mesh & Matting
1½ Wood
Drainage Launder
1½×9 Wood
1½ Hard Wood Liner
Filling Launder (section)
DETAILS OF LAUNDERS & BARRIERS
FOR
SAND FILLING

Diagram VI.

the object of forming a permanent support to the mine working so as to protect the surface from the danger of future subsidence, the permanency of the barrier must be considered. If there is a possibility of water finding its way into the filled area, it will not be sufficient to construct a barrier that will last little longer than the period required for filling and drainage. Such water, except perhaps in the case of flat workings, would in time wash down the sand through any perished parts of the barrier into the lower parts of the mine. The construction of a permanent barrier is costly, but on many mines dikes, faults and unpayable bodies of ore can be utilized so as to serve as natural barriers. On the Simmer and Jack every advantage is taken of these natural barriers, and areas requiring filling are often considerably extended so as to permit of their use. (See diagram V.) It sometimes happens that immediate use cannot be made of such a barrier, owing to the fact that all the ore has not yet been removed from the area ultimately to be filled. In a case of this kind where there is urgent necessity for filling a portion of the area, a temporary barrier of cheap construction may be erected enabling filling operations to be carried on immediately.

The number of openings due to previous mining operations through these natural barriers is not usually great, enabling large areas to be completely closed at comparatively little cost. For some time the general practice on the Simmer and Jack has been to close up the drive, boxhole or other opening with a masonry wall provided with drainage pipes as described by Mr. A. R. Hughes in the discussion of the previous paper on sand-filling.* More recently such drainage has been effected through a bed of sifted clinker resting on a perforated platform. It has been suggested by Mr. W. R. Dowling that in order to make this drainage still more efficient and permanent a system originally employed for the leaching of chlorination vats might be adapted for the drainage of sand-filled areas. The system briefly described, consists in starting a drain with coarse material such as rock 9 in. in diameter, and gradually working up with rock of diminishing size and finally with clinkered ash until the fineness of the material to be drained has been reached. (See diagram VI.) This system has everything to commend it in the way of economy of construction, permanency and efficient drainage, and there is little doubt but that it will be a success in practice. The drainage just described is supplementary to the main drainage which is effected by means of drainage launders. The drainage launders as originally constructed were of wood or iron framing, square in section, covered with cocoanut fibre matting. These

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*See this *Journal*, October, 1910, p. 173, Vol. XI.

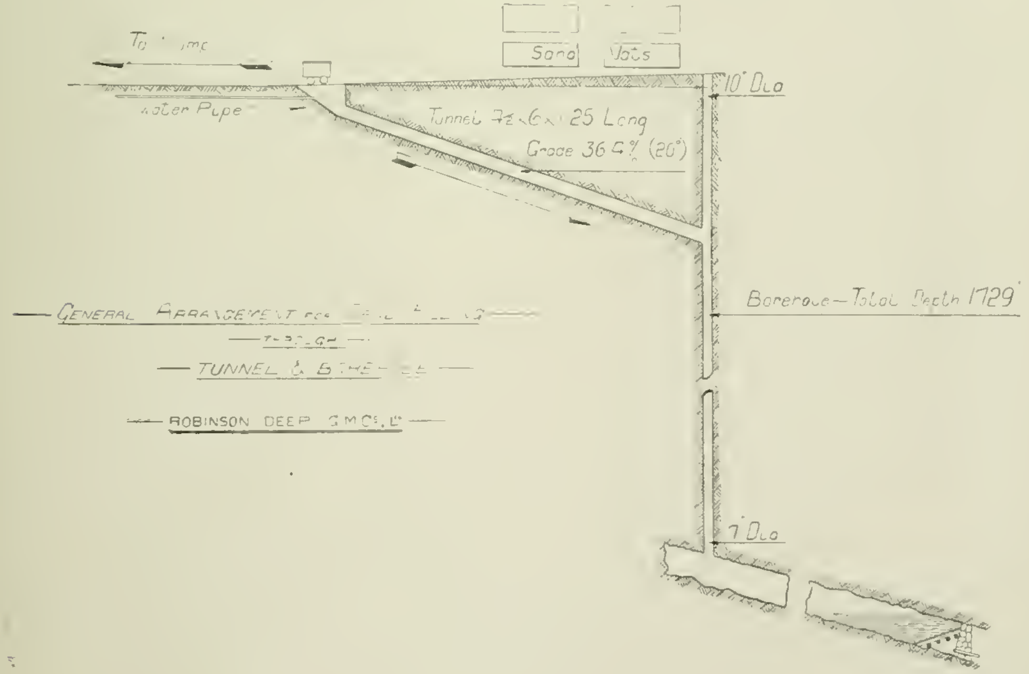


Diagram VII.

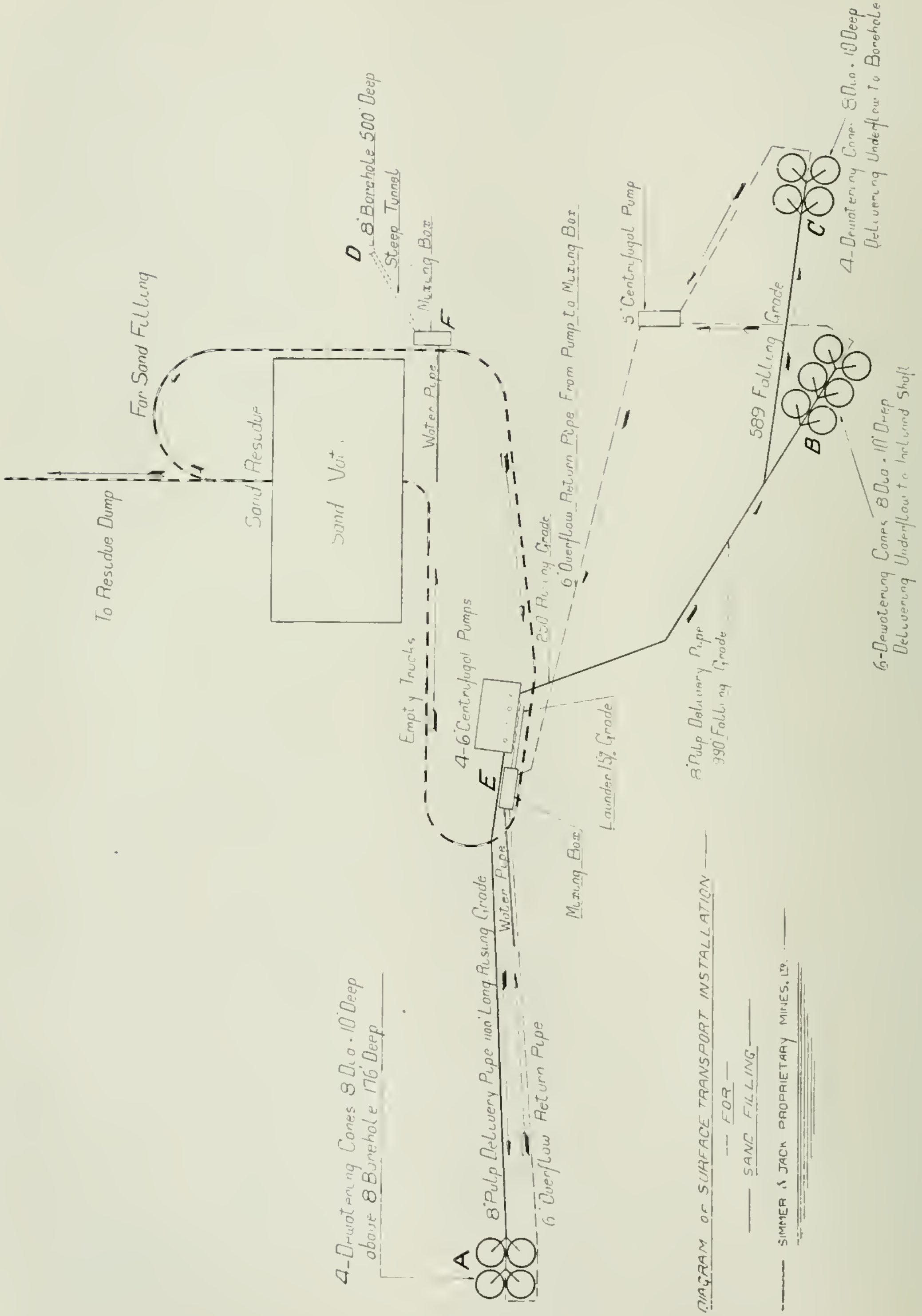

4-Dewatering Cones 8' Dia. 10' Deep above 8' Borehole 176' Deep
A
8' Pulp Delivery Pipe 1100' Long Rising Grade
6' Overflow Return Pipe
Water Pipe
Mixing Box
E
Empty Trucks
4-6' Centrifugal Pumps
Launder 1½% Grade
8' Pulp Delivery Pipe 990' Falling Grade
210' Rising Grade
6' Overflow Return Pipe From Pump to Mixing Box
To Residue Dump
Sand Residue
Sand Vat
For Sand Filling
Water Pipe
Mixing Box
F
D
8' Borehole 500' Deep
Steep Tunnel
5' Centrifugal Pump
589' Falling Grade
B
C
6-Dewatering Cones 8' Dia. 10' Deep Delivering Underflow to Inclined Shaft
4-Dewatering Cones 8' Dia. 10' Deep Delivering Underflow to Borehole
DIAGRAM OF SURFACE TRANSPORT INSTALLATION
FOR
SAND FILLING
SIMMER & JACK PROPRIETARY MINES, LTD.

Diagram VIII.

were found to be not only expensive but liable to collapse owing to the perishable nature of the matting. The greater portion of the sand filled has been drained through ordinary square sectioned box launders perforated with holes. These launders are very cheaply constructed but have two disadvantages. One of these is that they require constant attention in order to prevent clogging up by sand before the plugging can be accomplished. The other is that they are liable to collapse, although of course not to so great an extent as the cocoanut fibre matting launders. It is now proposed to employ a stouter box-shaped launder of smaller internal area. The perforations will be covered with cocoanut matting placed over wire meshing. It has been suggested that these might be further strengthened by filling with a core of sifted clinker. Where the drainage at the bottom of the areas is particularly good and the filling material clean sand free from slime, it is possible to dispense with the drainage launder.

Little change in the form of the filling launder has been made on the Simmer and Jack since sand-filling was first started. For a time wooden pipes were employed, as they recommended themselves both on the score of cheapness and the facility with which they could be erected. The use of these has since been given up and the launder at present used is stouter than that originally employed and wears better. It is also deeper in proportion to its width so as to prevent overflow of pulp. The bottom is lined with hard wood and the corners provided with hard wood fillets which can be renewed when worn. In future the hard wood lining will be replaced by old rubber belting.

Where the dip of the workings is not sufficient to carry the pulp freely along the launders water is added from some underground source of supply. This is preferable to thinning the pulp at the underflow of the cones or mixing box at the top of the bore hole with water which has subsequently to be pumped to the surface.

The whole of the underground work in connection with sand-filling is under the control of a special official, who being thoroughly conversant with what is being done in the various sections of the mine is able so to arrange matters that operations are conducted with the greatest economy and smoothness of working. This arrangement has been found better than placing each Mine Captain in charge of that portion of the sand-filling conducted in his own section. These officials have seldom the time to spare from their ordinary routine duties to attend to special work of this kind, and to keep themselves in close touch with what is going on in other sections of the mine.

It is interesting to note that when recently it was found necessary to re-open a drive through the centre of a sand filled area, this was done without any difficulty, and the sand was found thoroughly well drained and packed.

Where financial considerations did not interfere, and it was possible to delay milling operations until development had been completed on the lowest level of a mine, ideal conditions would exist for systematic sand-filling from the bottom upwards.

The cost of filling per ton of sand residue lowered varies considerably from month to month owing to fluctuations in the tonnage, and the expenditure incurred underground in providing for current as well as future requirements. The following are the costs incurred on the Simmer and Jack Proprietary Mines during a recent period of nine months (Table I., p. 132):—

The following statement illustrates more detailed operating costs for lowering 22,541 tons of sand during the month of November, 1912, (Table II., see p. 132):—

In considering the above average costs it should be noted that they include a considerable amount expended on preparatory work on areas where a large amount of filling remains to be done. It is probable that the monthly tonnage lowered will be increased shortly. These two factors should appreciably decrease the cost per ton of sand lowered in the future.

Mr. E. Pam *(Member of Council)*: It is a great pleasure to me to propose a vote of thanks to Messrs. Caldecott and Powell for a paper on a subject which interests me so much. I think we should congratulate ourselves that the whole of the sand-filling discussion on the Rand has been started and carried through at this Society's meetings. Further, I think we have a right to be pleased at the rapid progress which has been made in a period of about three years since sand-filling started. At the present moment there must be between 12 and 15 mines which are using this method, and I think it says a great deal, not only for the fact that sand-filling is good, but for the progressive minds of the men in this country. There are one or two points which struck me in this paper and which I should like to criticise in the comparison between the method advocated in this paper and that described by me in 1910.

I do not think it is made sufficiently clear that both methods are good, and that either should be used according to the circumstances under which it is needed. For instance, I believe that, on the Simmer and Jack, the pumping of sand and dewatering in cones is very excellent; but there are many conditions under which this system

TABLE I.

Period.	Tons of Sand lowered.		Average Surface Cost.		Average Underground Cost.		Average Total Cost.	
	Total.	Average per Month.	per Month.	Per Ton of Sand lowered.	per Month.	Per Ton of Sand lowered.	per Month.	Per Ton of Sand lowered.
			£ s. d.	d.	£ s. d.	d.	£ s. d.	d.
October, 1912, to June, 1913, inclusive.	172,535	19,171	269 0 1	3·368	523 10 3	6·554	792 10 4	9·922

TABLE II.

	Surface Cost.		Underground Cost		Total Cost.	
	Total.	Per ton of Sand lowered.	Total.	Per ton of Sand lowered.	Total.	Per ton of Sand lowered.
	£ s. d.	d.	£ s. d.	d.	£ s. d.	d.
White Labour	24 13 11	0·263	201 15 2	2·148	226 9 1	2·411
Native Labour	68 6 8	0·728	60 13 1	0·646	128 19 9	1·374
Stores	53 7 6	0·568	75 2 8	0·800	128 10 2	1·368
Workshops...	120 10 4	1·283	62 3 10	0·662	182 14 2	1·245
Power	46 19 7	0·500	—	—	46 19 7	0·500
Totals	313 18 0	3·342	399 14 2	4·256	713 12 2	7·598

could not be successful. I think the authors' will agree that as they lower their sand into koko pans (trucks) it would be equally simple—granted there was no obstruction in the way—to push those trucks by hand or mechanical haulage to a point near the borehole or shaft and tip them there, so saving entirely the pumping cost. When once the sand is in the pans I think it will be agreed that the cost of transference for a distance of, say, a thousand feet will be very small. Of course, if we could flush out our tanks and save the considerable cost of discharging by hand, then Dr. Caldecott's system would undoubtedly be the only one. The advantage claimed that this method is not intermittent as compared with other methods is not, in my opinion correct. Both methods are primarily dependent on the supply of sand coming forward in cars from the tanks or dump and the use of a storage bin must make the process less intermittent, as if the supply of sand is delayed the store in the bin can be used, and if there is any stoppage underground, tramming continues and the bin is filled again.

The method as described in use at the Robinson Deep is, of course, very similar to the Silesian method, excepting that the tunnel is much longer. The mixing bin at the Robinson Deep is not described, and it would be interesting to hear how it is constructed. The success of the borehole is a most important matter. I must own I have been sceptical, and I am exceedingly pleased to hear that on the Robinson Deep a 1,600 or 1,700 ft. borehole has been successful, and that there has been no choking. The difficulties have been tremendous where pipes are in use in deep vertical shafts. With regard to permanent barriers, Mr. Powell is apparently lucky —or unlucky, whichever way you take it—in having plenty of dikes and unpayable ground. We have no unpayable ground at the Geldenhuis Estate, and but few dikes. The result is, we had to build barriers. I do not contend for a minute, and I do not suppose anybody else will, that these barriers are going to last for a considerable time The only conclusion we came to was that it was absolutely necessary to arrange the packs in such a way that if there were any rush of water it could drain off between or at the side of the packs. The drainage system proposed by Mr. Dowling is very similar to the one used on the Ferreira Deep where a number of 2 in. pipes with small holes, about an eighth of an inch, were covered with ashes, fine and coarse rock.

With regard to launders, on the Robinson and Crown Mines, they found that concrete block in the form of egg-shaped liners have been very successful.

There is one very important point which no one has discussed yet: how are we going to sand-fill a mine 4,000 ft. deep? The only method which appears to me likely to prove successful would be to put down a borehole near a shaft, and after surveying, to rise from the shaft at an angle of about 50° and intersect the borehole about 500 ft. from the plane of the reef. The pulp would be caught in a hopper cut in the solid rock and thence led into a pipe running down the rise to the shaft. In this way the necessary head would be obtained to force the pulp east and west horizontally and the whole area of a large mine could be commanded from one borehole. The alternative of several boreholes along the northern boundary of the mine would be too expensive in practice, and the dilution with water mentioned in the paper is costly, and where it is desired to drive the pulp nearly horizontally impracticable.

I hope to give this paper the consideration which it deserves, and will try to obtain a few notes with regard to costs; I can state now that the costs mentioned in the paper are low in comparison with those on the mines with which I have been connected, but this is, I think, due to circumstances rather than to the method adopted.

I would like to propose a very hearty vote of thanks to the authors' for an exceedingly valuable paper.

Mr. J. Chilton (*Member*): I have great pleasure in seconding the vote of thanks. Probably sand filling on the Witwatersrand first started on the Wemmer section of the Village Main Reef about the year 1888. As a mine captain, looking for high-grade ore, I am exceedingly grateful to those early fillers, for the sand run down averaged 6 dwt., most of which has now gone through the mill a second time. I am afraid our successors will not have a similar cause for gratitude. On the Village Main we seem to have solved the drainage difficulty without the use of launders. We have divided the sand filling districts into three sections. Into one section we run for about an hour, and then we change over to another section, continuing there for a similar period, then run into section three. During the remainder of the day the sand is allowed to drain, and the water finds its way out of the sand very rapidly without the use of launders.

The President: Through the courtesy of the authors I have been able to inspect the sand-filling arrangements at the Simmer and Jack, and I consider that it would be difficult to improve upon them. I think mining engineers would like to have some details regarding the drilling of the boreholes, their cost, and so on, as this is a very important point, especially in view of the question as to the best method of transferring sand to the workings of a deep level mine.

THE RECOVERY OF BLACK SAND AND FLOATING PARTICLES OF METALLIC MINERALS.

(Read at March Meeting, 1913.)

By J. M. Neill (Member).

REPLY TO DISCUSSION.

Mr. J. M. Neill (*Member*): In replying to the limited discussion on my paper, I must thank those who took sufficient interest to contribute, and especially Messrs. K. L. Graham and Prof. Stanley for their kind remarks.

It would have been more enlightening, however, had a broader view been taken, and discussion directed more towards the principle involved rather than towards the method of illustration used.

I endeavoured to show that more gold was recovered in the mill by extracting more of the black sand and metallic minerals, and that their escape into the cyanide works was not followed by a reasonable recovery of their gold contents. Nothing having been advanced against this conclusion, I feel justified in maintaining that the extraction and separate treatment of such material would tend to lower final residues.

Turning to the contribution of Mr. Smith, whom I heartily congratulate on being the recipient of the student's medal, it is evident that conditions known to him are such as would lead him to believe that further refinements are not applicable or warranted.

The point for students and all of us to remember is that dissolved and undissolved gold finds its way to sand dumps and slime dams, and therefore we are not justified in endeavouring to show that a method for recovering this gold before cyaniding is not applicable because of certain conditions. We must also study the conditions to see if faults can be remedied so that we may apply that method or some better one.

As the contributor is a student I will go somewhat fully into his contribution, so that another point of view may be presented to him. "That the cyanide works can be relied on to extract anything leaving the plates" is a remark that the contributor will doubtless modify when he thinks of the residues. He will not then lightly suggest possible increases of 10%—15% by amalgamation, but will work to turn possibilities into certainties. When this 10%—15% is safely in the retort, he is certain of 100% extraction from it; on the other hand when this 10%—15% goes to the cyanide works neither Mr. Smith nor anyone else knows just what it will yield, but he can be certain that it will not yield 100%.

The impression conveyed, that efficiency in crushing is accompanied on the whole by careless and inefficient amalgamation, cannot be accepted, more especially when it is known that many plants on the Rand, having excellent crushing results, obtain 65% to 73% of their gold from within the crushing circuit. There are some exceptions showing low milling extractions, not necessarily because of high crushing results, but possibly illustrating Mr. Smith's remark that "careless dressing of plates is in many cases a source of more loss than is the excessive water." The statement that "to obtain a high stamp duty an excessive amount of water is fed into the boxes" will not bear examination. The factors which determine the stamp duty are roughly nature, quantity and grading of ore fed to boxes, crushing blow of stamps in foot pounds, number of blows per minute, quantity of water, height of discharge, available area of discharge of screens, and area of aperture. It is obvious that where the efficiency of those factors is established, the variation of ore only is liable to have an adverse effect. Excessive water for example has a most adverse effect; a greater inflow results in a greater outflow with increased velocity through the screens. This means in effect that particles much too large to pass the aperture are constantly carried to and held against the screen and therefore hinder rather than help crushing, by decreasing the effective discharge area. As a case in point showing that more than one factor must be varied to obtain efficiency, a mill of 1,030 lb. dropping 6″ at a speed of 116 per min. and using 4·8 water to 1 dry ore crushed 4·2 tons per stamp per day. Increase of height to 9″ and reduction of speed to 98 drops per min. gave a crushing of 5·1 tons per stamp per day. The water used was then 5·2 to 1 of dry ore, so that 0·9 of a ton more ore was crushed with the addition of 0·4 of a ton of water. Again, stamps of double the weight crushed 12 tons per stamp per day using 4·5 tons of water to 1 of dry ore; increasing the water ratio increased the duty per stamp up to a point, but doubling it had a contrary effect. It will be found that excessive water is used, owing to wrongly constructed launders and bad fall arrangements rather than for any benefit in crushing, and that the ratio of water to ore remains fairly constant in properly set out plants. Some of the remarks *re* dressing of plates will be considerably revised by Mr. Smith when he has had more extended experience. To say that "practice proves that plates should be dressed every four hours" is erroneous. Practice does not furnish proof, whereas results do prove that plates should be dressed before their efficiency for catching gold has become impaired. The time between dressings on different plants must be determined by results obtained and must also be influenced by the nature of the pulp and by the area of contact. Alkali-acid changes exercise a marked effect and so also do changes in temperature. The quantity of free gold per ton of ore need not be considered as gold amalgam is the best dressing, and the efficiency of the plate increases the more free gold there is. The application of the necessary mercury is easy and effective in this case without dressing, although extra scraping may be required. On one mine on the Rand good results were obtained by scraping and dressing once in twenty-four hours. On another the plates required dressing with only two hours interval owing to the large quantities of fine iron and sulphides present; on a third maximum efficiency is obtained with six hour intervals, the time in this case having been fixed after trials embracing longer and shorter intervals; on a fourth, scraping and dressing and four subsequent dressings, five handlings of the plates in all, gave results equal to those obtained with six and eight handlings, but better than results obtained with four handlings. Proof that plates should be dressed every four hours is therefore not established.

Before going further I must call Mr. Smith's attention to the fact that the conditions and methods of collection (not extraction) of black sand were as nearly as possible the same during the periods of trial as set forth in my paper.

Therefore, although I agree when he says "It is well known that gold amalgam is brushed off the plates along with black sand," I take exception when he says "The increase in value obtained by Mr. Neill was probably due to a larger amount of amalgam being removed." "This would be the case if the plates received a better working up with the brushes."

Reading those sentences in conjunction, it is evident that the contributor did not credit or did not understand that the author seriously meant that the black sand was collected by the same methods during both periods. Such was nevertheless the case.

This apart, I fail to see where an increase of 5·88% extracted from screen value could be obtained by varying the proportion of amalgam taken up with the black sand, as the sum of those two gives the total recovered. It would be as reasonable to place £40 in one pocket and

£60 in another and by varying the proportions taken from each to expect to obtain £105·88 as the result. Regarding the increase of 5·88 shown by the means employed, the remark that "this increase is by no means a large one" is hardly warranted when the progressive recovery results compiled from the same basis are examined. These results showing mill recovery were roughly before trial :—

2 Years.	1 Year.	During Trial.	Year Following Trial with Apparatus.
58%	66%	69%	72%

The year during trial was split into three periods of four months each, being a period without apparatus, and a period with complete outfit as shown in drawings accompanying my paper.

Coming to mental arithmetic for amalgamators, too much importance must not be attached to any scheme of this sort. The quantity of mercury to be used depends not only upon the recoverable gold per ton of ore passing on to the plates, but very largely upon the physical condition of the gold, and upon variations in the temperature of the pulp which naturally affect the quantity of mercury and its manner of application. Temperature variations taken every half-hour during one month in a non-condensing plant show, highest 88° F., lowest 66° F., average 75·4° F. The influence on the quantity required or the manner of using mercury was small. On a condensing plant however, alterations of 30° F. were frequent and sudden, and care had to be exercised to prevent loss. The plates became too soft on a rise of temperature and excess mercury had to be removed ; on a fall in temperature more mercury had to be added. Experience shows that Mr. Smith's 70 oz. mercury to 30 oz. gold is too arbitrary. This equals 0·4285 oz. gold recovered for 1 oz. mercury used dressing, whereas on one plant the average recovery for some years was 0·546 oz for 1 oz. mercury used dressing, lowest 0·53 highest 0·57. In the case of crushing some ore rich in coarse gold (not included in the above average) the recovery rose to 0·67 oz. per oz. of mercury used in dressing. The foregoing will show that experience combined with close observation must be the only guide to the use of mercury. The reasons advanced as to why a small portion at the top of a plate yields the best results and the rest but little of the black sand and gold are only partly correct even for a badly run plant, but do not apply at all to the type of plant which it is best to discuss, *i.e.*, that plant which is set out and run to return the best results from all departments. The contributor's reasons are, slow speed of pulp at the top of plate increasing to high velocity at bottom, excessive water and excessive mercury. In the first place the most of the gold and black sand is caught at the top of the plate because that is where the richest pulp first arrives, and where fresh areas of contact are constantly exposed owing to "splash" which, as it embraces height and time of fall, allows a certain amount of classification to take place and so assist in the recovery of valuable particles composing the pulp owing to their high scientific gravity. As for the flow being most rapid towards the bottom end of the plate, this should not be, nor is it so where recovery by amalgamation is expected—the grade is so arranged that the pulp will just escape without silting. That the flow is usually towards the bottom of plates is shown by silting taking place there first should anything go wrong.

In this connection consider resistance due to air affecting the watery surface of the pulp, and the decrease in velocity brought about by friction, which is directly proportional to the wetted area of plate surface so far as the water is concerned and is materially affected by the sand, and the reason becomes more apparent.

The remark that my apparatus would serve to check the flow of pulp is partly answered by the foregoing, when the small wetted area it exposes is considered ; but, further, the pulp passing down the plate must be thought of as passing in layers, a slowly moving layer of sand in close contact with the plate and a more rapidly moving layer of water surrounding and covering the sand, and of at least ten times the volume of the latter. It will be seen, therefore, that my apparatus, which floats, can only check the velocity of the water and so increase its volume at that point, and as the sum of the velocities must remain constant if silting is not to take place, it follows that the sand itself must move faster.

That excessive water, excessive mercury and excessive velocity of pulp have little to do with the poverty of the lower part of the plate in catch of black sand is apparent when even under the best conditions the yield from there is small.

Having fairly shown that Mr. Smith's reasons why my apparatus in some cases did work and in others did not are not correct in either event, I must fall back upon the original contention that small particles of metallic minerals are capable of attaching to themselves minute quantities of air and are thereby enabled to float, not because it "may be in part correct," but because it is wholly correct and worthy of further investigation.

I have somewhat elaborated my reply, as I feel that it is the duty of all members of this Society to assist students and each other to assign observed effects to their true and proper causes.

Mr. Dures has noted a fact with which I dare say most of us have already had some acquaintance. Turning to Mr. G. A. Robertson's contribution, which throws some valuable light on the subject, I cannot see where his difficulty lies in reconciling an increase in both quantity (weight) and quality (value) of the recovered black sand. He must have overlooked the essential point of my paper which claims increased extraction of this material. It surely follows that if each ton of pulp yields more of its heaviest and richest constituent the ultimate result must be a gain in both weight and value. The interesting facts brought forward by Mr. Robertson, *i.e.*, the surprising quantity of gold that floated in one case, and the prominently adhering quantity of black sand due to the canvas baffle in the other, should also help him over his difficulty. It is reasonable to suppose both those cases applied to one pulp passing under a floating apron. The metallic mineral and black sand would be brought down on the plate, and although some portion would amalgamate, a good deal would not do so but would be caught and lifted with the black sand and would therefore increase both the extracted weight and value of the latter. Replying to Mr. Robertson's questions, the cost of the aprons complete was less than ten shillings each, they lasted over a year and gave little trouble from scale, which was easily broken away.

No extra water was required when the float was adopted. Regarding the tests which are being made I only know of one, but have had no particulars. I would be pleased to furnish aprons for test purposes on request. In this connection it must be remembered that the pulp dealt with by me was not tube milled and contained about 30% only – 200, and would, therefore, as a pulp, be composed of very different percentages of groups of equal hydraulic value compared with groups in a tube milled pulp containing 50% to 70% – 200. An alteration of the S.G. of such a pulp, or the lengthening of the time for segregation might be necessary to get good black sand extraction results.

GOLD PRODUCTION IN RELATION TO HUMANITY ; A STUDY OF THE QUESTION : HAS GOLD DEPRECIATED IN VALUE?

(Read at April Meeting, 1913.)

By Mr. Eustace M. Weston (Member).

DISCUSSION.

Dr. R. A. Lehfeldt (*Visitor*) : I have not had the advantage of listening to Mr. Weston's paper and the discussion which followed, but I read the published versions of them. The author's paper is a most interesting contribution in the history that it gives of gold and its uses ; but I do not agree with some of the theoretical conclusions which he draws. The general drift of the paper goes to show that the production of gold is not excessive I do not agree with that, and I cannot help thinking that the author has fallen into a little trap—a very common one—with regard to prices. One reckons the value of anything else by saying it is worth so many pounds per ton, or whatever it may be, and the more pounds per ton it is the more it is worth. The value of gold is reckoned in exactly the opposite way. When prices are rising, that means that one has to give more gold for the same amount of other things and, consequently, the more prices rise the less the gold is worth. Of course, stated baldly like that, anybody sees it, and it seems, perhaps, too obvious to be worth dwelling upon ; but, as a matter of fact, one is rather liable to fall into a mistake on this point just on account of that inversion. To explain what I mean, I must quote a passage from the author's paper where, I think, he has dropped into this fault and which, I think, has led him astray in his theoretical conclusions. Just listen to this :—

"Prices have risen because owing to the needs of an expanding world more and greater efforts have been put forth year by year to obtain a sufficiency of the medium of exchange. That is to say, every year the world's gold has cost more in human effort and capital expenditure to produce it."

Well, that may or may not be true ; but, supposing it is true, if it takes more effort to produce the gold, then it means that one has to exchange more wheat and iron, and things of that sort, for an ounce of gold—in other words, the price of wheat and iron and other things will have gone down. Later on, for instance, he says :—

"One scarcely cares to guess at what may happen as the world continues to expand and the increase of Rand production is swallowed up by declines in other countries. Increase of demand must tend to raise prices, and shortage of gold to create financial panics."

It is exactly the other way, of course. Increase of demand for gold would reduce prices. A simple way to get over this confusing point is to think of prices in terms of something else. Take wheat as a standard of value. It is a perfectly intelligible statement that one goes into a shop to buy a new coat for eight bushels of wheat, or the monthly rent must be paid with 20 bushels of wheat : one would not pay wheat, of course, but would give a cheque on the bank for a corresponding amount. If an ounce of gold some years ago was worth 20 bushels of wheat and is now only worth 15, the value of gold, of course, has fallen and its price expressed in wheat has fallen. Putting that the other way round—in the ordinary language of taking gold as a standard—it means that one used to be able to buy 20 bushels of wheat for four sovereigns, now only 15 bushels can be bought.

I think people are agreed that prices have risen. There is no doubt about that. They have risen since 1896 to something like 25% or 30% taking the average all over the world. I think most people are agreed that if more gold is produced than is needed for a given amount of commerce, that will cause a rise in prices. It does not, of course, follow that the rise in prices which has occurred is due exclusively to the increased production of gold, but there is a general consensus of opinion that that is the leading cause. There are other causes for the rise in price which contributors to the discussion have pointed out. It is possible that monopolisation is one; to some extent, perhaps increased cost of marketing and other circumstances. But one cannot look at the world's rise in prices which has gone on now for something like 20 years without seeing that it is mainly due to a simple cause that affects all prices. Gold is practically a foot-rule by which things are measured. For instance, if 20 years ago this room was measured and it was found to be 45 ft. wide, and that foot-rule had since contracted and it was measured again now and found to be 50 ft., that would illustrate what has happened with regard to prices.

Now, taking the question as to whether the production of gold is excessive, one must bear in mind that it is the production of money which regulates price: that is not the same thing as the production of gold. The world's trade is always increasing and requires more money. The question is whether the amount of money in the world is increasing at the same rate as the world's trade. The money of the world is, for all practical purposes, mostly cheques and bank deposits, and, of course, they are based upon gold in the sense that the bankers have to keep a stock of gold proportionate, more or less, to the amount of credit which they offer to their customers.

I was much interested a little while ago in looking up some statistics on this point and I found a rather striking circumstance in America where there are statistics available as to the total amount of bank deposits and the total amount of gold retained by the banks. The ratio of deposits to cash kept steady for some time round about 1895 to 1900, gradually rose until 1907, by which time it had risen to something over 25 per cent. relatively to what it was at the earliest date of these statistics. It is well known, of course, that 1907 was the occasion of a catastrophe in credit in the United States. The bankers found that they had gone too far, and credit collapsed for the time, the banks had to reduce the amount of credit that they offered and increase their supplies of gold. Now they have gone back, as a result of that practically, to the same ratio as they had in 1896; so that, experience seems to show that in a given country the ratio of gold required to the money used is fairly constant, although sometimes there are changes in the ratio which lead to an inflation of trade and correspondingly of course to a severe contraction. If there were an exact ratio it would mean that the world's gold would have to expand at the same rate as its business, so as to keep things level. If prices have been rising lately it is probably due to the stock of gold increasing more rapidly than the amount needed. Against that one has to set the obvious fact that people do not cease to want gold. Every shipment of gold sent to London finds ready purchasers. Hence it is argued by authorities whom one must respect that the gold is not any more in amount than is needed. I do not agree with that view. I think it is a very difficult point to disentangle. I speak under correction; but this is how it presents itself to me: If there is an increase in production of some other commodity—say wheat, two things happen, the price falls and the total utility is increased. Let me put it the other way round; it will be plainer: Supposing the production of wheat contracts, the price rises, and utility is reduced,—some have to go hungry. That is the case with every commodity, but it is not the case with the medium of exchange. Gold, as money, is reduced in amount, the value of it increases, but the total utility of it does not decrease, because, overlooking the temporary inconvenience, suppose that for every sovereign there is in the world at the present time there were next year only half-a-sovereign—of course there would be a temporary upset, but when that is over, then half-sovereigns would be exactly the same worth as former sovereigns; the world's trade would be carried on in exactly the same fashion: so, reduction in the circulating medium does not mean reduction in the utility of that medium. Conversely, if the amount of gold in the world is increased too rapidly that means a fall in its value as indicated by the change in price, but it does not mean any improvement in its utility. The two thousand millions of gold in the world at the present time do not do any more useful work than the thousand millions of gold that might have been there if the Witwatersrand and other centres had not been discovered. All that is needed is some sort of a coin with which to carry on business transactions. If all the prices were halved and there was only half as much gold the world as a business world would get on just as well as it does now. There is not an excess of gold in the sense that there is more than enough to carry on the world's business, but there is a falling off in value of it as there would be with regard to any other article which was produced in greater quantity than was demanded, and one is so much used to associating an increase in supply with an increase in utility that when one sees that the utility is not increasing one thinks that the value is not decreasing, but I do not believe that is so really.

I should like just to refer to the recent papers of one of the leading authorities on these points. I mean Prof. Irving Fisher, who has ventured on a forecast. It is a rash thing in any case to attempt to forecast history, and Prof. Fisher may have been rash. He has come to the conclusion that prices will rise for some time to come. He is perfectly sound in insisting on the importance of paper money with regard to rising prices. Prices would not have risen so much if it were only a matter of the production of gold. Taking the United States alone, under similar conditions of banking, the ratio of deposits to gold has remained constant it is true; but taking the world at large a progressive economy in gold will be found: one finds the United States and England and the Colonies are the only countries in which the cheque system is thoroughly well developed and everybody uses it. In France and Germany cheques are not used to anything like the same extent; bank-notes are used there more freely. Then, passing to India and China, the circulation is nearly all metallic. As the world gets to more efficient methods and a more complete banking system, gold will be economised relatively to the amount of business to be done. The most economical country in the world is England on account of its excellent banking system. In the course of 10 or 20 years the world in general will no doubt, become more efficient. If the amount of gold is already increasing more rapidly than is needed, it will increase still more rapidly in consequence of that change.

Prof. Fisher attributes the rise in prices to a considerable extent to this progressive economy in the use of gold for money. He is no doubt quite right in drawing attention to the point; but how rapidly the world will change in that respect is, of course, a very disputable point, as also, I need not mention, how rapidly the supplies of gold will increase. I will not myself attempt to forecast that at all. I merely mention that, according to Prof. Fisher, the rise in price is likely to continue for a long time to come.

Mr. de Launay, who has written a most interesting book on the production and use of gold, points out that, whatever may be the present circumstances, the mines always give out in the end and scarcity is the thing to look forward to as the normal state of affairs. If gold continues to play the same important part in commerce as it does now the chances are, in the course of time, there will be a scarcity of it. We have had experience of that already. From 1873 to 1896 prices fell more rapidly than they have risen in the last 20 years. There has been so much talk recently about the rise in price that people have forgotten the fall. There have even been periods of more rapidly rising prices than the present. The fall in prices was attributed, I think, quite rightly, to the growing scarcity of gold when the world's business was increasing and the mines were not very productive. I take it that if the present position of gold as currency continues, and if the world's production does not go on increasing, but rather tends to fall, in a few years there will very probably be a fall in prices. One does not know how long the business world will continue to use gold in the same fashion as it does now. It is quite possible that improvements in credit, and in national confidence, may make it less necessary; so that I think it is really too rash to attempt to forecast the future: but, with regard to the present, I am convinced that the rise in prices is chiefly due to the excessive production of gold.

The President: I think we can congratulate ourselves now that we know a little more about this rather difficult subject than we did before.

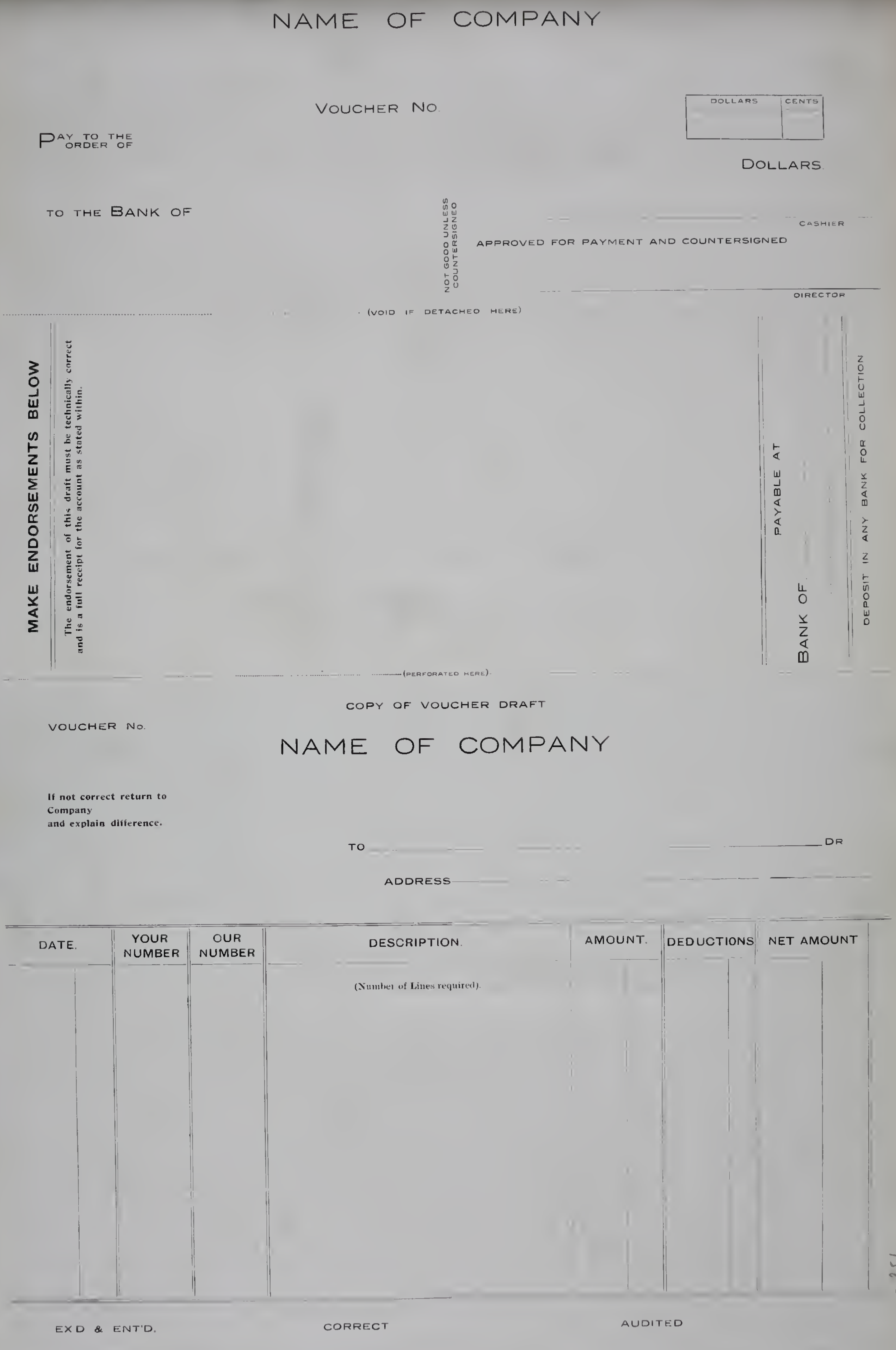
A SYSTEM OF KEEPING MINE AND MILL ACCOUNTS, COSTS AND METALLURGICAL RECORDS.

By M. W. MacLachlan, C.A.

The subject to be dealt with in this paper has already received considerable attention and lends itself to unlimited discussion.

The object of this paper is to briefly outline a system of centralization of work in preparing mine operating costs, records and general accounts as personally installed and supervised at a group of mines in the Republic of Mexico. The group consisted of eleven operating mines and two mills, besides which there was a considerable amount of exploration and development work being carried on at other mines comprising the

FIGURE 1. (Actual size of Form).



NAME OF COMPANY

VOUCHER NO.

DOLLARS	CENTS

PAY TO THE ORDER OF

DOLLARS.

TO THE BANK OF

NOT GOOD UNLESS COUNTERSIGNED

CASHIER

APPROVED FOR PAYMENT AND COUNTERSIGNED

DIRECTOR

(VOID IF DETACHED HERE)

MAKE ENDORSEMENTS BELOW

The endorsement of this draft must be technically correct and is a full receipt for the account as stated within.

PAYABLE AT

BANK OF

DEPOSIT IN ANY BANK FOR COLLECTION

(PERFORATED HERE)

COPY OF VOUCHER DRAFT

VOUCHER No.

NAME OF COMPANY

If not correct return to
Company
and explain difference.

TO DR

ADDRESS

DATE.	YOUR NUMBER	OUR NUMBER	DESCRIPTION.	AMOUNT.	DEDUCTIONS	NET AMOUNT
			(Number of Lines required).			

EXD & ENT'D.

CORRECT

AUDITED

NO.

NAME OF COMPANY

AMOUNT $

DISCOUNT

TOTAL

REGISTERED 191

DISTRIBUTION

([illegible])

VOUCHER No

COPY OF VOUCHER DRAFT

NAME OF COMPANY

If not correct return to
Company
and explain difference.

TO ____________________ OR

ADDRESS

DATE	YOUR NUMBER	CUR NUMBER	DESCRIPTION	AMOUNT	DEDUCTIONS	NET AMOUNT
			(Number of Lines required).			

[illegible] CORRECT AUDITED

same group and in the vicinity. The mining properties extended over a radius of about ten miles—the two mills being about that distance apart.

General Accounting.—The system adopted in keeping the accounts differed widely from the system adopted at the time the writer was in South Africa, and all tended to reduce clerical labour.

The following books were used :—Cash Book, Cash Voucher Register, Columnar Journal and Ledger.

All payments, with the exception of wages, were made by Voucher Drafts (Figure I). These Voucher Drafts were issued in triplicate, the original with the attached draft being sent to the payee, the duplicate being retained in the files, supported by the necessary documents, and the triplicate, in the case of materials, sent to the Stores Department. Owing to the laws of Mexico, requiring the Banks to retain all cheques in their possession and in order to prove payment, only the detachable body of the Voucher Draft was returned to the Company, through the Banks

The Voucher Drafts were registered in the book kept for that purpose (Figure II), the distribution charge being entered at the same time. At the end of each month the total debits and credits in this Register were transferred by means of a journal voucher to the Columnar Journal.

This Journal was arranged in columnar form and contained a record of all transactions, the details being carried on the Vouchers. The nature of the Voucher was shown together with the debit and credit figures.—(Figure III). As, therefore, each mine and mill column in this book showed the total monthly operating charge, against that mine or mill, the book served as a controlling account, and enabled the cost clerk to see at a glance if he had included everything in his cost sheets. By this system, great accuracy and speed were obtained. All Journal Vouchers were numbered and as all details were attached to and filed with the Voucher, the source from which the figures were obtained was readily traced.

Cost Sheets.—Each Mine and Mill had a separate cost sheet, as shown in the attached forms (Figures IV. and V). Details for these cost sheets were kept on draft form from day to day, these forms carrying a total charge column with which the sub-distributions to the cost account were required to be in agreement, and the totals in this column being in agreement with the total charges as shown in the Journal as before described, rendered it impossible to omit any item from the Costs.

A weekly cost sheet was also prepared for each mine and mill by means of which any leakages or excessive costs were ascertained before they could materially affect the total costs for the month.

FIGURE II. (Actual size of this form is 22″ x 11½″).

REGISTER OF CASH VOUCHERS

Vo. No.	Date	To whom payable	Amount	Disc'nt	Head Office	Ware-house	General Ex-penses	Distribution of Charges					Accts. Receivable
								Admin-istrat'n	Engin-eering	Assay-ing	Mine	Mill	

FIGURE III. (Actual size of this form is 22″ x 11½″).

DEBITS IN BLACK INK.

CREDITS IN RED INK.

JOURNAL.

No. of Vou.	Date	DEBIT	CREDIT	Nature of Voucher	Head Office	Ware-house	General Exp'nse	Accounts					Accts. Receivable
								Admin-istr'tn	Engin-eering	Assay-ing	Mine	Mill	

FIGURE IV. (Actual size of this form, holed for loose leaf binder, is 22½" x 18"). **NAME OF**

....Mine.

Tons to Mill, This Month.......................... *Assay: Ag*.......................*Kgms.*

,, ,, *Last Month*.......................... ,, ,, ... ,,

Acct. No.	ACCOUNT.	Labour	Power	Compressed Air	Illuminants	Explosives	Timber
	Exploration and Development						
1	Diamond Drilling						
	Drifts, Winzes, Raises & Crosscuts						
12	Driving						
13	Mucking and Tramming						
14	Timbering						
15	Tracking						
	TOTAL						
	Fills Development						
12½	Driving						
13½	Mucking and Tramming						
14½	Timbering						
15½	Tracking						
	TOTAL						
	Shaft Sinking						
22	Sinking						
23	Timbering						
24	Pumping						
25	Hoisting						
	TOTAL						
	TOTAL EXPENDITURE EXPLO. & DEV.						
	Ore Production						
31	Stope Preparation						
32	Breaking						
36	Handling in Stopes						
34	Timbering						
33	Tramming						
35	Track Maintenance						
37	Filling						
	TOTAL ORE BREAKING						
	Surface Handling (Ore and Material)						
61	Patio						
62	Sorting						
	TOTAL						
	TOTAL ORE IN MINE BINS						
	Transportation to Mill						
65	Surface Transportation						
67	Aerial Tramway Haulage						
	Local Administration						
71	Superintendence						
72	Maintenance Buildings						
	TOTAL						
102	Lighting and Telephone						
	TOTAL MINE OPERATION						
101	Drainage						
103	Surveys, Plans, Sampling & Assaying						
103½	Warehouse Expense						
	Proportion						
	Less profits sales to contractors						
	Proportion Genl. Mine Admin.						
	,, Direction Expense						
	,, General Expense						
	TOTAL						
	GRAND TOTAL						
	Detail of Distributed Accounts						
	Air Service						
51	Compressor						
52	Pipe Laying						
50	Pipe Maintenance						
	TOTAL						
	Hoisting						
41	General Hoisting						
42	Maintenance						
	TOTAL						
	Development Hoisting						
	Maintenance						
	TOTAL						
	TOTAL HOISTING						

COMPANY.

Month of..191...

Au......... .... *Gms.* *Contents : Ag.* .... *Kgms.* *Au.*... .... *Kgms.*

" .... " " " .... " " .... "

Lubrication	Miscel. Supplies	Mechanical	Electrical	Hoisting		Total	Cost per ton	REMARKS.
								Meters Drilled Cost per Meter $
								Meters Driven Cost per Meter $
								Meters Driven Cost per Meter $
								Meters Sunk Cost per Meter $
								Lineal Meters prepared Chutes built Cu. Meters Broken Cars Ore Level Stope " " " " " " " " " " " " Cu. Meters Filled
								Tons Ore Tons Waste Total Tons " " " " " " " " " "

FIGURE V. (Actual size of this form, holed for loose leaf binder, is 22½" x 18").

NAME O

TONS MILLED ....

TONS CYANIDED......... ... ...

....MIL

ACCOUNT.	LABOUR.		SUPPLIES.		Shoes Tires Pebbles	Dies Liners Filter Leaves	Screens	Power
	Operating	Repairs.	Operating	Repairs				
MILLING.								
Receiving & Delivering to Bins								
Weighing								
Handling								
TOTAL								
Crusher								
Batteries								
Chilean Mills								
Wilfley Tables								
Johnston ,,								
Elevators								
Classifying and Separating								
Tube Mills								
Mill Sampling								
TOTAL MILLING								
CYANIDING.								
Agitating (Pachuca Tanks)								
,, (Other ,,								
Slime Pumps								
Filter Plant								
Solution Pumps								
Zinc Room								
Melting Room								
General Supplies								
TOTAL CYANIDING								
TOTAL MILLING & CYANIDING								

(Repeated as above).

Statistics.—In addition to the cost sheets, a considerable amount of statistical work was carried out, showing from day to day the tonnages produced from each mine, assays, etc., and with regard to mills, the tonnages milled, assays of mill heads, tailings, etc. Attached are Figures VI. and VII., showing the nature of these statistics. In addition to these, charts of all monthly costs and production were kept for ready reference.

Storekeeping.—All requisitions for material to be used in the mines or mills were made out in triplicate—one original and two carbon copies, and signed by authorized employes. These requisitions gave information where the material was to be used, the number of that account, and, of course, the nature of the material. Two requisitions were given the storekeeper, the third—a tissue paper carbon, being retained by the party signing same. The storekeeper filled in weights or units, price and total value. At the end of the day each storekeeper filled in a distribution sheet, as per Figure VIII, and sent this sheet, approved by the mine captain, together with the original requisitions, to the central office. There the daily statements were thoroughly audited and the totals tabulated in a monthly distribution sheet, as per Figure IX. This system applied also to the mills, so that at the end of each week and month the value of stores consumed in each mine or office could be readily ascertained.

In dealing with the mill costs, the above applied only to sundry materials, the consumption of such

COMPANY.

MONTH OF..191...

Compressed Air	Shop Work	Mechanical.	Electrical Dept.	Total	Cost per ton		Total	Cost per ton.
						Melting Room Supplies		
						Fuel Oil		
						Crucibles		
						Fluxes		
						Misc. Supplies		
						TOTAL		
						Gen. Cyaniding Supplies		
						Cyanide		
						Lime		
						Zinc		
						Lead Acetate		
						Others		
						TOTAL		
						Summ. of Milling Costs		
						Ttl. Cyaniding & Milling		
						General		
						Lighting		
						Assaying		
						Warehouse Exp.		
						Supervision and Clerks		
						TOTAL OPERATING		
						Overhead Charges		
						Direction		
						General Expenses		
						Depreciation		
						Taxes		
						TOTAL COST		

(Repeated as above).

materials as cyanide and zinc, being dealt with in a separate daily mill report. A physical inventory of these stocks was taken monthly, to prove the accuracy of the mill superintendent's daily reports, and the corresponding charge to operating costs.

Timekeeping.—The method adopted in this department, and employing native timekeepers only, was as follows:—

All pay-rolls were made up weekly—the week ending the night shift of each Wednesday and payment was made on Saturday afternoon. On going to work, each man—mine or mill—was given a ticket on which was shown his name, occupation, rate of pay and number, and was stamped with the date of termination of the week. The ticket showed spaces for seven working days—sub-divided into quarter shifts. Underground time was kept by the shift bosses, who, by means of a special punch (each mine having a different punch mark) certified on the quarter sections of each day the time worked by each man under his supervision. At the same time he marked, in indelible pencil, the number of the account or accounts to be charged with the time. Until recently, underground timekeepers were employed to take the distribution of time, and a check was made at the end of each week, between the pay-roll as made up from their books, and the time shown on the tickets as punched by the shift bosses and collected from the men. This, together with the check made by the Inspector of timekeepers, who daily made an underground inspection of the men at their work

FIGURE VI. (Actual size of this form 20½" x 10").

NAME OF COMPANY.

....MINE MONTH OF................................191...

Date.	To-day.	TONS DELIVERED TO MILL.			WEIGHED INTO MILL.			CONTENTS.			Average Assay To Date.	NO. OF SHIFTS WORKED.			Tons per Man to Mill.	
		Total to Date.	Average per Day.	Rate per Month	Weight	Assay.	Total tons to Date	To-Day.	To Date.	Rate per Month.		To-Day.	To Date.	Rate per Month.		
1 to 31 Last mon. This mon.																

FIGURE VII. (Actual size of this form 20½" x 10").

NAME OF COMPANY.

ORE SHEET.. MILL................................191...

Date.	MINE DELIVERIES TO MILL (BATTERY BINS).								DAILY.		TIME LOST.					Remarks.
	Tons To-Day.	Tons to Date.	Tons Rate	Assay To-Day.	Assay to Date.	Contents To-Day.	Contents to Date.	Contents Rate.	Tons Milled Est	Tons on Hand Est.	Power.	No. Ore.	No. Stamps Run.	Stamp Hours.		
1 to 31																

(Actual size of this form 20½" x 10").

NAME OF COMPANY.

OPERATION SHEET.................................. MILL......................................191...

Date.	Mill-Heads to Date.	FILTER.							Ag. Assay Ext. to Date.	Au. Cont. Conc. to Date.	Ag. Cont. B'llion to Date.	Kilos Cons. p-t. NaCN to Date.	Kilos Cons. p-t. Acetate to Date.	Kilos Cons. p-t. Lime to Date.		Remarks.
		Heads To-Day.	Heads to Date.	Tails to River To-Day.	Tails to River to Date.	Washed Tails To-Day.	Washed Tails to Date.	% Eff. to Date.								
1 to 31																

FIGURE VIII. (Actual size of this form 23″ x 14″).

Name of Company.

Daily Report of Material Issued from Warehouse.

.................................191...

ACCOUNT NUMBERS FOR DISTRIBUTION.

Date.	No. of Requisition.	Article.	AMOUNT.		Price.	Value.	< Twenty Blank Columns. >
			Weight.	Pieces.			

FIGURE IX. (Actual size of this form 21″ x 10″).

Name of Company.

Detail of Daily Issues from the Warehouse

For the Month of..... ....191...

ACCOUNT NUMBERS FOR DISTRIBUTION.

Date.	Total.	< Thirty-one Blank Columns. >
1 to 31		
Total		

FIGURE X. (Actual size of this form 23″ x $11\frac{1}{2}$″).

Name of Company.

DISTRIBUTION OF PAY ROLL.

Week ending............................ 191

	RATES.						CONTRACTORS.						
Account No.	No. of Shifts.	Total Earned.	No. of Shifts.	Total Earned.	Ditto. Twelve Columns.	Total Charged to the Account.	Days.	Gross Amount.	Taxes.	Materials Supplied		Total Deductions	Nett Amount

FIGURE XI.
(Actual size of this form, holed for loose leaf binder, 22½" x 18").

NAME OF COMPANY.

Month of ....191..

STATEMENT OF ORE DELIVERED TO MILL.

	Kilos	ASSAY		CONTENTS		VALUE.		Gross Value.	RESERVE FOR METAL LOSS			Gross Recovered Value.	Reserve for Selling Expense.	Net Recovered Value.	Reserve for Treatment Charge.	Net Value.	Total Reserve Charged to Mines
		Ag.	Au.	Silver.	Gold.	Silver.	Gold.		Silver.	Gold.	Total.						
......District																	
......District																	
Both distri'ts																	

RECOVERY STATEMENT.

		Kilos.	ASSAY.		CONTENTS.		VALUE.		Total Value.
			Ag.	Au.	Silver.	Gold.	Silver.	Gold.	
.... Mill	Ore on Hand First of Month								
	Difference for Quotational Fluctuations								
	Ore Received during Month								
	TOTAL								
	Ore on Hand last of Month								
	Increase Metals in Process								
	Difference in Lag								
	ORE MILLED DURING MONTH								
	Production :—Bullion								
	Concentrates								
	Matte								
	TOTAL RECOVERY								
	Per cent. Recovery in Bullion and Matte								
	,, ,, in Concentrates								
	,, ,, TOTAL								
	,, ,, Metal Loss								
....Mill	Ore on Hand First of Month								
	Difference for Quotational Fluctuations								
	Ore Received during Month								
	TOTAL								
	Ore on Hand last of Month								
	Increase Metals in Process								
	Difference in Lag								
	ORE MILLED DURING MONTH								
	Production :—Bullion								
	Concentrates								
	Matte								
	TOTAL RECOVERY								
	Per cent. Recovery : In Bullion								
	,, ,, In Concentrates								
	,, ,, TOTAL								
	,, Metal Loss								

FIGURE XII. (Actual size of this form, holed for loose leaf binder, 22½" x 18").

NAME OF COMPANY.

Production.

Month of.............................191...

BULLION AND CONCENTRATES

	Lot No.	Date Shipped.	Dry Tons.	ASSAY.		CONTENTS		VALUE.				Gross Value.	Smelter Selling Expense.	Net Smelter Value.	Local Selling Expense.	Net Value.	TOTAL SELLING EXPENSE.	
								Silver.		Gold.								
				Ag.	Au.	Silver.	Gold.	Price.	Value.	Price.	Value.						Amount.	Per Cent.
Bullion																		
Concentrates																		
Matte																		
TOTAL																		
Bullion																		
Concentrates																		
TOTAL																		
TOTAL BOTH MILLS																		

SHIPPED TO SMELTER

in at least one or two mines, prevented any "dummy" men getting on the pay rolls. This system of employing underground timekeepers, however, has been dropped, and now the time is taken as the men enter and leave the mine, the distribution being made from the tickets when turned in at the end of the week. The amount due on these tickets is made up independent of the pay-roll, checked against the pay-roll and handed back to the workmen. All pay-rolls are sent to the central office, where they are audited in detail before being paid. The pay-rolls, paid tickets, a list of unclaimed wages and the corresponding cash are returned to the central office on Monday mornings. There the paid tickets are checked against the pay-rolls and the list of unclaimed wages verified,—the corresponding cash being handed over to the cashier. All tickets and distribution of labour were certified to by the mine captains and timekeepers.

The distribution of time is turned into the central office, as per Figure X., not later than two days after the termination of each week, entered in the monthly and in the weekly cost sheets, and thoroughly checked. In addition to the mines and mills, there were numerous other departments—such as workshops, engineering, mechanical engineering, etc., etc., whose pay-rolls were dealt with in a similar manner and a distribution made for the weekly cost sheets. Altogether there were on an average not less than 35 different weekly pay-rolls to be audited and distributed.

Workshops, etc.—Engineering workshops were operated on an extensive scale, and a considerable quantity of iron and brass castings were made, as well as all repair work. In order to keep a close record of the costs of such work, a system of shop orders was installed, enabling the final cost of a certain work, whether new or repair, to be accurately determined. This system of shop orders was carried into all branches of operating and construction, enabling all parties concerned to see at a glance how much a certain work had cost, irrespective of whether or not the expenditures had been charged out as incurred. This also gave a clear record of cost for future reference. This is a point of great importance, but it is rarely carried out to any extent, especially in operating repair work.

Metallurgical.—The two mills were practically operating as customs mills. The mines received full credit for the values of ores sent to the mills, and were charged with metal loss, selling expense, and costs of milling. Reserves were provided out of mine earnings to meet metal losses, selling expense and milling costs, on stock of ores on hand at the mills. This system, which I think is entirely opposite to the practice in the South African fields, proved very satisfactory and accurate, as well as being easily handled. Attached are Figures XI. and XII. showing the records of ore delivered, recovery statement and statement of bullion, concentrates sold, and metal sold.

General.—This system of centralization of the detail work usually carried on at the mines proved very successful, and has reduced the clerical costs considerably.

The staff employed on this work was as follows:

A chief auditor, 1 accountant, 1 bookkeeper, 2 cost clerks, 1 metallurgical clerk, 1 general clerk and 1 stenographer.

This staff also acted as general auditors in checking all work coming in from stores and time offices. The total monthly cost of this department was about £200.

Advantages.—A few of the advantages gained through this system were:—

(1) The low operating cost of the department.

(2) The accuracy obtained in the costs, by reason of each item being audited before charged.

(3) The collecting of all data relating to the actual operations in one central office.

(4) The saving of time required to get out the costs statements. The total costs were out on the 4th and detail costs ready for circulation not later than the 7th of each month.

(5) The preparation of weekly costs and earnings statements, enabling excesses to be discovered before running long enough to materially increase costs.

(6) The taking up in the accounts and crediting the mines with the values of all metals delivered to the mills, and not merely the bullion extracted, which obviously would not be fair nor show the correct mine earnings.

(7) The saving in supervision of clerical labour, one auditor looked after the accounts of all mines.

The writer, not being in close touch with present accounting practices on the South African fields, is not in a position to compare the two practices, but trusts that this brief description of a proved system may be of some use to Members of the Society.

VENTILATION OF THE MINES OF THE RAND: THE PROBLEM OF OBTAINING HEALTHIER CONDITIONS.

G. H. Blenkinsop, M.Inst.M.M. (Associate).

The report of the Commission on Miners' Phthisis and Pulmonary Tuberculosis, 1912, states on page 22: "The problem of prevention still awaits solution. . . . . . The experience of the past few years has shown that the prevention of disease is a very complex and difficult problem. Although the main principles involved—an adequate supply by efficient general and local ventilation of fresh air to all working places and the control of dust by means of water—are universally understood and accepted, the detailed application of these principles in the elaboration of the best practical methods to secure the ends in view has proved to be far from a simple matter. During the last three or four years, as we have already stated, much attention has been devoted by those responsible for the control of mines to the question of mine ventilation. . . . . This important work should effect a considerable amelioration of underground conditions. Detailed attention is also being everywhere increasingly paid to dust control by means of water." Since the date of this report mine ventilation and dust control have effected a further amelioration of underground conditions, but the problem of prevention still awaits solution.

In this paper I propose to examine in detail local conditions and, if possible, to ascertain why a system of ventilation which fulfils practically all requirements in Europe fails to attain the desired result on the Rand. As compared with Europe we have a much higher daily mean temperature, a susceptibility to pulmonary complaints owing to the high altitude. Our methods of mining are different, inasmuch as we have in the mines large worked out open spaces to which Kafirs have access and so make it extremely difficult if not impossible to keep the mines in a sanitary condition. These open spaces in some mines are being filled with sand residues, but it will always be practically impossible while Kafirs are employed underground—and the employment of white labour instead of Kafirs is not at present within the range of discussion—to reach the sanitary standard of European mines. Further, the mines in Europe in which the system of fan ventilation has proved effective are nearly all coal mines, and the problem there was to remove the excess of carbon dioxide, carbon monoxide and other dangerous gases. Dust in coal mines and soft rock mines does not affect health conditions as in the hard rock mines of the Rand. Fan ventilation is capable of keeping the percentage of carbon dioxide, carbon monoxide and the gases generated by the explosion of the gelatine within the limits required by the mining regulations, and it is also able to reduce the very high temperature which obtained on the mines of the Rand when they depended solely on natural ventilation. It is scarcely necessary to say that a current of air through mine workings—especially when it is charged with moisture—although it may feel cool, does not mean decreased temperature or decrease of impurity. Fanning oneself in, say, a theatre gives a cool feeling, but it does not lower the temperature of the place nor decrease the impurity of the atmosphere. Assuming that all difficulties connected with the fan ventilation of the mines of the Rand, such as the tendency of air from the downcast shaft to take the least line of resistance, which would be the most direct route to the upcast shaft; that the current of air from the downcast shaft can be diverted to ventilate the stopes and large worked-out spaces so that the latter shall not become breeding places for dangerous bacilli; that doors, wood stoppings and brattice cloths shall always be in order—brattice cloth is particularly liable to destruction by the acid water of the Rand mines—assuming that all these difficulties are overcome, does fan ventilation fulfil all the requirements of the Rand mines? My answer to this query is that it does not. The temperature of the air passed through the mine workings by means of fan ventilation will on the Rand be above the limit of safety, and it will also contain a lot of the finest particles of dust which are carried in suspension for the reason that the air from fan ventilation has to traverse a long distance underground before it reaches the working places. A large percentage of the dust produced by drilling, blasting and shovelling has been allayed by the carrying out of the regulations for the allaying of dust, and a further improvement will be effected when, as is probable, electric firing from the surface becomes general. Still it will be practically impossible to allay all the fine particles in suspension and even a very small quantity of dust is dangerous when contained in air of high temperature. Ventilation by fan tends to dry a mine as it removes an enormous quantity of water as moisture in the air, and therefore it will cause very fine particles of dust to be in suspension notwithstanding the use of sprays, &c. The temperature of the air breathed by the mine workers, and also the amount of moisture contained in it, are important factors in determining the health conditions of the Rand mines. At this high altitude all residents are more susceptible to pulmonary complaints than those living at a lower altitude; this applies especially to workers underground, whose lungs are subjected in the ordinary course of their occupation to more strain than those working at the surface. The Mines Medical Inspector, Dr. S. V. van Niekerk, M.D., in his report for 1912, states as regards native employees: "The disease which causes the greatest mortality is undoubtedly pneumonia, which in some months accounts for more than half the deaths." The Commission on Miners' Phthisis and Pulmonary Tuberculosis on Mines within the Union of South Africa, 1912, states: "A hot moist atmosphere has an injurious effect on persons working in it and consequently a lowered physical resistance to disease, and with an increase of temperature there is a corresponding increase in the amount of moisture the air can carry." Underground mine workers on the Rand are, therefore, normally, owing to altitude and the temperature of the air in which they work, particularly liable to be affected by small quantities of dust in suspension in the air and any other impurities which the air may contain. Mr. R. T. A. Innes, the Union Astronomer, has kindly favoured me with a table showing the temperature conditions at the Observatory. For practical purposes we can assume the temperature conditions at the surface of the Rand mines to be the same. The average mean daily surface temperature for the 12 months July, 1912, to June, 1913, was 59·5° F., and the average mean daily maximum temperature for the same period was 69·5° F. To ascertain the mean temperature at which air from fan ventilation would reach mine workers must be added the increase owing to rock temperature. The minimum increase of rock temperature is usually put at 1° F. for every 220 feet of working traversed, but in the case of the Rand mines must be added heat produced by oxidation of pyrite, decay of timbers, &c. We shall be making a very moderate allowance if we place the increase of temperature at 1° F. for every 200 feet of mine working traversed. It will be within the mark if we assume an average distance of 6,000 feet traversed, or say an increase in temperature of 30° F. before air from fan ventilation reaches the mine workers. The greater the distance air has to travel underground before it reaches the working places the greater will be the degree of impurity. The average mean daily surface temperature is 59·5° F., this figure plus 30° F. gives 89·5° F. as the temperature at which air from fan ventilation would reach the mine workers. Of course temperatures taken at the bottom of a downcast shaft would give lower figures than this. Temperatures will vary in different mines. The New Zealand Safety Commissioners' report states that in some mines an increase of rock temperature of 1° F. was obtained for each 33 feet and 43·5 feet respectively. If there is an error in fixing the temperature at which air from fan ventilation would reach the working places after it has traversed a distance of 6,000 feet underground—this 6,000 feet includes both vertical and longitudinal distance—at 89·5° F. it is that this figure is below rather than above the actual temperature. Authorities on ventilation state that injurious physiological effects begin to be produced when the wet bulb temperature rises above 75° F. It has been laid down by Commissions in several countries, including the Phthisis Commission, 1912, that the maximum temperature should not exceed 80° F. The maximum mean surface temperature at the Rand mines during the eight summer months is 74·0° F., so that the margin between the day summer temperature and that at which injurious physiological effects begin to be produced on underground workers is very slight. Fan ventilation has produced an amelioration of former underground conditions, inasmuch as it has reduced the percentage of noxious gases and reduced the very high temperature which obtained in the mines of the Rand when they depended on natural ventilation. It has been shown, however, that air passed through the workings by fan ventilation absorbs so much heat, moisture, dust and dangerous gases and bacilli from imperfect sanitation that by the time it reaches the working faces it constitutes a great source of danger to those breathing it. Fan ventilation certainly does not give "a supply of fresh air to all working places," and therefore does not solve the problem of prevention as set forth on page 22 of the report of the Commission on Miners' Phthisis and Pulmonary Tuberculosis.

I will now attempt to show how healthier conditions can be brought about in the mines of the Rand. If, instead of passing air for ventilation through the mine workings in the manner that is now done by fan ventilation, it were brought down in pipes from a pressure blower and liberated at the working places, the workers would have a regular supply of absolutely fresh air free from dust and noxious gases and bacilli and containing no greater amount of moisture than at the surface. In order to do this I propose to have at suitable places at the surface one or more blowers of the Roots' type. The particular type of the blower is not of consequence provided it will give a definite pressure of, say, half or threequarters of a pound. From the blower the air should be taken in suitable size pipes to air receivers or tanks at each station, and from these tanks or receivers it would be taken in small light pipes to the working places. The total cost of the installation would not be great, the figure would of course depend on the size of the mine. For a mine employing a total of 2,000 workers underground the cost would be less than £10,000. If at the working places there is a constant supply of fresh air the problem of obtaining healthier conditions will be solved and a big forward step taken in the direction of rendering underground work as healthy as underground work can ever be under the climatic conditions of the Rand. In a few mines on the Rand, such as the Crown Mines, which have a number of shafts connected by a wide and high main haulage way, a smaller quantity of fresh air per person at the working places will be necessary than in the case of mines having a lower standard of natural and fan ventilation and a smaller quantity of air passing through the main workings. In the case of mines having only one or two shafts and depending solely on natural ventilation, a system of ventilation such as I have described, which will take absolutely fresh air to the working places, is urgently required. It can be quickly installed and it is not costly.

In conclusion I would quote from the annual report of the Mines Department, 1912: "If health conditions were improved, better men would compete for the comparatively high wages on the Rand: both white and coloured men would put in a better day's work, and it is reasonable to suppose that more intelligent men would tend to reduce the accident rate. If these arguments were considered insufficient from a humanitarian point of view, it might be further urged that improved health conditions, by bringing better and more efficient miners, might tend to actually reduce working costs and cause larger profits to be made. Working under unhealthy conditions the efficiency of the skilled miner must remain low, and a larger number of persons must be employed to do a certain amount of work than would be under normal conditions. As regards white miners, the skilled miner with all his faculties clear and in working order breaks more ground and uses less explosives than a mere labourer, and the difference between placing holes rightly or wrongly is very great indeed, and has a far reaching effect, not only on the immediate output from the mine, but on the future cost of timbering and supporting weakened excavations."

Dr. J. Moir (*Past-President*): I should like to ask members to accord a very hearty vote of thanks to Mr. Blenkinsop for this paper. He seems to maintain an entirely new outlook on the question, and if his calculations with regard to the comparatively moderate cost of his scheme are correct, it would certainly seem to be a better one than fan ventilation, which certainly raises a lot of dust and does not purify the bad places, to say nothing of other questions with regard to miners' phthisis. Mr. Blenkinsop's figures for mine temperatures are all wrong however: he would have done better to have consulted the original authorities instead of attempting to calculate the temperatures. He would have found the correct figures in a paper by Whitehouse and Wotherspoon in the *Journal* of some years ago.*

Mr. E. Pam (*Member of Council*): I am sorry that this paper is rather destructive than constructive.

No one claims that fan ventilation is perfect, but I believe it is more practicable than the proposed blower method. I would ask the author to furnish us with fuller details in his reply to the paper, such as quantity of air supplied per person, number of blowers with capacity and horse-power, size of pipes for mains and branches, material of pipes, etc.

With regard to the raising of dust, I would point out that the air delivered by the blowers must come to the surface again through the mine workings and in so doing stir up dust if the strata are not wet, so that the method has the same disadvantages as fan ventilation. The ideal method of ventilation would consist of intake and exhaust pipes in the immediate vicinity of every workman, so that fresh air would reach each working place and be removed again as soon as vitiated.

The meeting then terminated.

Review.

PAST AND PRESENT METAL MARKETS.—Part I., 8vo 45 pp.: Part II., Charts. By W. E. FIGGIS. (*Australian Mining Standard*, Sydney and Melbourne).

The charts form a record of the market prices of copper, tin, zinc, lead and silver from 1890 to 1912 and space is provided for continuance to 1920, so that the figures relative to any period can be ascertained and the trend of prices noted at a glance.

The volume briefly deals with the market position at the various times and indicates the causes to which the variations in prices were due.

Naturally, the knowledge of these factors is of extreme importance to producers, no less than to the consumers, and in spite of the opinion held in some quarters that a mining engineer or metallurgist has enough to do in attending to the technical side of his profession, he cannot afford to neglect the commercial side, and a perusal will be well repaid.

Over and under production, manufacturing prosperity and depression, sympathy with movements of other metals, and no less the operations of "cornering" syndicates and rings; bulls and bears, and suicides and bankruptcies, all receive mention, and most of these are beyond the immediate ken of the technical man, much as they ultimately affect his "profit per ton."

The book, though short and necessarily statistical, is eminently readable and full of interest, and concludes with some notes on sources of supply and future prospects in which the actual and relative productions of the chief producing countries of the world are shown. (G.H.S.)

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*See also Moir, this *Journal*, Vol. XI., July 1910, p. 8.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Determination of Water in Coal.—"A weighed sample of the 80-mesh coal is placed in a 100 cc. pressure flask connected with a weighed U-tube containing quicklime, the latter being in the form of lumps in the limb nearer to the flask and in fine powder in the other limb. The U-tube is connected—through a sulphuric acid drying tube and an intervening tap—with a Sprengel water pump. A vacuum of about 700 mm. having been created in the flask, the latter is immersed in boiling water and the lime tube in a boiling aqueous solution of calcium or sodium chloride. At the end of about 30 minutes, the water and calcium chloride baths are removed, the apparatus disconnected from the pump and air slowly allowed to enter through the drying tube; the lime tube is then washed, dried, and re-weighed. The increase, it is stated, is due solely to water from the coal, no other volatile constituent of the latter being retained by the lime at the temperature employed: no carbon dioxide or phenol was evolved from any of the coals used by the author, but the lime tube was found to be attacked by the hot calcium chloride solution, necessitating a correction for the most accurate work. Excellent agreement was obtained as the result of triplicate determinations made upon various kinds of coal."—P. L. Teed, *Journal of the Society of Chemical Industry*, June 16, 1913, p. 588. (F. W. W.)

Detection of Hydrocyanic Acid.—"One drop of a 3% solution of copper acetate is mixed with 1 cc. of a 10% solution of disodium phosphate and 4 drops of a saturated solution of benzidine acetate, and the solution under examination added drop by drop to mixture. In the presence of 0·000027 gm. of hydrocyanic acid per cc., a turbidity and perceptible bluish-violet coloration are produced, whilst the presence of the phosphate prevents the benzidine and copper salt from reacting with iodides, ferrocyanides, etc. The test may be rendered still more sensitive by boiling the solution under examination from 5 to 10 minutes with sodium hydroxide, filtering it, and passing through the filtrate a current of carbon dioxide. This liberates the hydrocyanic acid, which is then carried forward into the reagent (1 drop of 3% copper acetate solution, 5 drops of benzidine acetate solution, and 0·5 cc. of water). If metals forming double salts with cyanides in alkaline solution are present, the cyanides must be separated before applying the test."—C. Pertusi and E. Gastaldi, *Journal of the Society of Chemical Industry*, June 16, 1913, p. 599. (F. W. W.)

Contribution to the Chemistry of Gold.—"Of a large number of organic and inorganic substances tried, only sulphurous acid and arsenious oxide proved capable of reducing auric to aurous compounds. Sulphurous acid is the more satisfactory; when a solution of sulphurous acid is added to a solution of auric chloride containing a considerable quantity of such chlorides as those of sodium, potassium, calcium, magnesium, zinc, and cadmium, a colourless solution is obtained with a quantity corresponding to the equation:

$$AuCl_3,xMCl_2+SO_2+2H_2O = AuCl,xMCl_2+H_2SO_4+2HCl,$$

and the end point is so sharp that the reaction can be used as the basis of a method for the volumetric determination of gold. In absence of the chlorides mentioned, auric chloride is reduced to metallic gold by sulphurous acid. Many types of organic compounds reduce auric chloride to metallic gold in alkaline solution, but only a few reduce it in acid solution. Many organic liquids immiscible with water extract auric chloride from its aqueous solution: the compounds most effective in this respect are aldehydes, ketones, aliphatic esters and cyanides."—V. Lenher, *Journal of the Society of Chemical Industry* (from *J. Amer. Chem. Soc.*, 1913, *35*, 546-552), June 16, 1913, p. 599. (F. W. W.)

An Active Modification of Nitrogen Produced by the Electric Discharge.—"For preparing and storing the nitrogen to be submitted to the electric discharge, two 15-litre aspirator bottles are arranged as a gas holder in the usual way, a muslin bag containing chopped phosphorous being suspended in the gas space. The gas-holder is filled with commercial nitrogen. All traces of oxygen are removed in 2—3 hours, and when the fumes arising from the phosphorus have subsided, the gas is ready for use. On its way to the discharge tube the nitrogen is dried by passing it first through a column of phosphorus pentoxide, 30 cm. × 5 cm., and then through a tube packed with copper gauze and cooled with liquid air. As the gas is withdrawn from the holder the water rises and finally drowns the phosphorus, dissolving out the oxides and leaving it ready for further use. The gas-holder should be covered with a black cloth to protect the phosphorus from the action of light. Contrary to statements by other workers (see Tiede, this *J.*, 1913, 232), the phenomena of active nitrogen are not due to the presence of traces of oxygen. Oxygen in fact acts unfavourably and 2% prevents the phenomena altogether. Hydrogen and carbon dioxide are less harmful than oxygen, but water vapour, even in traces, is very prejudicial. Active nitrogen combines with the vapour of mercury, cadmium, zinc, arsenic, sodium, and sulphur to form nitrides, which are decomposed by water or potassium hydroxide solution, with evolution of ammonia. Carbon bisulphide yields polymeric carbon monosulphide and a blue polymeric nitrogen sulphide. Sulphur chloride gives ordinary yellow nitrogen sulphide. All organic compounds tried, except carbon tetrachloride, yield hydrocyanic acid freely, but not cyanogen. In the case of halogen compounds cyanogen chloride is formed. Benzene appears to yield cyanobenzene."—R. J. Strutt (Roy. Soc. Proc., 1913, A.*88*, 539), *Journal of the Society of Chemical Industry*, July 31, 1913, p. 750. (J. A. W.)

Determination of Small Quantities of Lead. "The method proposed consists essentially in precipitating the lead in the presence of ferric iron by means of ammonia, dissolving the precipitate, and reprecipitating the lead as molybdate after the iron has been reduced to the ferrous condition. To the lead solution is added at least ten times as much iron (in the form of ferric chloride) as there is likely to be lead present, ammonia is added, and the mixture is boiled. The precipitate is collected, dissolved in hot concentrated hydrochloric acid, the solution neutralised with ammonia, and $N/2$ hydrochloric acid added to the extent of 2 cc. for every 0·01 gm. of iron used, with a minimum of 10 cc.; after the addition of not less than 1 gm. of sodium sulphite, the solution is warmed until the iron is reduced, ammonium acetate is then added, and the boiling solution is precipitated with ammonium molybdate solution. The precipitate is collected, washed with hot water, dissolved in hot hydrochloric acid, and the solution, after the addition of a small quantity of tartaric acid and dilution to 50 cc., neutralised with ammonia. The solution is then acidified with a few drops of hydrochloric acid, ammonium acetate added, the mixture boiled for a short time, and the precipitate collected, washed with hot water, ignited with the filter paper, and weighed. When the lead solution under examination contains phosphorus, antimony, and tin, it is necessary to precipitate the two latter together with the lead by means of hydrogen sulphide: the tin and antimony are then separated from the lead in the usual way with ammonium sulphide, and the lead sulphide is dissolved in hydrochloric acid containing bromine, and determined as molybdate. The method may be applied to the determination of lead in quantity not exceeding a few mgm. in 50—100 cc. of solution, and also to the determination of lead in such alloys as brass, phosphor-bronze, etc., where the amount of lead does not exceed 1%."—A. V. ELSDEN and J. F. STANSFIELD (*Chem. Soc. Trans.*, 1913, *103*, 1039), *Journal of the Society of Chemical Industry*, July 31, 1913, p. 769. (J. A. W.)

NEW METHOD FOR REDUCTION OF FERRIC SALTS AND THEIR TITRATION WITH $KMnO_4$.—"The ferric solution to be reduced is placed in an Erlenmeyer flask with a little pure granulated zinc and a piece of platinum wire, and acidified with sulphuric acid in the proportion of 1 to 2 cc. per 100 cc. of solution. 2 to 3 gm. of ammonium sulphate is then added, the mouth of the flask covered with a small funnel, the liquid brought to a gentle boil and kept boiling until reduction is complete. The object of adding ammonium sulphate is to form ammonium ferrous sulphate, which is very stable in sulphuric acid solution. The filtered ferrous solution is titrated with permanganate in the usual way. Details are given for using the method in the case of blood, thus:—Incineration is effected in presence of a few drops of sulphuric acid, which causes the mass to swell and enables it to be calcined without decrepitation. The ash is treated with a few drops of water and evaporated. If the residue adheres to the sides of the dish, 5 to 10 gm. of potassium acid sulphate is added, and gently heated to fusion, and this state is maintained until the iron oxide has disappeared. The mass is treated with water, filtered if necessary, and titrated as described."—A. LECLERE (*Pharm. Chim.*, June 16, 1913, 587) *Journal of the Society of Chemical Industry*, July 31, 1913, p. 769. (J. A. W.)

CYANAMIDE IN AGRICULTURE.—"Rapid growth in the fertilizer industry that has attended the development of agricultural science, and the great increase in the number and extent of chemical industries during the past fifty years, has emphasized the necessity for artificial methods of maintaining and increasing the world's stock of combined nitrogen. One of the influences that stimulated immediate action was the introduction in 1887 by MacArthur and Forrest, and at about the same time independently by Siemens and Halske, of the cyanide process for leaching gold and silver from their ores. This produced a strong demand for cyanides, which had hitherto been used to the extent of only a few hundred tons a year, principally in the dye industry and to a smaller extent in electroplating.

The introduction of the electric furnace in 1894 for producing carbides on a large scale, gave new direction to the researches. Siemens and Halske, among others, adopted it for the working out of the problem of nitrogen fixation. In 1898 it was found that barium carbide alone, heated to a temperature of 700° to 800° C. in an atmosphere of nitrogen, readily absorbs nitrogen and forms a product in which about 30% of the nitrogen is present as barium cyanide and about 70% as barium cyanamide.

It was further found that on fusion of this mass with soda the cyanamide is converted into cyanide. This mass can then be leached with water, and the cyanide solution thus obtained can be converted to sodium ferrocyanide, which is easily separated in a pure form, and which can be sold as such or be converted into pure sodium cyanide by fusion with metallic sodium.

The interruption in the production of gold in Africa, due to the Boer war, caused a decline in the price of cyanide, which stimulated the search for cheaper methods of production. It was found that calcium carbide could not only be manufactured at a lower cost, but it had the advantage of a lower molecular weight. With calcium carbide a temperature of from 1,100° to 1,200° C. is required for absorption, but in this case the nitrogen is fixed exclusively in the form of calcium cyanamide. On fusion with alkaline salts the cyanamide is readily converted to cyanide, and this can be extracted and purified in the usual manner.

Agricultural experiments with the crude calcium cyanamide showed that this material is suitable for use as a nitrogenous fertilizer, and patents were issued in 1910 protecting the use of cyanamide for fertilizing purposes.

The large demands of agriculture for cheap nitrogenous fertilizer materials have directed the efforts of the manufacturers toward the production of cyanamide rather than of cyanides and other derivatives.

Calcium cyanamide is formed in accordance with the equation

$$CaC_2 + N_2 = CaCN_2 + C.$$

The raw materials, therefore, are calcium carbide and nitrogen. Calcium carbide is formed in accordance with the equation

$$CaO + 3C = CaC_2 + CO.$$

The raw materials for this reaction, therefore, are lime and coke.

For the production of carbide, lime from 1 in. to 2½ in. in size, and coke, about ½ in. are mixed, in the proportions demanded by the equation, to produce a carbide about 80% pure. The mixture is shovelled continuously into a three-phase electric furnace. The furnace consists of a supporting base and four retaining brick walls which are filled with the lime and coke mixture, into which are submerged three carbon electrodes carrying the current. The resistance of the materials to the passage of the current raises them to a temperature at which the lime melts freely and combines with the coke to form liquid carbide. At intervals, as the carbide accumulates, it is tapped off into iron cars. After cooling, the carbide is crushed and ground to a fine powder, and is then packed in perforated cylindrical steel cans in the axis of which is a cylindrical paper core. The can with the carbide is set in a brick walled oven, slightly larger than the can, and is covered with an iron lid. A carbon pencil is run through the paper core in the axis of the can. Upon passage of the electric current, the carbide is heated, nitrogen is admitted to the can, and the absorption takes place very readily, raising the temperature to about 1,100° to 1,200° C., at which point it is maintained until the absorption is complete. The reaction $CaC_2 + N_2 =$

$CaCN_2 + C$ is exothermic, and the heat given out is almost sufficient to maintain the mass at the combining temperature. Upon completion of the reaction the cans are removed and the product is allowed to cool. It is a bluish black solid, containing small glistening crystals of pure calcium cyanamide.

The nitrogen required for the manufacture of cyanamide is separated from the air by either the liquid air process or the copper oxide process. The copper oxide process is based upon the ease with which heated copper combines with oxygen. The very finely divided copper suspended in a mass of asbestos or other inert material is contained in boiler shells provided with means for supplying heat externally. Air is pumped through the retort, which is at about 800° C., and is completely deprived of its oxygen. The nitrogen is then freed from carbon dioxide by passage through a caustic soda tower. The moisture is removed by refrigeration, followed by passage of the gas through layers of lime, and finally through calcium chloride.

The crude cyanamide obtained from the nitrifying ovens, after cooling, is crushed to a fine powder, and is passed into rotating, slightly inclined, steel cylinders, in which it is treated with sufficient water to eliminate the carbide and to slake the calcium oxide. A further slight addition of water is made before the material is run through briquetting machines.

The resulting bricks harden rapidly and are stored until they can be crushed just prior to shipment. Crushing is done in a series of rolls with intermediate screens, the final product having a coarseness of about 15 to 50 mesh screens. It is packed in ordinary fertilizer bags and distributed in carload lots to manufacturers of mixed fertilizers. The material so prepared contains nitrogen equivalent to about 19% ammonia.

At present, cyanamide is used in America principally for agricultural purposes. It is known to have a fertilizing value equal or superior to that of other common products used for the same purpose.

The particular advantage of cyanamide as a fertilizer material lies in the advantageous effects that follow its admixture with other constituents of complete fertilizers. It greatly improves the mechanical condition of such mixtures, and by its alkaline properties prevents the escape of valuable nitrogen oxides and of bag-rotting hydrochloric acid, set free by the action of free acids in acid phosphate upon nitrates and chlorides in the mixtures.

It has been definitely established that cyanamide applied to the soil is completely converted in the course of a few days into urea, by the catalytic action of the colloids and other constituents of the soil.

The urea is further converted by bacterial action and possibly by chemical processes into ammonia, which further reacts with zeolites, humates and other soil constituents, to form double ammonium salts, that are retained as a part of the soil until further bacterial action or the solvent effect of plant roots makes them available to vegetation. Cyanamide is said to be completely utilized by the crop in from 60 to 80 days.

When cyanamide is subjected to the action of superheated steam under pressure of several atmospheres, it is converted almost quantitatively into ammonia and calcium hydroxide. It thus serves as a convenient and cheap source of pure ammonia.

By fusion of cyanamide with alkaline salts in the presence of carbon, cyanides are easily produced and can be separated and purified as desired.

Various compounds of cyanamide, and its derivatives, with alkaline salts and carbonaceous materials, are made and sold in Germany under the names of Ferrodur, Intensit, Hesolin and others. These compounds, known as 'hardening' or 'cementing' powders, are used in place of cyanides, ferrocyanides, waste leather and similar materials, for case-hardening of steels. These products containing cyanamide are not only cheap, but they are efficient for the purposes for which they are intended."—E. J. PRANKE, *Engineering Magazine* (from *The Chemical Engineer*), June 1913, p. 432. (A. McA. J.)

CATALYSIS AND CATALYSERS.—"For some years catalytic action has been employed in industrial manufacture. Catalysers have grown very numerous: Haber, the inventor of the synthetic manufacture of ammonia, employed successively iron, nickel, manganese, chromium, and platinum. Unexpectedly, it was discovered that the rare metals were excellent catalysers. Osmium at a temperature of 550° and under a pressure of 175 atmospheres causes nitrogen and hydrogen to create ammonia, but as osmium is very expensive, uranium has been most beneficially substituted. The catalytic power of uranium nitrate is 63, whilst the others give 0, 2, 3, 5, except chromium, whose relative energy is represented by 21·2.

The problem of catalytic action has, says the *Echo des Mines*, been singularly widened by the researches of Gabriel Bertrand on catalytic manures. What is the nature of the action of the metals that become absorbed in the growth of vegetation (as well as in that of animals), of which 31 have been detected in land and marine trees and plants in various states of combination? All these bodies are catalysers, they do not enter into definite compounds, but act simply by their presence. Manganese is a catalyser which fixes the oxygen in the plant without which it would die, but an infinitesimal quantity is sufficient. A hundred-thousandth part of zinc assures a good harvest of asparagus; without it, nothing. Indeed, it is now generally considered that the metalloids and metals occurring only in feeble quantities may play the part of catalysers in the reactions, which may be compared to those of ferments.

Reflecting on these phenomena of catalysis, it would appear as if some other action may be associated with them. Radio-active fertilisers are now much commended, and may not catalytic action be a phenomenon of radiation? Why should the salts of uranium possess a greater catalytic force than all other substances? Is it because they are radio-active, and are not all bodies which Nature utilises in plant and animal life more or less endowed with this principle, which is susceptible of exciting molecular activities of every description, and which may be regarded as latent energy? What renders this conception reasonable is that, with the addition of light, catalysis augments considerably the molecular changes. Light alone, the ultra-violet rays, produce some decomposition; light is therefore radio-active, but is 300 times less so than if it operated in presence of a catalyser like nitrate of uranium.

It is obvious, concludes our contemporary, that we are faltering on the border of things new and strange, and are at that stage where the pioneers of electricity found themselves in the far past. Radio-activities of different natures will soon be recognised. Perhaps each substance has its proper radio-activity; to-day we know well only those of uranium, thorium and barium, to-morrow we shall possibly recognise that of chromium."—*London Mining Journal*, July 26, 1913, p. 722. (A. R.)

The Production and Consumption of Chemical Fertilisers in the World.—"A publication of the Bureau of Agricultural Intelligence and Plant Diseases of the International Institute of Agriculture (Rome) has recently been issued entitled 'Production et consommation des engrais chimiques dans le monde.'

Some of the data therefrom concerning the world's production are given in the subjoined table:—

	Production in metric tons.		
	1903	1910	1911
Phosphatic fertilisers:			
Mineral phosphates	2,433,779	5,344,981	6,055,073
Basic slag	2,243,500	3,275,845	(3,485,500)
Superphosphates ...	5,130,900	9,604,260	—
Guano	(56,000)	(66,044)	—
Potash salts (for agriculture):			
Potash salts (calculated as pure potash)	301,144	766,583	840,000
Indian salpetre ...	20,570	15,581	15,273
Other potash fertilisers (calculated as pure potash)	—	—	40,000
Nitrogenous fertilisers:			
Nitrate of soda ...	1,466,993	2,432,949	2,487,000
Sulphate of ammonia	537,520	1,045,905	1,187,425
Cyanamide	—	30,000	52,000
Nitrate of lime ...	25	25,000	50,000
Total nitrog. fertil.	2,004,538	3,533,844	3,786,425

It is believed that 1,000,000 metric tons of chloride of potash, corresponding to 630,000 metric tons of pure potash, could be produced yearly in the United States from seaweeds. It has also been proposed in the United States to use felspars for the extraction of potash in a soluble form; a yield of 400,000 metric tons of pure potash could thus be obtained yearly. The production of sulphate of ammonia has increased five-fold in twenty years. The Mond Frank-Caro process would allow of the manufacture of 88 lb. to 176 lb. of sulphate of ammonia per metric ton of peat. The production and probable consumption of synthetic nitrogenous fertilisers has been calculated for the years 1913 and 1914. The figures which have been estimated are enclosed in brackets:—

	Calcium Cyanamide.	Nitrate of Lime.
	Metric tons.	Metric tons.
1903 ...	—	25
1904 ...	—	550
1905 ...	—	1,600
1906 ..	500	1,600
1907 ...	2,200	15,000
1908 ...	8,300	15,000
1909 ...	16,000	25,000
1910 ...	30,000	25,000
1911 ...	52,000	(50,000)
1912 ...	95,000	(75,000)
	204,000	—
1913 ...	(97,000)	(140,000)
1914 ...	(208,000)	—

The world's consumption of fertilisers in 1911 represented a value of about £80,000,000, and was made up as follows:—

	Metric tons.
Lime phosphates	5,669,000
Superphosphates	8,604,000
Basic slag	3,300,000
Guano	70,000
Potash salts	4,100,000
(Pure potash)	(848,400)
Nitrate of soda	2,313,450
Sulphate of ammonia	1,100,000
Synthetic nitrogenous fertilisers	100,000

Data are given on the consumption of fertilisers per unit of area in the different countries. Belgium, Mauritius, and Luxemburg use upwards of 278 lb. per acre of cultivated area; Germany and the Netherlands from 89 to 178 lb. per acre; Denmark, United States (Southern States), France, England, Australia, Italy, and Switzerland from 45 to 89 lb. per acre; next follow the countries consuming from 9 to 45 lb. per acre of cultivated land: Austria, Hungary, Spain, United States (North-East), Norway, Dutch East Indies, Portugal, Sweden. All the remaining countries consume less than 9 lb. per acre or an unknown amount."—*Journal of the Society of Chemical Industry*, Vol. xxxii, No. 15, August 15, 1913, p. 801. (J. A. W.)

Reactions of Aluminium Nitride.—"Aluminium nitride prepared by heating aluminium powder in a current of nitrogen, powdering the product, and renewing the treatment for 1—2 hours at 1100°—1200° C. may be purified by heating for a short time to a red heat in the presence of dry hydrogen chloride; free aluminium and iron, which impart a grey colour to the crude nitride, are thus volatilised as chlorides, leaving a white product. The purified nitride is decomposed by fused alkalis or by aqueous solutions of acids or alkalis, with the production of ammonia, and may be analysed by fusion with potassium hydroxide, or better by 5—6 hours gentle boiling with dilute sulphuric acid (1 to 4) and subsequent distillation with alkali. It is slowly attacked, on heating, by dry halogens and more rapidly by sulphur chloride vapour, giving aluminium halide and nitrogen: it is slowly oxidised by heating with air or oxygen, and is more rapidly attacked by basic oxidising agents (lead oxide, sodium peroxide, etc.) than by those of a neutral or acid character (potassium nitrate, chlorate, etc.); it is slowly attacked by sulphur, carbon bisulphide, or phosphorus at a high temperature, and on heating with a dry mixture of sodium and potassium carbonates and carbon in an atmosphere of carbon monoxide, at 750°—800° C., gives cyanide: ethyl alcohol, at 230° C., reacts to form triethylamine. The amorphous nitride prepared by the authors' method differs in specific gravity from the commercial crystallised product, but the two are identical as regards chemical properties."—F. Fichter and A. Spengel (*Z. anorg. Chem.*, 1913, *82*, 192) *Journal of the Society of Chemical Industry*, August 15, 1913, p. 789. (J. A. W.)

METALLURGY.

Stamp Mill Curves and Speeds.—"For uniform lifting velocity, the involute cam, working on the flat-faced tappet, is used, and has the proper shape for the purpose. The effect depends upon the diameter of the inscribing circle and not upon the distance between centres of the shaft and stem, although, to reduce the side thrust as much as possible, it is better to make this distance equal to the radius of the circle. It is a property of the involute curve that the velocity of lift, during the time the involute is in action, is the same as the velocity of a point on the inscribing circle, from which follows the necessity of making the diameter of this circle to correspond with the velocity of lift and number of drops or revolutions selected. The neglect of this point will result in a failure to produce the desired end.

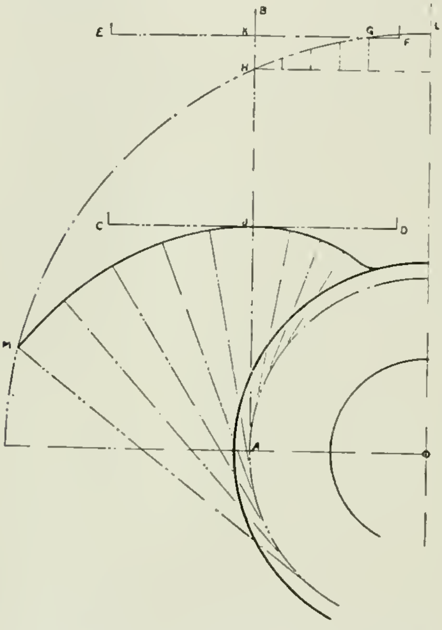

Fig. 1.—Method of Laying out Cam Curves.

Fig. 1 shows the method of laying out a cam for a given set of conditions: Given 11 in. inscribing circle, 6 in. drop, 108 drops per minute. Draw an 11 in. circle, with stem centre line *A B*, line *C D* at lowest desired position of tappet for 6 in. drop, and *E F* 6 in. above it for the highest position. Mark at *G* where the cam leaves the tappet (taking into account the cutting off of the cam corner), and draw a circle with radius *O G*. This will be the outside diameter of the cam, and it will show, by its intersection with the line *A B*, how much of the total lift is done at the uniform velocity and how much at a decreasing velocity. The lift from *J* to *H* will be uniform; at that point the involute ceases to act as such, and the remainder of the lift *H* to *K* will be at a velocity which decreases as the distance of the point of contact from the shaft centre line *O L* decreases. The involute is drawn in the usual way and terminates at *M*. A point on an 11 in. circle, making 54 r.p.m., has a velocity of 2·6 ft. per second, which will be the lifting velocity from *J* to *H*, a height of 4 15/16 inches.

Fig. 2 is a diagram showing the time occupied by the various parts of the stamp movement. The line *A E* represents 0·405 second and is the time for the full movement, *A B* is the uniform velocity lift, *B C* the decreasing velocity lift, *C D* an allowance for final stopping of the stamp, and *D E* the curve of the drop. At the velocity under consideration, the theoretical retardance curve, due to the velocity at *B*, and which resembles closely the upper part of the curve *D E*, would appear very little above *B C*, and there is no doubt but that frictional resistance will keep the tappet on the cam. It is, however, evident that the work on the cam during this part of the lift will be confined to friction. At higher velocities there will be a greater tendency to leave the cam at *B*, but whether or not it will do so will depend upon the friction.

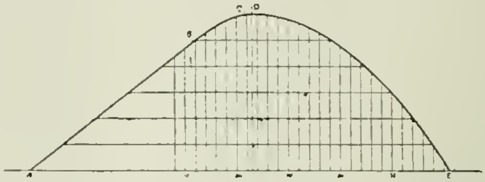

Fig. 2.—Velocity Curve for Cam Lift and Drop.

As the allowance *C D* depends upon the final lifting velocity at the instant the cam leaves the tappet and upon frictional conditions, it will be somewhat indefinite, and will vary with the diameter of the inscribing circle, even with the same lifting velocity. For the curve *D E*, 7% has been added to the theoretical time required for falling bodies, as shown by experiments noted in 'Ore Dressing' by R. H. Richards. In Richards' table of computed drops per minute[*] there is an error, as may easily be determined by a few minutes with a slide rule. None of the inscribing circles will give a lifting velocity of 2·66 ft. at the revolutions noted. The data are given in the table below.

Beyond *E* there will be a certain time allowance for rest, which, with the movement time, will make up the total time for one drop. In this case, at 108 drops, the total time is 0·555 second, leaving a rest period of 0·15 second. By multiplying the number of feet, 2·6, by the numbers of seconds, 0·15, the 'clearance' of 4·67 in. is found. This 'clearance' is the distance travelled by a point on the inscribing circle between the instant when the tappet reaches its lowest point and when it is struck for the next lift. It represents an effective clearance between cam and tappet and is a convenient means for comparing various working conditions.

There are two ways in which the number of drops per minute can be increased: by increasing the velocity of lift and by decreasing the rest period. How much the rest period may be safely reduced will depend upon operating conditions, and especially upon the steadiness of the speed and the closeness to which the drop is held to that selected. With fixed height of drop and speed, a material increase in the number of drops may be made by reducing the rest, provided a corresponding change is made in the involute. When the drop increases, by reason of wear, from that originally set, the clearance de-

Drop, in.	3	4	5	6	7	8	9	10	14	18
Drops, min.	143·50	126·80	115·20	105·40	97·60	91·30	85·50	80·30	65·50	55·60
Circles, in.	6·00	7·30	8·46	9·57	10·58	11·54	12·53	13·47	17·08	20·49
Velocity, ft	1·88	2·02	2·13	2·20	2·26	2·30	2·34	2·36	2·44	2·49
Circles should be ...	8·52	9·66	10·60	11·58	12·50	13·40	14·30	15·20	18·65	21·95

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[*] 'Ore Dressing,' Vol. I., p. 216.

creases very rapidly, as it loses at both ends of the movement. Overspeeding of the shaft also soon results in interference, as it reduces the total time per drop, but does not make a corresponding increase in velocity of lift. It is experience with cams running under these unsuitable conditions which has given rise to the idea that high-speed cams are out of the question.

At the present time, cams are ordinarily run to give a lifting velocity of about 2·6 ft. per second, which seems to be about the limit with the standard form of tappet held in place by friction. With the higher speeds, it will be necessary to use a more secure method of holding the tappet, and it will be advantageous to cushion the blow struck by the cam. By sacrificing speed, cams may be made to reduce the violence of the blow by a modification of the curve to permit picking up the stamp at a point closer to the shaft. The effect is obtained only when the tappet drops to the point for which it was designed.

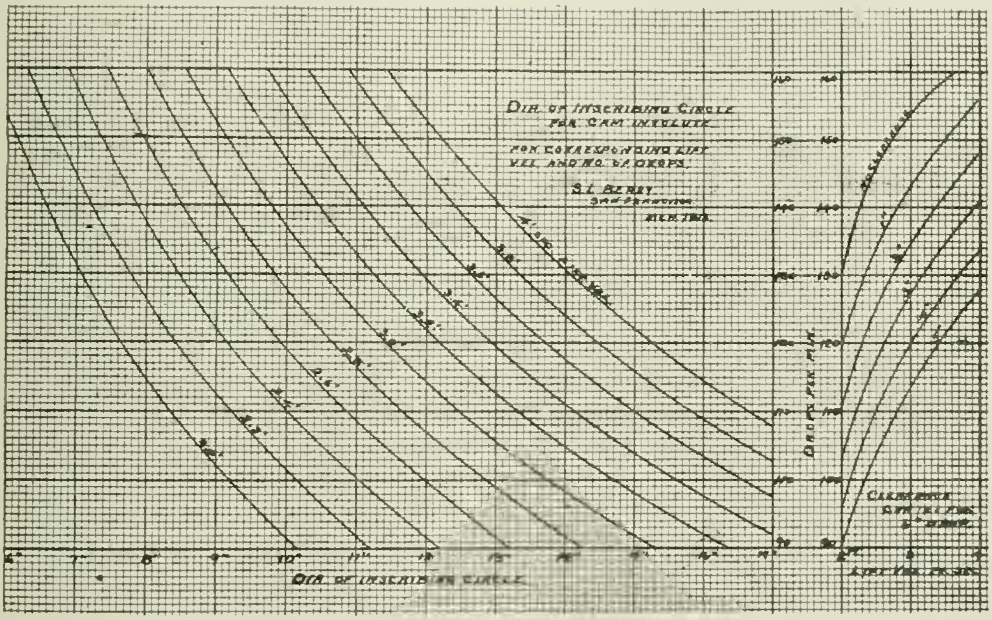


FIG. 3.—Diagram for Clearance and Diameter of Inscribing Circle.

It will be noticed that, in the foregoing, no account has been taken of an easement on the point of the cam. The desirability of this will depend upon the relation of the curve BC (Fig. 2) to the theoretical retardance curve and upon friction. It will be of more use in slow than in high speeds. Neither is account taken of the slowing of the drop when the stamp strikes ore.

Fig. 3 gives the diameter of the inscribing circles for certain velocities of lift and number of drops per minute. The only conditions involved are the two given: for instance, at 110 drops and 2·4 ft. per second, a 10 in. involute is required. Whether it is possible to obtain that number of drops at that velocity must be determined by other means. The diagram of clearances for 6 in. drop answers the question for that particular case. The clearance at 2·4 ft. lifting velocity and 110 drops is about $3\frac{1}{4}$ in., and the time by calculation is 0·13 second, which is ample. For 130 drops at 2·6 ft., a 9·15 in. involute is required, and the clearance for 6 in. drop is about $1\frac{3}{4}$ in., while for the same number of drops at 3 ft. per second, a 10·5 in. involute is indicated and the clearance is about $2\frac{5}{8}$ in.; and at 4 ft., 14·1 in. involute and about $4\frac{1}{2}$ in. clearance.

The above analysis provides a means by which high-speed cams can be designed, but, as stated before, for speeds much above 2·6 ft. per second, it will be necessary to improve the method of holding the tappet and for the higher speeds to cushion the blow."—S. L. BERRY, *Mining and Scientific Press*, May 31, 1913, p. 831. (A. McA. J.)

ALUMINIUM PRECIPITATION AT NIPISSING.—"In the course of preliminary experiments on the ore to find out the conditions of extraction of the silver by cyanide, it was noticed that the solutions, after being precipitated with zinc and used again, rapidly lost their dissolving efficiency.

At first I thought the trouble might be due to gradual accumulation of reducing agents in solution, because when tested with permanganate, as suggested by Clennell, it displayed considerable reducing power. I do not, however, consider that reducing power, as indicated in this way, is necessarily any criterion of the dissolving efficiency of a solution, because substances may react by the permanganate method as reducers, though they may not have power to abstract oxygen from the solution (e.g., KCNS) and if they do not absorb the dissolved oxygen from the solution, they cannot be classed as reducing agents in the sense of being detrimental to extraction on that account. Still, I have thought it a point for careful investigation in this case, and tried aëration of the solution even to an exaggerated extent, and also the addition of hydrogen peroxide and oxidation by electrolysis, but all without effect in improving the dissolving power of the solution. It was then obvious that the trouble was to be looked for elsewhere.

Zinc Fouled the Solution.—The first thing to come under suspicion was the presence of zinc, seeing that a fresh solution that did not contain zinc gave a good extraction, whereas the same solution, after the zinc precipitation gave a bad one, and after a second precipitation, a worse one. By way of following up this suggestion, three extraction tests were made under similar conditions on a sample assaying about 30 ozs. of silver per ton. *A* was treated with precipitated stock solution, with the analysis shown in Table 1.

B was treated with new solution and *C* with stock solution as in Table 1, but after most of the zinc had been precipitated by sodium sulphide. The ore was ground to pass a 200-mesh screen, and 48 hr. agitation was given with a solution of 0·2% free cyanide.

The residues assayed, *A*, 6·55 oz. Ag. ; *B*, 2·31 oz. Ag. ; and *C*, 3·70 oz. Ag. This result pointed to the zinc as being the offender, and following this indication, two tests were made on another sample of ore, one being treated with a plain freshly made cyanide solution, and the other with a solution (also freshly made), to which 0·1% of zinc had been added. The residues after 48 hr. were: *D*, 4·28 oz., and E, 8·19 oz. per ton.

TABLE 1.—Analysis of Solution used in Test.

KCN (free)	0·09 per cent.
Alkali (protective) in terms of NaOH	0·13 ,, ,,
Ag, per ton of solution	0·80 oz.
Zn	0·113 per cent.
KCNS	0·069 ,, ,,
Sb	0·0025 ,, ,,
Cu	0·021 ,, ,,
As	0·021 ,, ,,
Fe	0·007 ,, ,,

It was not at all apparent why the zinc should act in this way, because as a rule I have found that the presence of zinc in solution has no appreciable effect on extraction. On looking through the list of substances in the foregoing analysis, the antimony and arsenic suggested themselves as being the most unusual, and since the amount of the latter was 10 times that of the former, the arsenic was selected as being a possible conspirator.

Arsenic Also a Hostile Influence.—Another series of experiments was thereupon made in which *F* was treated with new solutions: *G* with new solution to which 0·021% of arsenic was added: *H* with new solution to which 0·1% zinc was added, and *I* with new solution to which 0·021% arsenic and 0·1% zinc were added. Upon assaying the residues ran: *F*, 3·22 oz. Ag. : *G*, 4·90 oz. ; *H*, 4·78 oz., and on *I*, 7·05 silver per ton.

All these results seemed to point to the fact that the degeneration of the working solutions was due to the use of zinc as a precipitant in the presence of arsenic, and accordingly some other method of precipitation was sought. There was a possible choice of at least three other methods: electrolysis, sodium sulphide and aluminium.

As I have had long experience with electrolytic precipitation of cyanide solutions, and am well aware of its weak points, it was considered to be a remedy of last resort, though a possible one. Sodium sulphide would be a very suitable precipitant for silver solutions, except for two drawbacks. The first is that complete precipitation is not obtained without using an excess of the reagent, which would then have to be removed by a lead compound, the alternative being a tailing solution too high in silver to form an effective residue wash. The second drawback is the recovery of the silver as a sulphide instead of in the metallic state. Attention was therefore turned to aluminium.

Aluminium Precipitation Suggested by Moldenhauer.—As far back as 1893, Carl Moldenhauer patented the use of aluminium as a precipitant for gold and silver from cyanide solutions. He says in his specification: 'Zinc has heretofore been employed in practice by preference in precipitating gold from the cyanide solutions obtained by leaching auriferous ores. The employment of zinc for this purpose is found, however, to be attended with serious disadvantages. Now, I have discovered that aluminium can be employed for this purpose in place of zinc without the disadvantages attending the use of the latter. While zinc forms a combination with the bound or free compound of cyanogen and alkali contained in the cyanide solution, aluminium separates the gold very quickly from the cyanogen solution without entering into combination with the cyanogen, but simply reacting with the caustic alkali which is present at the same time. By the action of aluminium, the cyanide of potassium employed for leaching the gold out of its ore is regenerated, which is not the case when zinc is employed. But the zinc does not confine itself to entering into combination with the cyanogen of the cyanogen compounds of the gold, but also acts upon the free cyanide of potassium contained in the solution, so that a great part of the latter is consumed, but this is not the case when aluminium is employed. These results are of the greatest importance when the solution separated from the gold is to be employed in subsequent gold-extracting operations, as the whole of the cyanogen in the regenerated and liberated cyanide of potassium is enabled to renew its action, but the lyes resulting from the employment of zinc cannot be employed with the same advantage in subsequent operations for the extraction of the gold. Numerous attempts have been made to regenerate the zinc, but are found to be inconvenient and costly. It is consequently evident that an important saving in cyanide of potassium is obtained by the employment of aluminium.'

Aluminium Forms No Cyanogen Compound.—The inventor does not appear to have developed the idea on a practical scale, but it is worthy of more attention than it has received, because, as he points out, owing to the fact that aluminium does not form any compound with cyanogen, not only is the whole of the cyanide recovered which was combined with the precious metals, but also the additional loss of cyanide by direct combination with zinc is avoided. In the case of solutions strong in cyanide, the latter loss may be considerable.

Some may dispute the ground I take when I count as lost the cyanide that remains combined with zinc, because it is often stated that the double cyanide of zinc and potassium is almost as efficient for dissolving purposes as the simple cyanide. My experience is, however (at any rate in the case of silver ores), that the reading obtained by the use of KI indicator with excess of caustic, is worthless as a measure of the dissolving power of a cyanide solution, the efficiency being for practical purposes proportional to the 'free' cyanide reading obtained by stopping at the first faint opalescence without the use of KI indicator.

Caustic Alkali Necessary.—The fact that aluminium does not replace the precious metals in the cyanogen compound, renders necessary the presence of a caustic alkali, the reaction probably occurring theoretically thus:

$$6\ NaAgCy_2 + 6\ NaOH + Al = 6\ Ag + 12\ NaCy + 2\ Al(OH)_3,$$

the aluminium hydroxide at once dissolving in the excess of caustic to form sodium aluminate.

$$2\ Al(OH)_3 + 2\ NaOH = Na_2Al_2O_4 + 4\ H_2O.$$

If this explanation be the correct one, it should be possible to make one part of aluminium precipitate 12 times its weight of silver, but in practice I have so far not been able to reach this proportion. Perhaps in the case of low-grade solution, when alkali and aluminium are both present in excess, as must usually happen in practice, the following may more nearly represent what actually goes on:

$$2\ NaAgCy_2 + 4\ NaOH + 2\ Al = 4\ NaCy + 2Ag + Na_2Al_2O_4 + 4\ H.$$

In this case one part of aluminium would precipitate four times its weight of silver, and this figure approaches more closely the actual proportion I have found in practice, which is about three of silver to one of aluminium. For zinc precipitation Clennell gives the following equation:

$$NaAgCy_2 + 2\ NaCy + Zn + H_2O = Na_2ZnCy_4 + Ag + H + NaOH:$$

and according to this one part of zinc will precipitate only 1·7 times its weight of silver. This zinc reaction is also worthy of comparison with that of aluminium from the fact that two molecules of free cyanide appear to be used up in addition to the two molecules of cyanogen in combination with the silver.

The consumption of cyanide by direct combination with the zinc is illustrated by Clennell as follows:

$$Zn + 4\ KCy + 2\ H_2O = K_2ZnCy_4 + 2\ KOH + H_2,$$

The indications then would go to show that quite apart from any detrimental action that zinc may have upon extraction of the precious metal from its ore, the substitution of aluminium should be, on its own merits, a valuable improvement, and I shall try to show later on that this proposition is borne out in practice.

Aluminium Successfully Used at Deloro, Ontario.—Although it is 20 years since Moldenhauer so ably explained the advantages of aluminum over zinc, the commercial application of the process is almost unknown. In 1893, H. Forbes Julian experimented with it (cf. Julian and Smart, 'Cyaniding Gold and Silver Ores'), but does not seem to have met with any practical success. He first used vertical aluminium plates set ¾ in. apart in a standard precipitating box, but, owing probably to lack of surface, the result was unsatisfactory. Afterwards he used shavings, which showed a great improvement, but in time the metallic surfaces became coated with alumina, which impeded precipitation to a prohibitive extent. He was unable to overcome the difficulty, and finally abandoned the attempt to make use of the process.

The first instance I have been able to find of its successful commercial application, was at the Deloro smeltery, Ontario, where it was introduced by Prof. Kirkpatrick for dealing with a rich cyanide liquor resulting from the treatment of some special products. Warned, probably by Julian's failure, and having before his mind the wide use of zinc dust, he conceived the idea of applying the aluminium in the form of a powder, by which means he overcame the obstacles encountered by Julian, and made the process a perfect success.

In 1910, Kirkpatrick took out a patent for a special vat designed to effect a proper mixture of the dust with the solution. The difficulty in the use of aluminium dust is to get it to sink, because it is not easily wetted, and even when wet, tends to rise to the surface of liquid through the buoyancy imparted by minute hydrogen bubbles, unless a violent agitation is maintained. Kirkpatrick overcame this trouble by using a vat having a vertical shaft at its centre, carrying one or more screw propellers, which, when revolved by a suitable power connection, forms a vortex in the centre of the solution to be precipitated.

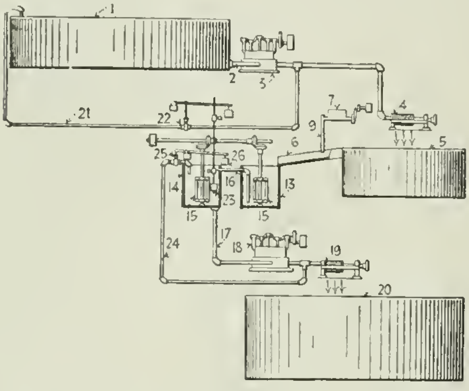
A sufficient quantity of the dust is then added, and as it spreads out on the surface, the vortex sucks it down to the bottom, whence it passes outward towards the periphery and up again to the surface, again to be sucked down at the centre. This motion is maintained until precipitation is complete, when the whole charge is pumped to a filter press for the separation of the precious metal. The same process has been in use at the O'Brien mill in Cobalt for three years or more in precipitating the working solutions in the cyanide plant and has apparently given entire satisfaction.

Rate of Precipitation Depends on Violence of Agitation. Before finally adopting aluminium as the precipitant for the mill, a series of laboratory tests were made, which showed that not only was the combined cyanide liberated, as indicated by titration, but also that no deterioration of the solvent power of the solution was apparent after using and precipitating a number of times.

The above mentioned tests showed, among other things, that the rapidity of precipitation depended to a large extent on the violence of the agitation used for mixing and that whereas a charge shaken up briskly in a bottle would have its reaction completed in about two minutes, a large charge stirred up in a tub with a stick needed from 10 to 15 min., other conditions being equal. It, therefore, seemed that it should be possible to get away from the intermittent system represented by the Kirkpatrick method and work more on the lines of the continuous zinc-dust precipitation. The appliances for the latter process, making use of the Merrill machinery, had already been installed, including clarifying press, dust feeder, emulsifier and two triangular presses. It was evident, however, before making a start that more time for contact would be required than had been provided for the zinc-dust by the 200 ft of pipe line between the emulsifier and the press. It was also evident that it would be difficult to feed the aluminium dust, even as an emulsion, into a stand-pipe connected with the pump suction, because, as already explained, the dust is difficult to wet, and even after it has been wetted, it will soon tend to rise to the surface and float as a thick scum unless a brisk agitation is maintained. If it were fed into a stand-pipe it would collect on the surface of the liquor and gradually fill the pipe instead of being drawn down into the current flowing to the pump. Therefore, the following modification of zinc-dust practice was adopted and found to answer admirably.

Modified Zinc-Dust Apparatus Used for Aluminium.—Referring to the accompanying illustration, 1 is the pregnant solution storage vat, from which the solution passes to a triplex pump and thence to the clarifying press 4. The vat 5 is now superfluous, having been installed as a storage for the zinc dust process, but the clarified solution falls into it as previously arranged, flowing out again at once by the launder 6 into vat 13. The feeder 7 for the aluminium dust, which discharges directly into the stream of solution, flowing in the launder 6. The emulsifier supplied for zinc dust has been eliminated as not serving any useful purpose in the present process. The launder is fitted with a cover and has a flap of canvas hanging over the discharge opening to avoid loss in dusting. Vats 13 and 14 are about 6 ft. deep by 5 feet in diameter, and have a central revolving shaft to which are bolted vertical planks extending from the bottom to the surface of the liquor contained therein. These comparatively narrow vertical strips when revolved at about 60 r.p.m., form an efficient agitator for mixing the emulsion and render unnecessary the propellers with their vortex effect described by Kirkpatrick.

It will be noticed that vat 13 is always full and that the emulsion has to pass down to the bottom and up the 6 in. connecting pipe in order to reach vat 14 and to avoid any tendency for heavy silver precipitate to accumulate in the bottom of vat 13, the revolving shaft has a pair of long arms or paddles about 3 in. wide, bolted on just above the bottom to keep such heavy particles in suspension, until they can be sucked by the current up the vertical pipe into the next vat 14. This latter vat may be considered chiefly as a pump feeder, from which the emulsion passes to the triplex pump 18, and thence to the precipitate press.



Arrangement for Aluminium Precipitation, using the apparatus designed for Zinc-Dust Precipitation with modifications.

Constant Level Automatically Maintained.—In order to maintain a constant level in vat 14 and avoid overflows on the one hand and the entrance of air into the pump suction on the other, a system of automatic control valves operated by a float, is used. It will be seen on examining the sketch, that each pump is provided with a bypass, 21 and 24, and both these are operated by a single float shown at 23 in the drawing. The valve 25, which is usually more or less open, is operated by a long lever which is hinged in such a way that it can be bent upwards from the hinge point, but not downwards, and is normally held in a straight line by the weight 26. The vertical rod passes freely through a slot in the lever and carries a tappet in such a position as to engage the lever at a certain height. When it is desired to increase the flow to the precipitate press, the float will rise in the vat until the tappet engages the lever, thus gradually closing the bypass valve 25 and throwing more work on to the pump. The flow may thus be increased until valve 25 is completely closed, this point representing the full capacity of the pump 18. If then the flow into the tank be further increased, the float will continue to rise and the lever will flex at the hinge, thus causing the upper tappet to engage the lever on valve 22, and by opening the same allow the surplus solution to flow back to tank 1, whence it came. With this device, it is perfectly simple to get any desired flow of solution up to the full capacity of pump 18, maintaining practically the same level in the pump-supply tank. The total time consumed between the point where the precipitant is added and the point where the emulsion enters the press is from 10 to 15 min. on the average, and precipitation is normally completed in this period.

Aluminium-Dust Feed Regulated by Solution Test.—The feed of dust is regulated from time to time by taking a sample of the effluent from the press and adding a little strong sodium sulphide solution to it. If the faintest coloration appears (due to the formation of silver sulphide), the feed is increased little by little until the test indicates complete precipitation. The effluent is tested in this way every hour or so and a fine adjustment of the feed is thus obtained.

It has already been stated that caustic alkali is necessary for this reaction, and therefore caustic soda is added to the pulp before cyanide treatment in sufficient quantities to bring the solution up to from 2 to 3 lb. NaOH per ton. In connection with the alkalinity of solutions, there is an important point to be noted. Lime must be reduced to the smallest quantity necessary for slime settlement, because it tends to form with the dissolved aluminium an insoluble calcium aluminate, which collects in the precipitate press, giving a low-grade product and one almost impossible to melt by the usual methods. It is, therefore, necessary, when contemplating the introduction of this process, to ascertain whether the slime will settle readily in a solution high in soda and low in lime and also whether the use of soda interferes in any way with the efficiency of the solution for extraction. I may say that neither of these points has so far given any trouble at the Nipissing mill.

Precipitation Results Good.—At the time of writing, this process has been in use about four months, so the data available should be fairly reliable. The figures for the first three months of 1913 are here given just as they stand in the official records and include the 'bad days' as well as the good ones.

TABLE 2. Precipitation Results.

Month.	Head Assay, oz.	Tail Assay. oz.	Precipitation
Jan.	8·23	0·23	97·2
Feb.	8·56	0·21	97·5
Mar.	8·50	0·15	98·2

These figures do not represent the full efficiency of which this process is capable, because a few bad assays due to mistakes and oversights, which are unavoidable in starting a new mill, have spoiled the averages. Usually the daily assay stands at about 0·05 oz. silver in the tails, and with ordinary care there is no reason why it should ever run over that figure.

TABLE 3.—Consumption of Aluminium.

Month.	Aluminium Lb.	Silver Precipitated oz.	Aluminium Used per oz. of Silver. lb.
Jan. ...	2926	141237	0·02
Feb. ...	2742	143141	0·019
Mar. ...	3014	142604	0·021

For comparing these figures with those for zinc dust, it is not easy to obtain representative data on the latter owing to the custom of publishing such figures in terms per ton of ore instead of per fine oz. of precious metal. I have, however, come across the following table in the *Journal* of the Chemical, Metallurgical, and Mining Society of South Africa for December, 1909, which may be of some assistance.

TABLE 4.—Consumption of Zinc Dust per oz. Bullion.

Oz. Au + Ag per ton Solution	Ratio Au + Ag	Lb. Zinc Dust Consumed per oz. Au + Ag Precipitated	Remarks.
0·02	2 : 2	6·60	Homestake low solution
0·15	2 : 2 : 1	0·91	Homestake weak solution
0·47	1 : 4	0·59	Cerro Prieto
0·49	1 : 4	0·57	Cerro Prieto
0·70	1 : 4	0·42	Cerro Prieto
1·84	1 : 19	0·19	Montana, W. J. Sharwood
3·29	1 : 99	0·16	An American mill

It is impossible to say what the zinc-dust consumption would have been at the Nipissing low-grade mill, because it was never tried there, aluminium having been used from the start. Table 4 shows, however, that the figure varies enormously with varying conditions and that, as far as the series is carried, the higher the metal content of the solution, the less the zinc dust consumption per ounce, so presumably the consumption on Nipissing solution would have been lower than the lowest in the series. On the other hand, the strength of cyanide in the solution, 0·2%, would tend towards a high zinc consumption by direct combination. All things considered, a consumption of about 0·1 lb. zinc per oz. of silver would probably not be far out as an estimate for comparison with the actual aluminium consumption shown.

TABLE 5.—Regeneration of Cyanide, Average of Daily Titrations.

Month.	Titration % KCN		Gain after Precipitation	
	Head	Tail	Per cent.	Lb. KCN per ton of solution
Jan. ..	0·196	0·228	0·032	0·64
Feb. ...	0·181	0·215	0·034	0·68
Mar. ...	0·179	0·215	0·036	0·72

The regeneration shown above approximates closely the actual amount of cyanide combined with silver in a solution assaying 8·6 oz. per ton. It must not be forgotten, moreover, that the cyanide shown to be actually recovered does not represent the total saving effected, because as already pointed out, there is in the case of zinc precipitation a loss of cyanide by direct reaction with the zinc.

Before we appreciated the deleterious effect on this process of lime in the working solutions, the precipitate presses contained a good deal more calcium aluminate than silver, the latter being sometimes as low as 10%. At present the precipitate assays under normal working conditions, between 25,000 and 27,000 oz. per ton, or from 85% to 92% silver. This product may be melted in crucibles without any flux, yielding bullion about 990 fine. At the Nipissing it is briquetted with a little soda and borax and melted direct in the oil-fired reverberatory, already described in the JOURNAL, producing bullion over 997 fine.

Ultimate Cost of Aluminium Less than Zinc-Dust Method.—The present price of aluminium dust laid down at the Nipissing, including freight and duty (15% *ad valorem*) is from 35 to 39c. per lb., and of zinc dust 7c. per lb. Taking the consumption of the two reagents at the figures already given, 0·02 lbs, per oz. of silver for the first, and 0·1 lb. for the second, the ore yielding 20 oz. of silver per ton would show as in table 6.

TABLE 6.—Comparative Cost of Zinc and Aluminium Methods.

Aluminium Dust.		Zinc Dust.	
0·4 lb. Aluminium at 38c.	$0·152	2 lb. at 7c.	$0·14
Caustic soda, 1·5 lb. at 2¼c.	0·033		
Per ton...	$0·185	Per ton	$0·14

In the figure for caustic soda, 0·5 lb. has been allowed for chemical consumption and 1 lb. for mechanical loss.

To offset the extra charge of 4½c. per ton of ore, there would be a direct recovery of 1·6 lb. of cyanide, at 15c. or $0·24 per ton; a further saving in cyanide, varying in amount with varying conditions, due to the absence of any action between the precipitant and the cyanide, a saving in the cost of melting and refining and in some cases in marketing the bullion. In the case of ores, which on a commercial scale fail to yield the extraction predicted by labaratory work on account of fouling of working solutions, a gain in extraction is obtained which may often amount to 7 , and which in the case of the Nipissing ores was as high as 14%.

Aluminium Granulations Suggested to Replace Dust.—As an alternative to the use of the dust, Charles Butters, consulting metallurgist for the Nipissing Mining Co., has suggested the use of aluminium granulations placed in a revolving tube mill through which the solutions to be precipitated is continuously passed, the attrition produced thereby serving to scour the metal surfaces and thus obviate the principal obstacle encountered by Julian in his experiments. Tests are now being made of this method, the chief advantage of which, if it should be a success, would lie in the reduced cost of precipitant, the price of granulations being only about 24c. per lb. as against 35c. to 40c. per lb. for dust.

In conclusion, I wish to acknowledge the valuable assistance of J. J. Denny, resident metallurgical chemist, in carrying out the various experiments and the facilities afforded by James Johnston, constructing engineer, for putting into practice the ideas conceived: also, to express my appreciation of the action of R. B. Watson, general manager, in permitting me to publish the official data included in this article."—E. M. HAMILTON, *The Engineering and Mining Journal*, May 10, 1913, p. 935. (H.A.W.)

MINING.

THE FORMATION OF METALLIFEROUS DEPOSITS.—"M. De Launay, in a lecture delivered to the Société de l'Industrie Minerale, discusses the genesis of mineral deposits. The following extracts have been made from the bulletin of the society. Modern works, many of which have been written by Norwegians and Americans, accentuate the relation between metallic ores and eruptive rocks. This inter-independence has led to the classification of metalliferous deposits into several groups. If the fundamental hypothesis of relation between ores and eruptives from their origin be accepted, we are induced to establish a corresponding connection between the nature of the deposits and the depth at which they are formed.

Let us consider an eruptive rock that is in course of crystallisation under cover of sedimentary strata. If the magma contains minerals they will manifest a tendency to ascend in the guise of metallic exhalations. Analyses prove that eruptive rocks possess traces of many metals, and that in the case of iron they include as much as 7% or 8%. These metals first form an unimportant aggregation of small disseminated grains that have received the name of 'inclusion deposits.' Some metals thus distributed have been found exploitable, as, for example, platinum, iron, chrome and nickel, and even the diamond.

After the inclusions, the elements group themselves more densely in the midst of the eruptive rock and still at a considerable depth, and we have 'deposits of metallic segregation.' These have a practical importance, since mineralised masses of over 150 ft. thick occur. They are deposits which have been formed at great depths, and can only be encountered in very primitive rocks, such as exist in Scandinavia and Canada, where they are well developed.

Succeeding these magmatic segregations, confined to the eruptives, there follows a different process, that of the deposition of metallic exhalations on the exterior of eruptive rocks. This is commenced by the appearance of sulphide ores massed at the contact of the eruptive with other rocks. They are designated as 'peripheral sulphide segregations.' In the mass of eruptives there are found only small quantities of sulphides but plenty of oxides, whilst at the junction rocks sulphides concentrate and crystallise abundantly. These types of segregation, up to now very rare, constitute important deposits of copper, nickel, etc., which are, for example, successfully exploited in Canada.

Still at a considerable depth, deposits have been produced that appear in the form of lenticular bodies with transverse veins. This type, termed 'diffused impregnations,' is of great consequence. There are certain regions with veins enclosing sometimes copper and often gold; they are troublesome to work, since they are very irregular and prone to peter out suddenly. Notwithstanding, they are considerably exploited on account of concentrations, due to earth movements, and give rise to alluvial deposits through disintegration.

We now reach the true fissure type of mineral deposition, at first those which are directly connected with eruptives crystallised at great depth, then with more superficial deposits associated with microgranites, and, lastly, microlithic. With granites exhibiting white mica, tin and its group is developed, sometimes accompanied by copper; then follows copper associated with lead and zinc, and, still higher, mercury—all in the true fissure group.

At surface little ore is formed directly from the rocks: these are only the results of mineral concentration derived from the denudation of previously existing deposits. The predominating proportion of minerals appear to have been crystallised at immense depths under varying conditions of heat, pressure, etc. We are now enabled to apprehend that each type of deposit corresponds to a determined and limited zone in the earth's crust below which its occurrence is improbable, and this tends to explain the wedging out of any metalliferous type when exploitation has been commenced too near its confines in depth.

We have just remarked that the occurrence of mineral deposits may be classed as inclusions, segregations, peripheral segregations, impregnations, and veins of fracture. A question arises, have these different deposits remained from the time they were deposited in the same state as we now find them, and are they produced in any region where similar conditions prevail? It is to be observed that deposits are not the same in all regions; and if a miner, for example, were transported from the basin of the Mediterranean to Norway, he would fail to recognise the types to which he had been accustomed, and if he endeavoured to apply brusquely the empirical results of his previous experience to these new types, he would have every chance of going wrong. Each region possesses, then, its own peculiar aspect, generally very different from others, and this is especially the case in passing from the north to the south of Europe. On the other hand, some parts remote from each other display certain analogies. Thus the same types obtain in the central plateau, Bohemia, or Saxony, and the geologists of Freiberg have noticed in Asia, especially in the Altai, deposits similar to those in their own district.

We can, therefore, arrange deposits in metallogenic provinces which are adjusted to the age of the corresponding tectonic zones, and that have successively permanently crystallised under the influence of regional metamorphism, which have since suffered only vertical movements and have usually remained without any further mineralising action. In Europe there can be distinguished from north to south a Huronian zone, a Caledonian zone, a Hercynian zone, and a zone influenced by the folding of the Alps; their equivalents exist also in other continents. In the Huronian and Caledonian of Scandinavia, the types of inclusions, segregations and impregnations are exceedingly well represented, and their equivalents are present in Canada, Brazil, India, Siberia and Central Africa. In the Hercynian of Central Europe has been developed the petrographic type of porphyries harmonising with a semi-profundity characterised by the occurrence of veins containing blende, galena and pyrites. More to the south are the folded Tertiary formations, accentuated by numerous volcanic intrusions, in which are found auro-argentiferous minerals and also mercury, a metal that is able to deposit itself in proximity to the surface. If we examine a particular region, outside the deeply-seated igneous masses, we shall find metalliferous aureoles showing an analogous succession—first tin, followed by copper, zinc and lead, and finally mercury. It should be observed that for each of the types we are led to class the metals by groups which overlap each other, the natural sequence, however, persisting. It must not be imagined that each metal occurs as an isolated deposit, but as forming part of a whole, so that it would be a blunder to have a concession for a single mineral.

We now pass to another question. As deposits are not independent of their position in space, neither are they so in their place in time. Since their formation metalliferous deposits have undergone modifications and have not been able to remain in the state in which they were crystallised out in remote ages. These changes have been determined by their approximation towards the surface, and the effects of superficial action must be conceded to extend to a depth of several hundred yards. Fissure veins have a limit both in depth and in height, and if we find them cropping out at surface it is because denudation has removed more or less of the crust and exposed them to-day. This exposure has induced reactions in the upper parts of lodes which have gradually transposed minerals to continually deeper portions of the veins.

There is everywhere a point beneath the surface where water is reached, and below this horizon aqueous actions and movements are decelerated. The nature of the region above this hydrostatic level is entirely different, and throughout this space the surface waters, charged with oxygen, carbonic acid, etc., percolate, reacting on rocks and minerals, and issuing from the surface as springs. It is interesting to note the alterations which are produced in this upper portion, and to a certain distance also under the hydrostatic level, by the incessant circulatien of superficial waters. The present was preceded by a higher surface, the vein extending up the interspace, and in which certain ores were placed in solution—for example, copper; this dissolved copper was partly washed away by the rains, but a portion was also precipitated in the vein at some distance beneath. This action continued to the gradual enrichment of the lower zone by cementation. At a varying depth under this zone the unaltered vein becomes, more or less, suddenly poorer. Gold lodes are affected in a similar manner, the lode above the unaltered zone assuming an abnormal productiveness, and the same for silver lodes. These phenomena entail important consequences, of which numerous instances could be cited; a prominent case will be sufficient. When ores are enclosed in calcareous strata these reactions attain a maximum of intensity; thus one is able to refer the development of calamine deposits to the phenomenon of cementation. They are the result of the transformation of blendes, and are not formed in depth. Similar action changes spathic iron to iron oxide, which is shown in iron cappings and in the extensive masses of hematite worked at Bilbao and other places. In the mines of Leadville galena has been converted into cerussite, but this action is less frequent than for zinc and iron, since the compounds of lead are less soluble.

To sum up, in the presence of metalliferous deposits it is essential to attribute considerable significance to this vadose action; if the country is accidented and if the water level is at a great depth, as in the Andes and Mexico, such alterations may reach a depth of 900 to 1,200 ft. The study of these transformations is interesting from a theoretical point of view, as it affords an explanation of facts inadequately interpreted by the theory of lateral secretion. It is possible thus to construe the presence of these enrichments, formed at an epoch almost contemporaneous, of bones of modern animals which have been enclosed in them. The alterations of minerals much more ancient is confirmed by similar crystallisations of sulphides and oxides often produced in old and abandoned mines. When we are prospecting a new country, we are enabled to grasp some knowledge of its general tectonic character, and to form an opinion on the nature of the enclosed ores, and to determine in what direction operations should be undertaken."—*London Mining Journal*, June 28, 1913, p. 635. (A. R.)

COMPRESSED AIR *v.* HYDRAULIC POWER FOR ROCK DRILLS.—"During the past 20 years numerous attempts have been made to produce a drill that would drill rock on an expenditure of 1·5 h.p. to 2 h.p., which is supposed to be the actual power required at the drill bit. Any number of electric rock drills have from time to time been launched on the market with very alluring claims as to efficiency, etc., furnished with motors of 1·5 h.p., 2 h.p. or 2·5 h.p. The proposition was so tempting that many of these drills found their way to the mines, but they must have long since passed through the scrap heap, for we hear no more about them in Mexico.

The fact of the matter is that nothing has yet been found that lends itself so well for operating a rock drill underground as compressed air, due to the many advantages it offers over any other form of power now available.

The Temple-Ingersoll electric-air drill, as its name implies, is a combination of air and electricity. It is compressed air that reciprocates the piston in a similar manner to the well-known air piston drill. The duty electricity performs in this case is simply to actuate the pulsator which impels the compressed air back and forth alternately to one and the other end of the drill cylinder; thus the air once compressed in the pulsator to abour 30 lb. is simply pushed back and forth through the air hose, forming an air spring which imparts to the drill piston its reciprocating motion. There is no exhaust from this drill, the same air being used continuously in a closed circuit. The novel features of this apparatus, that recommend it on the score of over-all efficiency, may be enumerated as follows:—

1. There is no heating of the air, as in an air compressor:
2. There is no clearance spaces, as in the compressor:
3. There is no exhaust;
4. There is no loss of pressure due to friction of the air through long lines of pipe with many sharp turns, elbows, valves and other restrictive fittings.
5. There is no loss of volume due to many small leaks which are usually found in such an air line.

The size electric-air drill that corresponds in drilling capacity to a 3 in. or 3¼ in. cylinder Ingersoll piston air drill is known as No. 4-E and is operated by a 5 h.p. electric motor. Under ordinary running conditions the drill does not consume more than about 4 h.p., but a 5 h.p. motor is provided to overcome difficult emergencies, such as the binding of the steel in a hole, etc.

This drill was a success from the moment it was introduced some six years ago, and several hundred of them are in use, not only in the United States, Canada and Mexico, but also in South America and in Europe.

Yet, notwithstanding its undeniable success in many places and its notable economy in power—requiring but one-third to one-half the power consumed by an air drill of equivalent drilling capacity—it is a fact that (with one or two exceptions) none of the most important mines in Mexico have adopted this machine or even considered it worth while giving a practical and thorough trial, which goes to show that a high percentage or power efficiency is not the first desideratum looked for in a rock drill by mine managers and superintendents. There are other advantages to be considered by them in laying out a plant for underground work. For instance, the question of ventilation plays an important part, as the exhaust from an air drill supplies fresh and cool air for the men, while the smoke and gases, after a blast, can be quickly blown out with the air from the pipe line—advantages not obtainable either with electric-air drills or with hydraulic drills.

Another reason that doubtless favours the use of air piston drills is the fact that in most large mines, in addition to the piston drills for development work, a large number of telescope-feed air hammer drills are used for stoping, and also some hammer-plug drills employed for various kinds of work, such as block-holing, cutting hitches, sinking small shafts, etc. Then again, many mines now use mechanical drill sharpeners. All these machines are operated with compressed air, and I do not know that any such machines are being built to be driven by water. It is not likely that any mine manager would favour the idea of introducing a double system of pipe lines throughout the mine—one for water for the hydraulic drills and another for air for the hammer drills. Whether hydraulic rock drills would show sufficient saving in power to more than offset advantages offered by air drills remains to be demonstrated.

To my knowledge, hydraulic drills have never been used in Mexico, and we have not heard of their having been adopted anywhere in America, for which reason we lack practical experience with that system." — G. A. DENNY, Mexican Mining and Metallurgical Institute.—*Mining and Engineering World*, May 31, 1913, p. 1040. (A. R.)

MINING METHODS AT PORCUPINE. "Winning the gold at Porcupine presents no very novel difficulties to overcome, so the modes of mining used are the standard methods adopted on somewhat similar conditions in different parts of the continent. Two different types are exemplified in the Dome and the Hollinger, the former using the 'Open-cut' or 'Glory Hole' method, and the latter, the regular underground method, with the "Shrinkage stope' system. In both systems the initial work is usually similar. After the timber is cleared off, the surface is stripped or trenched to locate the veins. Vertical shafts are then sunk at suitable points, and levels are opened up, usually at 100 ft. levels, though intermediate levels are sometimes run for local reasons. Reciprocating air piston drills are used, the favourite size being the 3½ in. machine. Except when the stopes are wide, the small hammer drills have come into very general use for stoping.

In some shaft sinking recent practice favours the adoption of small plugger drills in preference to the heavier reciprocating piston drills. The advantages claimed for the plugger drill are :—

1. Lighter and therefore more easily, handled, hoisted or lowered.

2. Only three men are required to handle two machines, instead of four men for two of the piston machines.

3. Time taken to set up and tear down the larger machines is saved, and the actual drilling time is thus increased.

4. A greater footage of drilling is even claimed for the smaller machine in a given length of drilling time.

5. Fewer holes can sometimes be used to break a round, by drilling a few holes at a time, and then blasting. Advantage can thus be taken of any peculiarities in the breaking of the rock. If this method were tried with the heavier machines, too much time would be consumed in setting up and tearing down

6. Small consumption of compressed air.

In the harder ground and particularly in the solid quartz the piston drill is still used for sinking on account of the large number of steel breakages (due doubtless to crystallization) and the consequent loss of holes.

In stoping when the ore is 20 ft. wide as is frequently the case in No. 1 vein, the ore is broken down in benches by long almost flat holes, piston drills being used.

32 holes, 7 ft. deep are employed to break a block of ore 20 ft. × 10 ft. × 7 ft., using from 6 to 7 sticks of powder per hole.

At the Dome mines the ore consists of large masses of quartz and schist, frequently alternating, banded and much mixed. The presence of quartz seems to be a necessary accompaniment of the gold values, though it is rather along the small contacts of the quartz and schist that the gold occurs, than in the quartz itself. The ore is, however, so mixed that very little sorting can be done, and the whole mass is sent to the mill if the average is of milling grade. Two 3-compartment vertical shafts have been sunk, No. 1 to 100 ft. and No. 2 to 250 ft. Sinking of No. 2 shaft is to be continued. A double tracked incline 568 ft. long connects the 45 ft. and the 100 ft. levels with the surface, and the mill. From the surface to the 45 ft. level the grade is 13·55%, and from the 45 ft. level to the 100 ft. level 18·19 .

The 45 ft. level was divided into 100 ft. rectangular blocks of ore by drifts and crosscuts. A number of raises were then run to the surface with a chute at the foot of each. The ore which was broken down from above in benches was drawn off through these chutes, trammed by mules to the foot of the incline and hoisted into the mill. As the ore is mined and drawn off, inverted cone-shaped excavations are formed. Fourteen of these open pits are gradually running together, making one large open pit or "Glory Hole." No timbering is done except in shafts and chutes.

Timbering employed is very simple. Square sets are used in the shafts. Stulls and lagging are placed in the stope above the drifts and the ore is broken down on to the timber by overhand stoping. Work is carried up on the broken ore, only sufficient being drawn off below through chutes to keep the broken rock far enough away from the back to allow work to proceed.

The Hollinger mine is being opened up by levels 100 ft. apart, the lowest at present being the 300 ft. The main 4-compartment shaft only goes to the 200 ft. level, but work will soon be resumed to connect it with lower levels. A winze connects the 200 ft. and 300 ft. levels. Underground workings, including drifts, crosscuts, shafts and winzes now aggregate over 1½ miles.

The following table will give an idea of the drilling averages in the different rocks at this mine. The drifts will average about 6 ft. × 7½ ft. and the machine used is a No. 43 Rand 3¼ in. :—

Drilling Averages in Different Rocks at Dome Mine.

Rock.	Holes. No.	Holes. Depth. ft.	No. sticks.	Description.	Depth. Broken ft.	Time. hr.*
Quartz and schist in porphyry ...	15	5	130	40% dynamite	4·0	14
Quartz and schist in basalt ...	14	5	110	40% dynamite	4·5	10
Quartz alone ...	19	5	160	40% dynamite Some gelignite in cut ...	3·5	18
Schist with dip	12-14	5	90	40% dynamite	5·0	15
Schist against dip	15	5	115	40% dynamite	3·75	18x
Powder 1¼						

Piston drills are used underground for drifts, cross-cuts and shafts ; hammer drills for raises, and small pluggers and piston drills for stopes.

In drifting in the schist it requires 17 to 18 holes 6 ft. deep to bring the round. The bottom cut is generally used. In some of the drifts where the ground is blocky the 4 hole diamond cut is used to advantage. The average drilling per shift is from 8 to 9 holes.

In drilling in the quartz, the same number of holes is used, but six to seven 5 ft holes is a shift's work. The average amount of powder used per round is 150 sticks, 50 of this being gelignite used in the cut and the remainder 40% dynamite for the square up. The machine used is the Rand 3¼ in.

In the glory hole stoping six plugger drills (Rand BC 26) are used to break ore for the mill. The average footage drilled per machine is approximately 56 ft. The powder consumption averages about 8 sticks per hole of C.L.X. Special, the holes having an average depth of 8 ft.

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x 3 hr. lost and 15 sticks of dynamite used in reblasting cut.

* This time includes drilling, shooting and setting up.

The total underground workings now amount to nearly 1¼ miles, without counting shafts and raises.

A 75 h.p. geared electric hoist serves the inclined shaft while a similar hoist of 250 h.p. capacity is installed at No. 2 shaft. At present, power is supplied by seven 150 h.p. Babcock & Wilson boilers, fired with soft coal. Two 375 K.W. Allis-Chalmers generators run by two Robb-Corliss engines give the required electrical power for operating the mill, hoists, etc., and two 12-drill Ingersoll-Rand air compressors supply the mine with the necessary compressed air."—A. A. Cole, *Canadian Mining Journal*, June 15, 1913, p. 375. (A. R.)

MISCELLANEOUS.

The Relation of the Mine Manager to Recent Graduates.—"Before the days of technical schools, there prevailed what was practically an apprentice system. A young man desirous of becoming an engineer, began his training in an engineer's office. He received no salary at first; instead, he paid a fee. He copied papers, specifications, estimates, etc., made drawings or maps of tracings and served as assistant in different capacities, according to his fitness and abilities. In the course of time he learned the elements of his profession. Occasionally, when not especially busy, the master engineer imparted knowledge directly, as a teacher. This was not required, however, and usually not even expected. The progress made by the apprentice depended on his own industry and capability. Given an earnest, hard working boy in the office of an able and conscientious engineer, with a large practice and we find the apprentice system at its best. It was the privilege of the apprentice to observe and study the current of professional business passing through the office; to get all he could out of it.

It is clear that this mode of learning the practice of engineering was inefficient and wasteful of time. It was indirect, haphazard and largely dependent on the personal equations of master and pupil.

The place of the apprentice system has now been taken by the engineering schools. These schools have attained a degree of efficiency not dreamed of two generations ago. At graduation the student of ordinary ability and industry has a grounding in mathematics, the natural sciences, and the different theoretical studies constituting the foundation of modern engineering, which the apprentice of the same length of service under the old regime could not possibly possess. What the young graduate of the present time lacks is experience and practice in applying his theoretical knowledge. In this, the technical school gives him a start through the medium of laboratory work, drafting and field surveying, together with some weeks of vacation work, in shops or mines or metallurgical establishments.

In the well organised engineering school the four undergraduate years are a strenuous period. All of the available time is crowded with the numerous courses of study which now make up the training of the engineering student.

Confining the discussion more particularly to our own professions, modern requirements demand that the student of mining or metallurgy shall have acquaintance not only with the specifically mining and metallurgical subjects, but that in addition to his training in the fundamental studies, common to all fields of engineering, he shall be given some knowledge of chemistry, and certain branches of civil, mechanical and electrical engineering. The young graduate then goes into the field to get his practical experience. It cannot be questioned that this practical experience is obtained in a measure at the expense of his first employers.

Mine managers have not infrequently expressed the opinion that this condition of things is wrong and unfair to the employer of the young engineer; and that the mining school should do something to bridge over the gap existing between the end of the school training and a condition of reasonable efficiency and usefulness in practical work. The implication in the manager's criticism is that the shortcoming might be remedied if the teachers of engineering would only take the necessary steps. Perhaps it is too much to expect the average mine manager, or other employer of embryo engineers, to have in consideration the conditions which formerly prevailed and to which I have already alluded. If the ground to be covered in the school were less extensive, undoubtedly more time could be devoted to the practical side of the student's work.

The employer's criticism cannot be answered in a word; possibly it is hopeless to expect a wholly satisfactory solution. My own opinion may be expressed as follows:—

1. That the wiser course is to train the student, as thoroughly, as time will permit, in fundamental subjects upon which all engineering practice rests; special attention being given to those subjects, like mathematics and the natural sciences, which are taught and learned with relative facility in the school, where the student has the advantage of instruction by specialists, and the use of expensive laboratory and other equipment. If these subjects are not learned, or at least well begun, in the school, they will rarely be mastered in later life.

If the justice of this view be granted, and realising that more 'practice' in the student years can be given only by reducing the time devoted to fundamentals, it would appear that the criticism of the employer of the young engineer is not well founded.

2. Another important consideration is this: Would it be possible in the school to give the student a sufficient variety of 'practical' work, to enable him at once after graduation to meet the manager's demands in whatever line of duty the young engineer might be called on to serve?

This would require special preparation. The young graduate might conceivably specialize in one or two directions, but could not prepare himself to fill immediately after graduation a position of responsibility. It is only reasonable that he should begin in a subordinate capacity. His *adaptability* is much more a matter of personality than of training; but, by reason of his school training, he can easily and quickly be broken in to practical work of whatever kind; and, [after all, what an employer chiefly wants in the young engineer is good, common sense.

It hardly seems to be open to question that the mining school training should be an all-round, training, not that of a specialist.

3. It has been held by some that, after completing the course of study in the mining school, the young engineer should have additional training and experience in the *detail* of mining and metallurgy, and of management, both on the technical and business side. But where can such training be so efficiently secured as in a mine or metallurgical works, under a competent superintendent or engineer? The young graduate will there get the real thing and be spurred on by the consciousness of personal responsibility for failure or success.

It is true that apprentice schools are satisfactorily maintained by some of the large engineering and manufacturing concerns; but these schools are the works themselves, not establishments conducted for educational purposes.

Suggestions have been made that if means were provided a more or less extended period of field study might be established at some mine, operated by the teaching staff of the school with the aid of students. Aside from the cost, the greatest objections to this proposal are that the work done would lack both variety and reality. It seems to me impossible that the independent mine or metallurgical works, operated for educational purposes, can ever be an economic success; and failing in this it would not serve the purpose desired.

In conclusion, I do not think the view that young graduates of mining schools are inefficient and unpractical, is held by any considerable number of employes. Facts prove the contrary. Suitable positions are quickly obtained in nearly all cases. The so-called practical man is no longer sought after. Most employers realise that, provided a young man is of reasonably good mental and physical quality, his technical training has not only afforded him the requisite knowledge to start him well in his profession, but that it has developed his powers to a high degree of receptivity, making him capable of promptly rising to the occasion and of meeting the demands upon him."—ROBERT PEALE, *Mining and Metallurgical Society of America*; *Mining Science*, July 1913, p. 27. (A. R.)

PARAFFIN AS A MOTOR CAR FUEL.—"A good deal is heard from time to time about paraffin as a motor car fuel, but a perfectly satisfactory method of carburetting so that a car can run, start, stop, and start again entirely on this liquid, seems still required. The range of ignition is much smaller than with benzine, and the mixture of paraffin vapour and a little air has to be heated before entering the cylinder and meeting the main air supply. At this stage paraffin unfortunately tends to 'crack,' giving a very uneven ignition; and there is also a strong tendency to deposit carbon on cracking, both in the preheating arrangements and in the cylinder itself." —R. J. DUNDERDALE, B.A., A.M.I.C.E., *The Chemical World*, June 1913, p. 201. (J. W.)

STATE GOLD RESERVES.—"Figures are given by the *Echo des Mines*, in its issue of March 24, showing the increase in the reserves of gold held by 27 nations and provinces in the eastern and western hemispheres. The reserves of these States at the close of 1900 were £485,825,800, and at the close of 1910 they were £867,014,000. Of the total gold production of the world during that decade, which is figured at £758,200,000, the increase of £381,188,200 in State reserves represents slightly more than 50%." —*Mining Science*, June 1913, p. 319. (A. R.)

GRAPHITE INDUSTRY OF MADAGASCAR.—"H.M. Consul at Antananarivo reports, on the authority of the Chief of the Mining Department of Madagascar, that the graphite industry in the island is making considerable progress, the quantity exported during the first quarter of this year amounting to 980 kilogs., as compared with 474 kilogs. during the corresponding period of last year, and 259 kilogs. for the corresponding period in 1911. It is believed that by 1916 the quantity exported will reach 20 metric tons annually."—(*Board of Trade Journal*, June 12, 1913), *Journal of the Society of Chemical Industry*, June 30, 1913, p. 656. (J. A. W.)

THE AFRICAN PALM OIL INDUSTRY.—"Considerable progress has been made in the installation in West Africa of machinery for the extraction of palm kernels and palm oil under European supervision. In the present article the results of the examination of samples of palm fruits, nuts, kernels, and oils from the Gold Coast, S. Nigeria, Nyasaland, Uganda, and Mozambique are given. It is stated that recently 'mixed' or 'semi-hard' oils have been shipped from W. African ports from which formerly only the more valuable 'soft' oils were exported. The 'soft' oils yield more glycerine as a by-product in soap manufacture than the 'hard' oils. An investigation carried out by the Agricultural Department, Gold Coast, 1912, showed that when palm fruits are allowed to ferment for a considerable period, they yield a 'hard' oil of much lower commercial value than the 'soft' oil prepared from the fresh fruits."—(Bull. Imp. Inst., 1913, *11*, 206), *Journal of the Society of Chemical Industry*, August 15, 1913, p. 797. (J. A. W.)

Reviews and New Books.

DETERMINATIVE MINERALOGY. By J. VOLNEY LEWIS. Cloth, octavo, pp. 150, illustrated. (New York: John Wiley & Sons; London: Chapman & Hall). Price 6s. 6d net.

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PRACTICAL STONE QUARRYING. By ALLAN GREENWELL and J. VINCENT ELSDEN. Cloth, octavo, pp. 564, with many illustrations. (London: Crosby Lockwood & Son.) Price 12s. 6d. net.

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RAND METALLURGICAL PRACTICE, VOL I. Second Edition. By W. A. CALDECOTT and OTHERS. Cloth, octavo, pp. 490, with many Illustrations. (London: Charles Griffin & Co.) Price 21s. net.

"In our issue of April a year ago we extended a welcome to this notable addition to metallurgical literature, and we are not surprised to find that our view has been shared by engineers generally, as is evidenced by the fact that a new edition has already been required. We may also include the publishers when distributing our congratulations on the success of the book, for to them is due no small share of this success. The contents of the new edition may appear to the casual observer to be practically the same as those of the first, but we can assure our readers that there are multitudes of detailed improvements, additions, and corrections; we have seen the proof sheets, so we know.

For the benefit of those who are not fully aware of the nature and object of this book, we may say that it has been written by a number of experts in Rand practice, headed by W. A. Caldecott. The first chapter is by Ralph Stokes, and contains an outline of the history of the Rand, its geology, and economic problems. Mr. Stokes is a well-known author, journalist, and mining engineer. He was one of the underground staff of the Crown Mines, and is now with the Canadian Exploration Company. The second chapter describes the sorting and breaking stations, and is written by J. E. Thomas. The chapter on the cyaniding of sand is by the same writer. G. O. Smart describes the stamp mill practice, and W. R. Dowling takes as his subject the tube mill and slime department. H. A. White discusses the principle of slime treatment, E. H. Johnson precipitation, cleaning-up, and smelting, and A. McArthur Johnston assaying and testing. Finally, W. A. Caldecott reviews the chemical and metallurgical problems connected with the treatment of banket ore. We have no doubt the book will continue to be a good seller for a long time; it is certainly a work that will be of permanent importance."—*Mining Magazine*, July, 1913. (A. R.)

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Lewes, Prof. Vivian B. Oil Fuel. Illustrated. 12mo., pp. 262. *Collins.* Net 1s.

Lewkowitsch, J Chemical Technology and Analysis of Oils, Fats and Waxes. Vol. I. Illustrated. 8vo., pp. 692. *Macmillan.* Net 25s.

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White, Herbert G. Telephone Erection and Maintenance. A Hand-book for the Contractor and Wireman. Cr. 8vo., pp. 138. *S. Rentell.* Net 1s. 6d.

Wickham, Louis, and Degrais, Paul. Radium, as Employed in the Treatment of Cancer, Angiomata, Keloids, Local Tuberculosis and Other Affections. Cr. 8vo., pp. 118. *Adlard.* Net 2s. 6d.

Willcocks, Sir William. The Assuan Reservoir and Lake Moeris, with text in English, French and Arabic. Illustrated. Royal 8vo. *Spon.* Reduced to net 3s.

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Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 412/t3. Hans von Kramer. Improvements in means for the regulation or construction of railway and like traffic. 26.8.13.

(C.) 414/13. George Spence. Improvements in vacuum filters of the drum type, as used in beneficiating precious metals, and for other industrial purposes. 29.8.13.

(P.) 415/13 John Conlong. An apparatus to act as air filter for gathering and allaying dust and the like in mines. 29.8.13.

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(P.) 418/13. Frank Ondra (1), Max Pollak (2). Improvements in percussive tools with detachable cutters. 30.8.13.

(C.) 419/13. Clifford Edwin Dowling (1), John Frederick James Palmer (2). Improved means to prevent bottles being refilled. 30.8.13.

(P.) 420/13. George Haggis. Improved air respirator for prevention of miners' phthisis. 1.9.13.

(P.) 421/13 Ernest Arthur Ford. Improvements relating to the purification of mine air. 1.9.13.

(C.) 422/13. Dr. Ernst Friederich. A tungsten incandescent lamp. 4.9.13.

(C.) 423/13. Charles Lyman Tolles. Improvements in and relating to traction belts. 4.9.13.

(P.) 424/13. Henry Robertson Lawson. Improvements in assay furnaces. 4.9.13.

(C.) 425/13. Raoul Pierre Pictet. Improvements in and relating to the manufacture of hydrogen and hydrogen mixtures. 4.9.13.

(C.) 426/13. Raoul Pierre Pictet. Process and apparatus for the separation of gaseous mixtures into their constituents. 4.9.13.

(C.) 428/13. Prince Zachari Andronikoff. Improvements in automatic locks. 4.9.13.

(C.) 429/13. Clifford Toon. Means for use in caulking lead joints and iron pipes and mains. 4.9 13.

(P.) 430/13. B. Gray (1), A. B. Smith (2). Improved clamp bracket for securing electric cables in shafts. 8.9.13.

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(P.) 435/13. Harry Silburn. An invention for the protection of pneumatic tyres used on the wheels of motor cars, motor cycles, bicycles and like vehicles. 13.9.13.

(P.) 436/13. Robert Brown. A safety device applicable to skips or cages used in incline or vertical shafts. 11.9.13.

(P.) 437/13. Andrew Allen. An improved apparatus for purifying the air in mines and works. 11.9.13.

(C.) 439/13. John Godfrey Parry Thomas (1), The Thomas Transmission, Ltd. (2). Improvements in and connected with the suspension of rail car bodies and the like. 11.9.13.

(C.) 440/t3. Callender's Cable and Construction Co., Ltd. (1), Chas. Warre Kay (2). Improvements in supports for electrical conductors. 11.9.13.

(P.) 441/13. Edward Hector Lonsdale Hart. Improvements in or relating to acetylene lamps. 13.9.15.

(P.) 442/13. George Anderson Robertson (1), Douglas Anderson Robertson (2). Improvements in mills or apparatus for crushing ore and other materials. 13.9.13.

(P.) 443/13. Alfred Salmon. Spark arrestor to prevent escaping from the smoke stack of a locomotive or other steam engine. 15.9.13.

(P.) 444/13. Ernest Arthur Ford. Improvements in means for binding the surfaces of sand dumps and the like. 17.9.13.

(P.) 446/13. Waleter E. Letty. Guide rail and adjustor for sanitary buckets. 18.9.13.

(P.) 447/13. Ernest Arthur Ford. Improvements in means for binding the surfaces of sand dumps and the like. 20.9.13

(P.) 418/13 Guy Aubrey Chalkley. Improvements in rock drilling tools. 20.9.13.

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Vol. XIV. OCTOBER, 1913. No. 4.

Proceedings
AT
Ordinary General Meeting, October 18th, 1913.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, October 18th, 1913, Mr. Alex. Richardson (President) in the chair. There were also present:

37 Members: Messrs. J. E. Thomas, H. A. White, T. Donaldson, J. Gray, E. Pam, John Watson, A. Whitby, Dr. J. Moir, J. R. Williams (Members of Council), A. S. Allam, P. von M. Anderson, S. Beaton, H. C. Boydell, Prof. J. Cellier, L. L. Ellis, R. C. Evans, B. C. Gullachsen, W. H. Jane, W. E. John, W. A. Ledingham, A. Lilley, H. H. Morrell, P. T. Morrisby, F. S. Pearce, W. S. V. Price, E. Roberts, D. F. Scott, A. L. Spoor, A. R. Stacpoole, W. A. C. Tayler, R. M. Taylor, W. E. Thorpe, J. H. Veasey, A. J. Walton, E. M. Weston and W. O. Williams.

15 Associates and Students: Messrs. G. F. Ayers, J. Davidson, C. L. Dewar, O. A. Gerber, J. Gibson, H. Godwin, R. W. Irwin, F. D. Long, W. S. New, R. Ramsay, V. E. Robinson, P. Scatterty, I. R. Trollip, W. Waters, and L. A. Womble.

21 Visitors, including Mr. H. Warington Smyth (Secretary for Mines), and Fred. Rowland (Secretary).

NEW MEMBERS.

The Secretary announced that since the last meeting of the Society the following Associates had been admitted by the Council:

HIGGINS, JEROME B., South American Development Co., Guayaquil, Ecuador, South America (Box 655). Mining Engineer.
MCPADDEN, JOHN FRANCIS, Reefton, New Zealand. Director Reefton School of Mines.
MERCER, JOHN W., 15 Broad Street, New York, U.S.A. Mining Engineer.

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

GENERAL BUSINESS.

THIRD ANNUAL MINING EXHIBITION.

The Secretary: As you have seen in the advertisement pages of the *Journal* the preliminary announcement is printed regarding the Third Annual Mining Exhibition. We wish to bring it to the notice of members at this early date, in order that they may have sufficient time to prepare any exhibits they may desire to bring before the public at that Exhibition. Further particulars will be published at a later date.

AMBULANCE COMPETITIONS.

The Secretary: As requested by the Council of this Society, the South African Red Cross Society has agreed to undertake the arrangements for the Ambulance Competitions this year. The preliminary dates have been arranged as follows:

Colliery Shields	Competition,	7th December.
Surface	,,	14th December.
Underground	,,	21st December.

If there is a large number of entries, there may be some elimination trials prior to those dates. The conditions for these competitions are laid on the table, and copies will be posted to each Mine Manager, Hospital Superintendent, and Mine Medical Officer, during the ensuing week.

ANNUAL DINNER.

The Secretary: The Annual Dinner of the Society will be held on the 22nd November, at the Carlton Hotel, and the tickets will be £1 each, exclusive of wines. This year the Council has resolved that ladies may be invited as guests of the Members at this function, and it is hoped that there will be a very large attendance.

VISIT TO UNION OBSERVATORY.

Mr. H. A. White (*Vice-President*): I should just like to refer to the Society's appreciation of the visit which we paid recently to the Union Observatory, and to express our thanks for the kindness and hospitality shown to us by the Director, Mr. R. T. A. Innes and Mrs. Innes, Mr. H. E. Wood, and the staff generally.

DISASTER AT THE UNIVERSAL COLLIERY.

The President: You will all have read of the terrible disaster which has overtaken the underground workers of the Universal Colliery, Cardiff, which, it is feared, has caused the death of 417 colliers, thus making it the worst that has happened in Great Britain for 70 years. As Members of a Society which is intimately connected with mining, we are deeply grieved at this calamity, and I am sure you will all join in passing a vote of condolence to the relatives and friends of those who have lost their lives.

The vote was passed in silence, all the members standing.

NOTE ON A PROPOSED METHOD OF ASSAY OF TIN ORES WITH SOME REMARKS ON THE ELECTRICAL CONDUCTIVITY OF MINERALS.

Mr. A. Whitby (*Member of Council*): In the course of some investigations of Beringer's method of tin assay by heating the ore mixed with oxide of zinc and metallic zinc, which we found somewhat impracticable, Mr. Littlejohn suggested the wet treatment of the ore with zinc fume and hydrochloric acid. It was found that under certain conditions many South African cassiterites lent themselves readily to this mode of attack. Some results obtained were as under:

	Pearce Low.	New Method.
Rooiberg concentrates	75·46	75·46
Swaziland	72·65	73·0
German S.W. African cassiterite ...	73·2	72·8
Rooiberg ore ...	3·3	3·07
,, ,, ...	4·14	4·07
Unknown	30·5	29·04

The method adopted is to cover the bottom of a 20 oz. flask with about 2 gm. zinc fume, mix intimately the very finely agated ore with 3 to 5 gm. of the fume, finally adding about 2 gm. of reductor zinc. The charge was slightly moistened with water and concentrated hydrochloric acid poured down the sides of the flask drop by drop, maintaining a brisk action without the introduction of so much liquid as to cause the zinc to become separated and float, or in other words keeping the mass in a pasty condition. More and more hydrochloric acid was added until from 50 to even 100 cc. had been used and when action had subsided the flask was fitted with a bent tube leading into boiled water, as for iron reduction, and boiled. When the sponge of tin was all dissolved, the water was allowed to enter, a fragment of calcite quickly added, and the flask cooled and titrated with iodine solution.

The points to be observed are fine agating of the portion to be weighed, and maintenance of intimate contact in the initial stages. Swaziland and Rooiberg concentrates were readily amenable and the entire operation could be carried out in 50 min. with them, but some black-looking cassiterite from German S.W. Africa took much longer and required more zinc. The process so far is purely academic, because the results obtained from low grade ores to which we thought it would be specially applicable have not been too good, and moreover considerable experience is required in the detail work. Arising out of this matter, however, is another which is of interest and which I do not think has been hitherto dealt with. Some experiments were being tried to find out if electrolysis would assist solution, and in the course of investigation, using a platinum anode and cassiterite cathode and very dilute hydrochloric acid as electrolyte tin was deposited on the cathode. This led to the assumption that cassiterite is a conductor and the further experiment was tried of partial immersion, keeping the terminal out of contact with the electrolyte. Afterwards various crystals of the mineral were placed dry in circuit of an ordinary dry cell battery and voltmeter. The battery voltage was 2·4 volts, and the highest voltage recorded, reading with the crystals in series, was 2·3 volts. Some further experiments were tried on such minerals as were obtainable in definite forms, with the following results in sequence of conductivity:—

Bornite, covellite, chalcocite, calaverite, galena, cassiterite and magnetite.

Chalcopyrite, tetrahedrite and pyrolusite.

Stibnite, hematite, cinnabar, siderite, limonite, monazite, scheelite, hübnerite, mangano-tantalite, columbite, lead vanadate, chromate and molybdate. Sulphates, carbonates and all oxides except those mentioned above, gave no indications with our somewhat rough instrument of measurement.

These experiments are so interesting that we believe they should be followed up in a laboratory equipped with the necessary appliances for the measurement of electrical resistances. It is possible that a research carried out with the appliances which are lacking in a commercial laboratory may lead to interesting discoveries showing, say, bias along certain lines of cleavage in crystals. Some such results we have obtained, but there are pitfalls as the crystals shown will illustrate, and so far our research has brought us to no definite conclusion as to why crystals which possess conductivity will have it more pronounced on one plane than another.

In concluding this little note, I wish to add my thanks to our assistant Mr. F. S. Green for his whole-hearted and enthusiastic collaboration in a long series of experiments partially outlined in this note.

Mr. H. A. White *(Vice-President)*: In connection with the very interesting note which Mr. Whitby has read to us, I would like to ask him if he has made any determination of the depth to which reduction of the tin proceeds when cassiterite is submitted to the action of zinc and acid. I submit that would have an important bearing on the process described.

Mr. A. Whitby *(Member of Council)*: As a matter of fact it proceeds through the whole mass of some cassiterites, that is to say the reduction is complete in possibly say 15 to 20 min., the whole of the cassiterite is converted into spongy metallic tin. Some cassiterites are more obstinate, and I think that fact has been remarked on before—I have not been able to get hold of the reference, but I think that has been found. Zinc fume*, however, was not then used, and it is a medium that readily gets at all sides of the crystal as a particle, and enables decomposition to be effected.

Mr. H. A. White *(Vice-President)*: What I mean is, has Mr. Whitby determined the depth of the metallic tin produced in a large crystal of cassiterite?

Mr. A. Whitby *(Member of Council)*: Would Mr. White kindly inform me how I am going to determine that depth? As a matter of fact, it is only superficial as far as I have been able to observe.

Mr. H. A. White *(Vice-President)*: Quite simply. All one has to do is to dissolve that tin, and then one can calculate from the area exposed and the tin determined, what the depth was.

Mr. A. Whitby *(Member of Council)*: Then we will proceed on those lines.

VACUUM FILTRATION AT THE VICTORIA MILL OF THE WAIHI GOLD MINING CO., LTD., NEW ZEALAND.

(Read at May Meeting, 1913).

By William Macdonald, A.R.S.M.
(Associate).

DISCUSSION.

Mr. Thos. B. Stevens *(Corresponding Member of Council)*: Mr. Macdonald is to be congratulated and thanked for having given us such a detailed account of vacuum filter practice at Waihi. While many descriptions have been written of vacuum filter plants in use in different districts, they have all hitherto carefully avoided giving those details which are of most importance to the practical metallurgist, namely:—the dissolved gold loss, and the detailed cost of treatment; in this respect the paper now before us fills a long vacant gap, and we can only hope that similar information from other sources will now be forthcoming. It is to be regretted that in the past the ever present question of pending patent litigation, and the keen advertising methods of exploiters of vacuum filter processes has restrained that free discussion of the subject of vacuum filtration which has now become so universal in other branches of our art. After carefully reading Mr. Macdonald's paper, one can only feel convinced that the Metallurgical staff of the Waihi mine have succeeded in evolving a type of plant eminently suited to their slime, and the author very rightly points out in his opening remarks that such a plant might possibly be an entire failure on other classes of material. In Western Australia we have in the same way produced a type of plant to suit the requirements of the slime which we are treating. In consequence of this very necessary modification of plant to suit the nature of material treated, vacuum filters in America, New Zealand, and Western Australia, although essentially the same in principle, differ very much in detail; doubtless operators on the Rand also have already introduced further modifications to suit their own special requirements. For slime treatment here we have the two main types of vacuum filters at work: the fixed frame type with separate vat into which the residues are discharged and mixed; and the travelling frame type known as the "Improved Ridgway Machine." Both types have most enthusiastic supporters and the honours as regards the erection of new plants are practically equally divided, so that I am afraid we cannot give our Rand friends any final advice as regards the relative merits of fixed and travelling frames. The filter frames or leaves in use at Waihi are different as regards the filling material between the cloths from those in use here, and, I believe, elsewhere also. On several occasions I have tried the use of corrugated steel as compared with the usual coco mat or wooden laths, and I have always had difficulty in obtaining a clean discharge of the cake, so that hand cleaning has been necessary. The steel certainly possesses the advantage of allowing the cloths to be readily removed for cleaning, but I doubt if in a country where unskilled labour is expensive this feature could outweigh the advantage of the perfectly clean discharge which can be obtained by air or water pressure alone when using cocoa mat. The use of the "Vacuum Thickening Plant" is interesting as showing a method by which the amount of water entering the cyanide plant can

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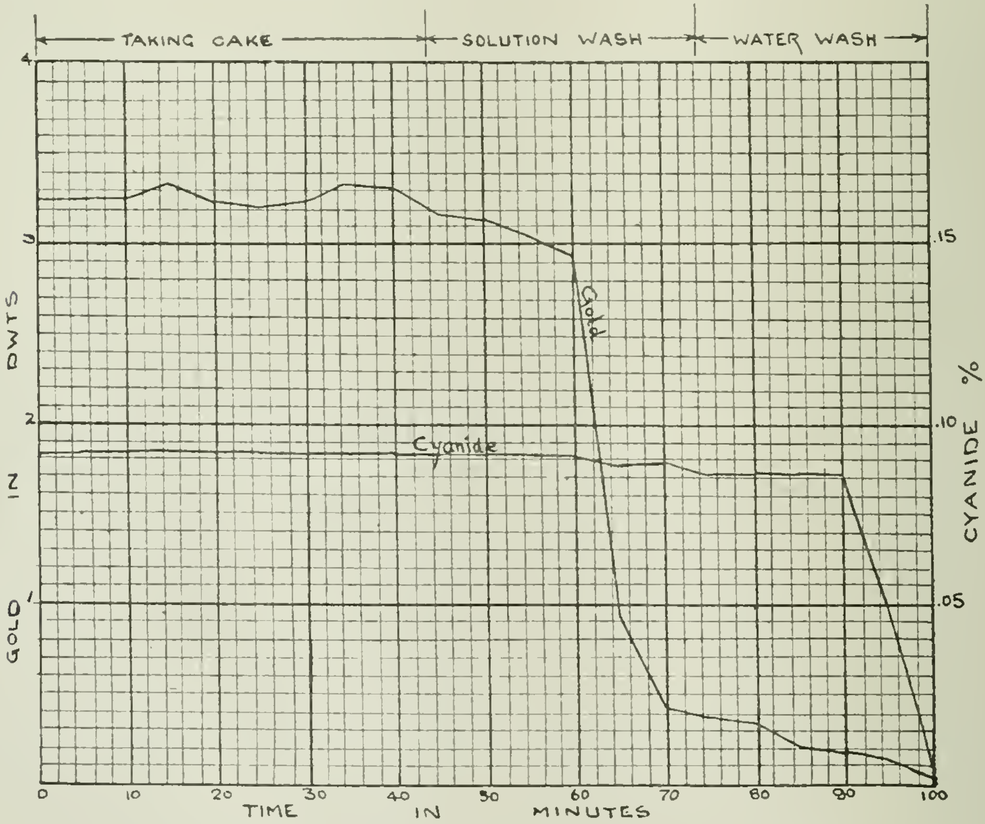
*I have since found that this has been used before in a gravimetric process described by Koomans. See this *Journal*, Vol. ix, July, 1908, page 30.

be reduced, but it will be noted that the dewatered pulp contains but little less moisture (35 to 40%) than is now obtained by decantation plants on the Rand. It would be interesting to know what is the percentage of moisture in settled Waihi pulp, as it would then be possible to form an opinion of the saving of gold and cyanide which is due to the use of the thickening plant. Considering the low values in gold and cyanide of the spent solution used as wash in the "Washing Plant," it appears doubtful if anything is saved by the thickening and it might be more profitable to thicken by settlement only and run more spent solution to waste. The author does not give any explanation as to why the filter cakes from the "Thickening Plant" contain 35% to 40% of water whilst those from the washing plant contained but 30% of spent solution. On account of this difference it must still be necessary to run to waste the excess of spent solution formed in addition to that which is discharged with the residue. In consequence of this the gold value of this additional spent solution should be added to the dissolved gold contained in the residue in order to obtain the total residue loss. It is fortunate that the Waihi ore apparently, like that of the Rand, yields its gold to weak strength of cyanide solution and that a spent solution low in gold can be obtained. It is these conditions that render the use of a final water wash in the vacuum plant unnecessary, and it is therefore not advantageous to crush in cyanide solution as it is in Kalgoorlie and in other districts. When a slime requires as high as 0·07% KCN solution to dissolve the gold, and solutions cannot be precipitated below 0·12 dwt. without excessive cost, the spent solution becomes so valuable that the final fresh water wash, with contingent crushing in cyanide solution, becomes the most profitable



Head : Total Gold 5·5 dwt. Gold Solutions 3·25 dwt. KCN 0·083%. Spent Solution Wash 0·12 dwt. 0·072% KCN. Residue 0·003 dwt. dissolved Gold per ton Dry Slime.

treatment. On such a plant residue must be discharged moist with practically only fresh water, and to accomplish this all of the plant make-up water must be added as wash in the vacuum plant. The saving in gold and cyanide to be obtained by such a method of washing is well shown by the curves given in Figure 1, which were plotted when treating a 5·5 dwt. slime, which is about the same value as that of Waihi. The soluble gold loss also in residue was much the same as is obtained there, being slightly under 1 grain per ton of dry slime. The curves show the strength of gold and cyanide of the solution drawn off by the vacuum pumps during formation of cake, solution washing and water washing. The method of determining the dissolved gold in the residue by squeezing out the gold-bearing moisture from a sample of the cake is open to the objection that the gold-bearing solution present in the filter frame and spacing material is not included in the sample. Possibly this is of little account when the spacing material is corrugated steel. My own experience has been that when using coco mat sufficient solution is discharged from it directly after the cake to appreciably increase the value of the residue in dissolved gold. I should be glad to know if this has been noticed at Waihi. One of the great advantages of discharging the cakes from a vacuum filter into a separate mixer for disintegrating before pumping to the residue dam, as in our Western Australian filters, is that a very accurate sample of the residue can be obtained which can be guaranteed to contain all of the dissolved gold. The primary reasons, however. for which these mixers were installed was to save time and water in discharging. The worker on other fields frequently hears surprise expressed at the great vitality of the decantation process on the Rand. The chief reasons for such vitality I would suggest are :—(1) Ideal grade of ore for the process as mentioned in Mr. A. Salkinson's remarks ; (2) the possibility of obtaining excellent extractions and precipitation with very weak solutions, so as to make total recovery of the cyanide solutions unnecessary ; (3) very little mechanical or air agitation being necessary to dissolve the gold, consequently a very thin pulp and weak cyanide solution can be used ; (4) an excellent water supply ; (5) the great difference in price between skilled and unskilled labour ; (6) a slime, which as compared with many others settles remarkably well and consistently.

In Kalgoorlie the filter press and vacuum filter bear much the same relation to each other as the decantation process and the vacuum filter do in South Africa. Casual consideration would lead one to conclude that owing to the large saving in labour in a place where unskilled labour is so expensive, the vacuum filter must yield very much higher profits. More mature consideration however will show that with water costing 7s. per 1,000 gal. the filter press shows a saving of moisture in residue amounting to over 4d. per ton as compared with the vacuum, and when the great ease with which a final fresh water wash can be applied to a filter press is also taken into account the reason why so many filter press plants have been retained is apparent.

NOTES ON THE ASSAY OF MINE SAMPLES.

(*Read at June Meeting, 1913.*)

By Robert Dures (Member).

DISCUSSION.

Mr. A. Whitby *(Member of Council)*: At first sight there does not appear to be much to discuss in this paper, but there is interest in the author's opening remarks. Much of the mine sampling done in the past has been more or less valueless, because some of the men appointed to this work did not know hanging from foot, because they were compelled by their superiors to take more samples per day than they ought, because the crushing and preparing of samples was mainly done by Kafirs, and because, as the author suggests, there has been much juggling with figures. I hold, and have always done so, that the mine sampler should be a past-graduate in his particular province, that he should not be allowed to take more than 30 samples a day, and that more care and attention is required in the taking, crushing and sampling down of what he brings up. It would tend to closer correlation between milling and mining figures. One of my managers of long ago remarked to me on the close relationship existing between the mine values and those returnable from the surface. Well, the sampler on this mine hardly ever exceeded 30 per day, the samples were all hand crushed under my supervision, and half a penny weight was the average difference in computed values. Since then that particular mine has at times rushed up to 150 samples a day, but I consider that the quality of the work must have suffered. Why employ an assayer and assistants to assay samples which from the methods of sampling are not representative of stope or drive?

In the author's next paragraph he deals with a vexed question. If a sample of mixed ore requires to be turned over 100 times, who is going to do it? On most of our large mines the chief might well be glued to his balance-stool for all the opportunity given him to supervise the work. Supposing that he deputes to a subordinate the duty of weighing, he should check that work, see every cupel, be sure of his partings, and generally keep an eye on fluxing, potting, not to speak of the preparation of his samples. Bear in mind that he is held solely responsible, and is often merely supplied with learners.

The author's tables do not present anything very new, but I cannot quite agree with his quantities. 90 gm. of soda seems somewhat excessive and I do not hold with his borax cover. According to the principles laid down by recent authorities, borax, even when only required in small quantity, best carries out its function when intimately mixed with the charge. I must refer Mr. Dures to Fulton's Manual of Assaying and previous remarks of my own on this subject. His remarks on varying amounts of silver on the same mine I endorse, but how is it to be obviated? Most mine samples are handed in with a bare number, and it is a fairly safe average to take silver fineness of mill bullion as a guide even supposing that the samples are truly representative to half a pennyweight or so.

Mr. Jas. Gray, F.I.C. *(Member of Council)*: I fully anticipated that a brisk and valuable discussion on assaying would have resulted from Mr. Dures' paper, for many points have been raised which afford an opportunity for criticism, and opportunity was also given for many little time-saving devices of much value to the busy mine assayer, but at present unknown beyond a limited circle, to be brought forward. For instance, it was while sojourning in Southern Rhodesia that I ran across a quick method of inquarting with silver by means of a charcoal block and a blowpipe, thus saving a cupellation.

It was my lot some years ago to occupy the position of assistant assayer on one of the local mines and in those days mine samples received very scurvy treatment. At one particular mine it was the custom for over 100 mine samples to receive a 'baptism of fire'—for it was nothing more—in the early hours of the morning. One A.T. charges were fluxed and placed in "F" pots and were then handed to the nigger for fusing. Each fire held about 20 pots and the cupellations were carried out in No. 5 cupels in batches of 32 and I have no doubt that the results were anything from 50% to 100% too low. I hope that these days are gone.

I had hoped that the discussion would have brought before us some comparisons between the old Cornish furnace, now rarely seen on the mines, and the reverberatory furnace. I am told that the amount of reducer necessary with the reverberatory is considerably greater, and I have heard of one unfortunate assistant who on his first experience of this type of furnace kept to the old quantity and nearly lost his job, as well as having to suffer a lot of lurid language until the chief had undergone the same experience.

No assayer will surely dispute the necessity of thoroughly cleaning the bullion from adhering particles of cupel and this is emphasised in the table given by the author.

Many assayers, experienced on the Rand, on proceeding to other districts neglect to determine the silver and follow the Rand practice of deducting 10% until Nemesis overtakes them. As the author points out, it is necessary, even on the Rand, to keep a careful check on the silver value which alters from time to time, and from place to place.

ELECTRIC BLASTING.

By Wm. Cullen, (Past-President), Thos. Donaldson (Member of Council), and W. Waters (Associate).

During the past few years the question of health conditions in the Gold Mines of the Rand has been receiving a large amount of attention and it is pleasing to observe that earnest endeavours are being made by those intimately concerned to bring about improvements in this direction. Considerable progress has been made with schemes of ventilation; the use of water sprays is becoming more general and various other means have been discussed and tried from time to time. It must be admitted, however, that none of these methods, nor even a combination of them, have proved effective in overcoming the dust evil, during, and immediately after, blasting. We think it is being gradually realised that the best way out of this difficulty is to bring all workers to the surface before blasting actually begins, and within the past few months serious attention has been given to the question as to whether or not electric blasting would prove of material assistance in this direction. It is a well known fact that electric blasting has, for many years, been extensively used in Europe and America, but we prefer to confine our attention to the position in South Africa. In Natal this system has been compulsory in "fiery" coal mines for many years, while in Rhodesia several gold mine shafts have been sunk by its aid. In the Transvaal and Orange Free State electric blasting is employed in several coal mines. The Transvaal and Delagoa Bay Colliery has used it exclusively for over a decade; the Middelburg Steam, Coal and Coke Company has practically done the same, while the Vereeniging Estates have also used it to a considerable extent. At the Premier Diamond Mine, when working yellow ground, electric blasting was largely used, and found to be more economical than safety fuse. When, however, blue ground had to be dealt with, the action of a row of simultaneous shots brought down huge masses of rock, which required to be reblasted and broken up for loading. Safety fuse blasting was, therefore, reverted to, as the rock was broken smaller by shots acting independently of each other. Electric delay action fuses could be used with success at the Premier, but as all the workings are in the open, the question of health conditions is not of primary importance. In view of the fact that in practically all of its branches Rand Mining Practice is probably the most up to date in the world, it appears somewhat extraordinary that until quite recently no organised attempt has been made to test the possibilities of electric blasting. In order to show that we ourselves have always considered that it was worthy of serious experiment in connection with gold mining operations, we will give a short resumè of what we have done in the past. The first attempt, so far as we know, was made at the Simmer & Jack East Mine in May, 1904. These experiments were conducted by Mr. Catlin, of the Consolidated Gold Fields, in conjunction with one of us, but, owing to reasons which we need not detail, the matter was temporarily dropped. In 1907 we again took it up and trials were conducted at seven of the most important gold mines. Our experiments, which in some instances were fairly extensive, were carried out intermittently, according to circumstances, and ranged over a period dating from May, 1907, to August, 1911. These proved conclusively that so far as actual rock breaking is concerned, electric blasting gave results at least equal to those obtained with ordinary safety fuse. At that time, however, the question of cost appeared to be the first consideration and, seeing that safety fuse blasting was apparently the cheaper, no material interest in electric blasting was aroused amongst Mine Managers. It was consequently left to the contractor to say whether or not he would adopt it. Now, as we all know, the miner throughout the world is a very conservative individual, and as he has to work underground under unpleasant conditions, he naturally asked whether electric blasting would give him a bigger cheque. If not he had no use for it. We suggested that one or more levels might be equipped with permanent cables and stationary exploders, in order to make the work as easy as possible for the miners, and enable them to carry out thorough tests, but such facilities were not forthcoming. We would emphasise the fact that we do not intend, or even for a moment wish, to criticise the attitude adopted by anyone concerned, as we must frankly admit that the important advantages to be gained were not then appreciated. In other words, that the benefits are not so much in the way of obtaining better blasting results, as in effecting improved health conditions for all underground workers. We shall, however, endeavour to show that not only are health conditions improved, but that under certain circumstances even better blasting results can often be obtained.

We have for several years contemplated writing a paper on this subject, but the apparent lack of interest taken led us to the conclusion that it would be premature. It is, therefore, all the more pleasing to find that at last a real and earnest interest in electric blasting has been aroused in the Transvaal, mainly because everyone is anxious to do his utmost to minimise and, if possible, eradicate, that terrible scourge miners' phthisis. The first practical step was taken by the City Deep Gold Mine, and sinking operations with electric blasting have been carried on there for several months. We must, however, give credit to the Albu Group for being the first to equip a whole mine for electric blasting. This has been done at the Meyer and Charlton G.M. Co., Ltd.

While on this subject, we should like to be allowed to refer to the whole-hearted personal interest which Mr Geo. Nathan, the Managing Director of the General Mining and Finance Corporation, and Mr. F. W. Girdler-Brown, General Manager of the Meyer and Charlton Gold Mine, have taken in the matter.

Advantages Claimed for Electric Blasting.—We will now consider in detail the probable effects of electric blasting and the advantages claimed for same. It is quite certain that the results will be far-reaching, and we will endeavour to deal with these under the three following headings:—

1. Health Conditions.
2. Safety.
3. Economy.

Health Conditions.—Undoubtedly this is the most important. At the present time it is not necessary to propound theories. It is an accepted fact that dust is "*the*" great danger to health, and that rock dust causes infinitely greater damage to human beings than is caused by noxious gases.

Reference to the interim report to the Minister of Mines submitted in June last by the Miners' Phthisis Prevention Committee will show that the maximum quantity of dust is found in the atmosphere of the mine *immediately* after blasting, and that the amount remains at a high figure for several hours. It is also interesting to note from the same report that when dust is present in very large quantities, respirators do not afford that protection which is commonly attributed to them. Ventilation which must in course of time be brought to a high state of perfection will undoubtedly reduce the influence of noxious gases to quite negligible proportions, except in places which are naturally difficult to ventilate. Mr. Penlerick in his interesting communication before this Society in August 1912, showed that improved ventilation can also reduce the dust danger, and while it is quite true that steps apart from electric blasting are being taken to minimise and overcome this danger, it must be admitted that it is far better for a miner to be inhaling pure air in the sunlight than to be deep down in the bowels of the earth at blasting time, when the dust is at its worst. It is, of course, well known that explosives cause fumes and dust, but we have good reasons for believing that few miners have any real conception of the amount of fumes and noxious gases which are produced by safety fuse alone. A number of laboratory and underground experiments have been made in this connection and reference to a paper entitled "Gases Resulting from the Use of High Explosives," read by one of us before this Society in November, 1908, will show that appreciably large quantities of both carbon monoxide and carbon dioxide are produced by burning safety fuse. Miners, of course, know from actual experience that safety fuse gives off a very large amount of smoke, which must naturally hold in suspension a large quantity of the very fine dust raised during blasting. Electric blasting will remove the necessity for white miners and natives having to rush around to light up at firing ("cheesa") time, and then travel through dust and fume laden drives, stopes, etc., as, except under special circumstances, no charges should require to be exploded before all workers are out of the mine. Instead of the rush always in evidence with safety fuse blasting, the electric system will enable the operators to explode the charges at their own convenience, and from a place far removed from danger zones. One might write very fully on this subject alone, but we consider the advantages claimed in this direction to be so self-evident that we need not dilate further on the matter

Safety.—On this question, as in many others, arguments can be adduced in both directions. At least one of us has had very considerable experience with both methods of blasting in South Africa, and in many other mining fields, and we unhesitatingly express our firm conviction that electric blasting is just as safe as blasting with ordinary safety fuse. Provided ordinary precautions are taken, and common sense exercised, there should be no danger of premature explosion when a portable cable and exploder are used, as the miner should only unwind his cable as he walks *out* from the working face, and would then reach a place of safety before any connection is made between the exploder and electric fuses. If, for any reason, it becomes necessary for him to return to the face before blasting, or in case of total misfire, he must always disconnect his cable from the exploder, and also make certain that no one will interfere with it in his absence. Where permanent installations are employed, these, as is shown elsewhere, can be made so safe and to all intents and purposes, so completely foolproof, that accidents from premature explosions should be practically impossible. We all know the dangers ever present when having to light up safety fuse while standing in close proximity to holes charged with explosives. Mistakes do occur, and many a good man is now in his grave owing to irregularities in the use of safety fuse. Electric blasting removes this danger.

While it is yet too early to express any opinion as to whether or not electric blasting should be made compulsory for general mining operations, we have no hesitation whatsoever in expressing our conviction that it should be made compulsory for shaft and winze sinking. It surely must be better for a miner, when working in a shaft or winze, to be able to explode his charges from a place of safety than to be obliged to light up each individual shot and then either climb up ropes or ladders or be hauled up by bucket or skip, with the ever present risk that some mishap may occur to prevent his getting safely away before any explosions occur. We will conclude this section by referring to the 1912 report of His Majesty's Inspectors of Explosives for Great Britain. On page 32 of that report it will be found that although an immense amount of electric blasting is carried on in Great Britain, only one miner was killed and eight persons injured during that year by accidents with electric blasting. One accident was caused by the shot-firer allowing another man to have charge of the battery while he himself was at the shot-hole, and the remaining accidents were due to the shot-firer firing the shots before persons at the working faces had taken cover. The report states :—

> "*Few, if any, of these casualties would have occurred had the precautions laid down in the 'Explosives in Coal Mines Order' been observed.*"

Economy.—This comes last in importance, and, in our opinion, should also be considered as least. Let us, however, look into the matter and deal with it under its various phases. On first consideration it would appear that blasting with electricity will prove more costly than with safety fuse. If we take the actual cost of materials used up in each case, the figures work out roughly as follows:—

Electric: 1,000 shots. Cost of instantaneous electric detonator and delay action fuses with detonators	£7 12 6
Safety Fuse: 1,000 shots. Cost of detonators, safety fuse and fuse lighters	£5 12 6

This leaves a difference of £2 per 1,000 shots in favour of safety fuse. We must, however, look for economy in other directions. So far as actual rock-breaking results in metalliferous mines are concerned, the only evident advantage to be gained by electric blasting over safety fuse is in connection with "cuts." In a development drive it is customary, on the Rand, to drill either a 3-hole or a 4-hole cut. For these cut-holes identical lengths of safety fuse are used, the ends being tied together so that all can be lighted simultaneously. The general idea is that each fuse will burn through with such regularity that all 3 or all 4, as the case may be, of the charges will explode at the same instant. Usually only one report is heard, thus apparently proving the above idea. We, however, know from practical experience and experiments, that no safety fuse in the world, no matter what price be paid for it, can be made so perfect in regularity of burning speed that identical lengths lighted simultaneously can be guaranteed to burn through in identical periods. Not only is there sometimes a difference of fractions of one second, but frequently a difference of several seconds occurs between each. Even a fraction of a second makes a great difference in the detonation of explosives. What really happens is this: The cut holes are so inclined that they nearly meet, or often actually meet, at the base, and the first fuse to burn through explodes its charge, which in turn explodes the remaining charges by concussion, or what is generally termed "sympathetic detonation." Although there is apparently only one explosion, so far as the human ear can judge, there is actually a slight interval between the explosion of the various charges, and the force of the first is on the wane before the last has fully developed its power.

Now, when cuts are fired with instantaneous electric detonators, all charges attain their maximum force at the same instant and better results are consequently obtained.

To illustrate what we mean, we may cite the example of three or four horses pulling at a load in individual jerks as against the same number pulling with a combined heave. The difference in effect is obvious. We might add that when cut holes are drilled too far apart at the base, it sometimes happens that more than one report is heard when safety fuse is used. When this occurs the miner generally anticipates trouble and usually his fears are realised. When using instantaneous electric detonators, separate explosions are, of course, impossible.

While on this subject of "cuts" in development drives, we may say we are quite aware that some people claim that by using instantaneous electric blasting, it is practicable to fire the cut and round at the same time. Our opinion is that, in certain classes of rock to be met with in outcrop mines on the Witwatersrand, and in such workings as railway tunnels, etc., where there is ample room to rig up rock drills to make holes at any desired angle, this may be practicable. When, however, we have to deal with the class of rock which exists in our deep level mines, we believe that only failure will result from any attempt to blast the cut and round simultaneously. We have no doubt whatever but that the "cut" should be fired first, followed by the "easers," and that the "shoulder" holes should go before the "back" holes, and the "knee" holes before the "lifters." We also know that there are people who, while admitting that it is necessary that all the holes should explode in the rotation above indicated, yet argue that it is possible when using delay action fuses, to light up the "cut" and the "round" at the same time, thus firing the "round" before making certain that the "cut" has come clean. We do not agree with this procedure. It is a well-known fact that when using safety fuse, at least 90% of the "cuts" do not break to the bottom at the first blast, and consequently require to be recharged with a few cartridges to clean them out. Should the "cut" and round be lighted up at the same time, and the "cut" fail to break, then the whole of the "round" will be hung up and consequently all the explosives wasted. Again should the "cut" have come fairly well but not quite to the back, the fact that it has not been cleaned out before firing the "round" may result in the "round" breaking anything from 6 in. to 18 in. short.

Where possible, a miner should always be given two drives so that he will be able on the same day to blast the "cut" of one drive and the "round" of the other. Should, however, it be found necessary to blast both "cut" and "round" of a drive each day, the difficulty can be overcome by blasting the "cut" by means of a portable cable and exploder at a convenient time prior to the general blast, leaving the "round" to be dealt with in due course at the general blast. This will enable the miner to make certain that the "cut" blast is satisfactory before blasting the "round." We are fully aware that in advising this procedure we are departing to a slight extent from the main object for which the use of electric blasting is advocated, but it will not appreciably effect the general result, and we feel that it would be foolish at the present stage of our experience to favour any other method of working.

Returning to the question of economy pure and simple, we anticipate that if the system employed at the Meyer and Charlton be generally adopted, the extra time afforded for actual work at the face will ensure greater footage being broken, and thus compensate for the higher cost of actual materials used. When a miner has to fire his own charges, as is done with safety fuse blasting, he has very often to rush through with his drilling so as to be able to charge up and fire in the time available. It thus happens that where he might have put in 5½ ft. holes, had he more time, he must be content with 5 ft. holes or even less. At the Meyer and Charlton the miners drill their holes and charge them up ready for firing, leaving the ends of the fuse wires unconnected. Their day's work ends there, and they go to the surface. Special shot firers then attend to the connecting up of the wires and the firing operations. The additional footage to be gained in this manner may prove to be considerable.

Again, if white men and natives enjoy good health, they must be able to do better and more efficient work. The economic advantages to be gained in this direction must prove very far reaching.

Lastly, there is the Miners' Phthisis allowance to be considered. If mining companies can save this heavy expenditure, or even part of it, by ensuring good health for their workers, the economics are self-evident.

General Use in Mines.—For cuts and hammer holes, where the hanging is good, instantaneous electric detonators should be exclusively used. For all other work delay action fuses are recommended. Seeing that delay action fuses act in identically the same manner as ordinary safety fuse, in respect of each hole exploding in the desired rotation, no comments thereon are necessary.

Work Done at the Meyer and Charlton.—Before proceeding to deal with the nature and construction of the appliances required for electric blasting, we will give a short resumé of the work done to date at the Meyer and Charlton G. M.

The first step consisted of a surface demonstration given by one of us at the request of the manager. The demonstration was intended to show what materials were ready at hand available for use, rather than to impart any instructions as to the use thereof. This demonstration, given in the morning, was followed up the same afternoon by practical underground experiments, the necessary charging up of the holes being under the supervision of the acting mine captain, Mr. Gillilands. (In passing we should like to mention Mr. Gillilands as having taken very great interest in the whole matter, he having in earlier days been accustomed to this work).

The first underground experiment proved highly successful, and the management lost no time in equipping the whole workings of one level with permanent cables and firing same from a switch box at the shaft station, utilising the electric lighting current. The next step was to equip four entire levels with permanent cables, connected through the shaft to a firing cabin at the surface shaft head. On the 29th September last the whole mine was electrically fired from the surface by Mr. Kotze, Government Mining Engineer, and the results proved more satisfactory than could have been anticipated. Notice of how matters have progressed at the Meyer and Charlton have appeared in the daily press and we must confine ourselves to the following record of a few days' work :—

October 1st.	580 holes.	3 misfires.
,, 2nd.	600 ,,	4 ,,
,, 3rd.	580 ,,	2 ,,
,, 4th.	586 ,,	4 ,,
,, 6th.	550 ,,	3 ,,
,, 7th.	620 ,,	5 ,,
,, 8th.	590 ,,	4 ,,
,, 9th.	565 ,,	2 ,,
,, 10th.	602 ,,	3 ,,
,, 11th.	630 ,,	5 ,,
Total: ...	5,903 ,,	35 ,,

Percent misfires ... 0·592.

For about two-thirds of the shots, delay action fuses were used, the remainder being fired by means of the instantaneous type. We are informed by the manager that to the best of his belief, when ordinary safety fuse was used, that is, prior to the introduction of electric blasting, on an average 6 to 7 misfires occurred daily, the number of holes blasted being about the same as given in the foregoing list, so that the average percentage of misfires with safety fuse was slightly over 1·0%. These figures speak for themselves.

The remaining part of our paper deals with the nature, construction, and use of the various appliances required for electric blasting.

Electric Detonator Fuses.—These are of two classes, viz. :—

1. Instantaneous electric detonators.
2. Delay action fuses.

1. *Instantaneous.*—There are two varieties of this class, viz. :—

(*a*) High tension.
(*b*) Low tension.

The detonating composition in both varieties is the same as in the ordinary detonator.

(*a*) *High Tension.*—In the high tension variety two thin strips of metal, insulated from each other, and to which the two wires are respectively soldered, enter the tube where they are embedded in sulphur, which acts as a waterproofing composition. The ends of these leads are not metallically connected together, but are surrounded by a flashing mixture, somewhat similar to a match head, but naturally much more powerful. The ignition of the flashing pill is effected by the passage of the electric spark between the two terminal points, and the flashing pill in turn explodes the detonator. High tension electric detonators are used in Great Britain to a certain extent, but are not exported as they are considered unsuitable for transport and long storage. These have also the defect that they cannot be tested before use with a galvanometer to ascertain their condition, nor after being connected up, to find out if the circuit is complete. On account of the high voltage required for the high tension type, these have to be connected up in parallel and are consequently unsuitable for conditions where a large number of shots have to be arranged in series.

(*b*) *Low Tension.*—These are similar in general construction to the high tension type, with this important difference, that the ends of the metallic leads embedded in the flashing pill are connected together by means of a very thin platinum wire, which is technically known as the "bridge."

Illustration No. 1 shows how low tension detonators are constructed. Ignition of the flashing pill is caused by the electric current passing through the bridge and heating same. Owing to the circuit being completed by means of the platinum bridge, this type can be tested with a galvanometer both before use and after being connected up. It has also the advantage that a large number can be connected up in series. Experience has proved that low tension detonators are very suitable for transport and long storage.

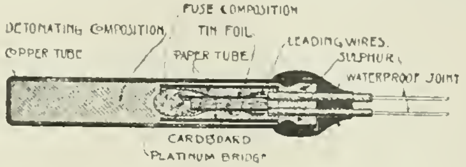


Illustration No. 1 (a).—Low Tension Instantaneous Electric Detonator (In Section).

Illustration No. 1 (b).—Low Tension Instantaneous Electric Detonator (Complete).

Delay Action Fuses.—When blasting with instantaneous or delay action fuses connected with a given source of current, ignition takes place simultaneously, and the charges connected with the instantaneous fuses naturally explode simultaneously. Simultaneous explosion is advantageous under certain circumstances, but there are conditions under which it is neither desirable nor permissable. It is to meet such conditions that delay action fuses have been introduced, for while they also permit of simultaneous ignition, their use makes it possible to explode the charges to which they are connected in any desired rotation. There are several varieties of delay action fuses, but in our opinion the only type which is suited to conditions in South Africa is that known as the "Venus." We will consequently confine our attention to

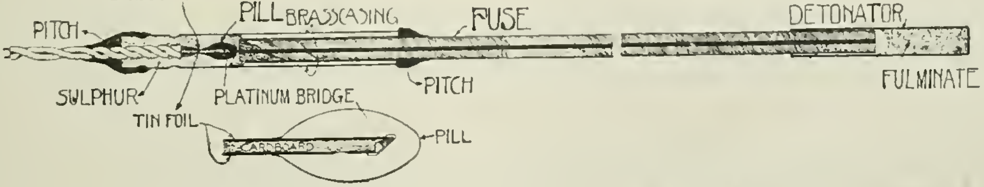


Illustration No. 2 (a)—Delay Action Fuse (In Section).

this particular type. The construction of the electrical portion of the Venus delay action fuse is the same as the low tension instantaneous type, in fact, the only essential difference is that a piece of ordinary safety fuse, varying in length, according to the delay required, is interposed between the flashing pill and the detonator. The electrical portion is situated at one end of a thin brass tube $2\frac{1}{2}$ in. long and is embedded in sulphur as in the instantaneous variety, while the safety fuse enters from the other end and is secured in the tube by squeezing the latter round it at one point. Further security is given by closing up the junction of the tube and the safety fuse by means of water-proofing composition.

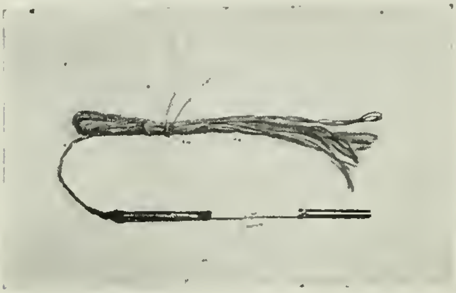


Illustration No. 2 (b).—Delay Action Fuse (Complete).

The number of delays which can be obtained from fuses of this class is practically unlimited, but for ordinary practice 8 to 10 delays are generally found sufficient, and it has also been found that in order to ensure that the charges explode in proper rotation, a difference in length of, at least, 2 in. between each fuse is required. We are able to state as a result of considerable experience with instantaneous electric detonators and delay action fuses, that both are reliable and it is only rarely that we come across a defective one. Nevertheless we recommend that every fuse should be tested before use. From this point onwards, for the sake of convenience, we will use the term "fuse" to indicate both electric detonators and delay action fuses.

Testing Fuses.—The test we are about to describe provides a means of detecting whether there is any flaw or break in the platinum bridge or the wire leads, and the importance of detecting any defect of this kind will be realised when we state that one defective fuse in the circuit is sufficient to prevent ignition of several or even all of the fuses in that circuit. For the test in question the galvanometer is used.

Illustration No. 3 shows how this test is carried out. When instantaneous fuses are being tested singly, the detonator is inserted through the hole in the inverted metal pot, and the leads are brought to the terminal of the galvanometer. One lead is placed in contact with one of the terminals, while, with the other lead, contact is made and broken intermittently with the other terminal. If the needle does not move freely the fuse is defective and should be discarded. The current produced by the type of galvanometer specially constructed for this work is necessarily very small as otherwise it might explode the detonator, but to prevent accidents the detonator, as already said, is placed in the inverted pot. Delay action fuses are tested in the same way, but when there is no detonator attached it is unnecessary to put the fuse in the pot when testing.

Illustration No. 3.—Testing Single Fuses by Means of Galvanometer.

Resistance of, and Method of, Connecting up Electric Fuses.—It goes without saying that electric fuses offer a certain resistance to the passage of the current and the amount of that resistance is of great importance, for it regulates the number of fuses which can be fired in a single series by current of a given voltage. In order to make this matter clear we will, at this point, briefly refer to the methods by which the fuses are connected up.

Method of Connecting Fuses. Broadly speaking there are two methods:—

1. In Series.
2. In Parallel.

Illustration No. 4 shows how connecting in series is done, and renders further description unnecessary. Connecting in parallel merely

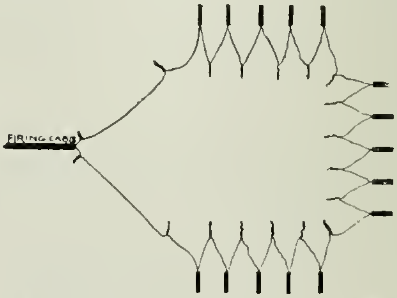


Illustration No 4 (a).—Connecting in Series (Diagram).

means that instead of arranging the fuses in one series, they may be divided into two or more series, which can all be connected with the same current supply.

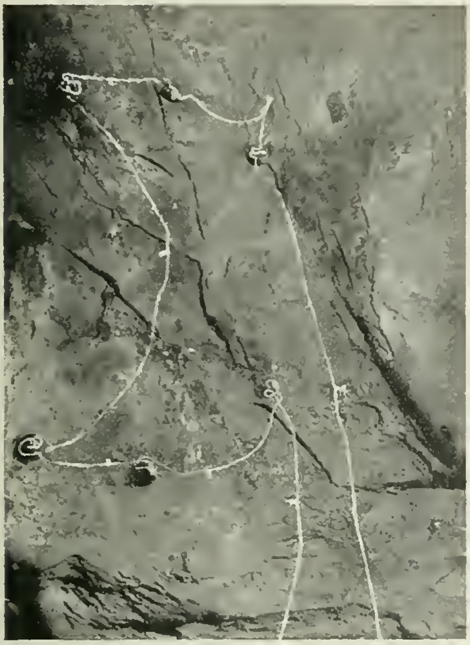


Illustration No. 4 (b).—Connecting in Series; Set of Charges Ready for Firing.

As will be shown later, on account of the resistance being distributed, this method has the advantage that the number of fuses which can be connected to, and fired by, the same supply of current is greatly increased. This brings us back again to the question of resistance. The resistance of a low tension electric fuse is due to the platinum bridge and the two wire leads. The bridge consisting as it does of very thin platinum wire offers a resistance of 1 ohm, the amount of the remaining resistance depending on the nature of the metal used for the leads and the thickness and length of same. For example, if these leads are of copper wire of the usual thickness (namely, about ·022 inches) and each 6 feet long, the total resistance of the fuse will be about 1·25 ohms, the copper wires being responsible for only 0·25 of the total. If, on the other hand, iron wires of the same length and thickness are employed, the total resistance rises to almost 3 ohms.

We have already indicated that given a current of a certain voltage, the number of fuses which can be fired in series depends on the total resistance of the series. In other words, it depends on Ohm's Law, which is

$$\frac{\text{Pressure (in volts).}}{\text{Resistance (in ohms).}} = \text{current (in amperes).}$$

Now the total resistance offered by a number of fuses connected in series is equal to the sum of the resistance of the whole of the fuses in that series. For example, if we connect up 120 fuses, each fuse having a resistance of 1·25 ohms, the total resistance will be 150 ohms, and if the power at our disposal has a pressure of 110 volts, the current in amperes which can pass through that series of fuses will be

$$\frac{V=110}{R=150}=0{\cdot}73 \text{ amperes.}$$

We know from experiment that when the voltage and resistance in a given circuit are so related to each other that 0·7 amperes can pass through the circuit, all of the fuses will ignite, so that a pressure of 110 volts will fire 120 fuses if their combined resistance is not more than 150 ohms. If, however, we connected up, say, 180 fuses, each one having a resistance of 1·25 ohms, the position would be as follows:

$$\frac{V=110}{R=225}=0{\cdot}4 \text{ amperes,}$$

and we know, again from experiment, that many, if not all of the fuses in such a circuit, would misfire. We cite this last example in order to show how important it is that the number of fuses in one series should not exceed certain limits. We have said that so long as a current of 0·7 amperes can pass through a series of fuses, the necessary heating effect to ignite all the fuses will be produced, but, for the sake of safety, we strongly recommend that the resistance, *i.e.*, the number of fuses in a series, should be so limited that a current of, at least, 1 ampere can pass through same; that is to say, the value of the resistance of the circuit, in ohms, should not exceed the value of the pressure expressed in volts.

Example: 80 fuses each with a resistance of 1·25 ohms = 100 ohms total resistance.

Then: $\frac{V=110}{R=100}=1{\cdot}10$ amperes.

It will, we take it, be understood that if fuses with iron instead of copper wires of the same gauge are used, the number which can be fired in one circuit must be greatly reduced; for instance: 35 iron wire fuses, each having a resistance of 3 ohms, if connected in series, will offer a total resistance of 105 ohms. As we have already said,

$$\frac{V=110}{R=105}=1{\cdot}05 \text{ amperes,}$$

so that if we follow the standard already set up, 35 of such fuses is the maximum number which should be connected in one series with a supply voltage of 110.

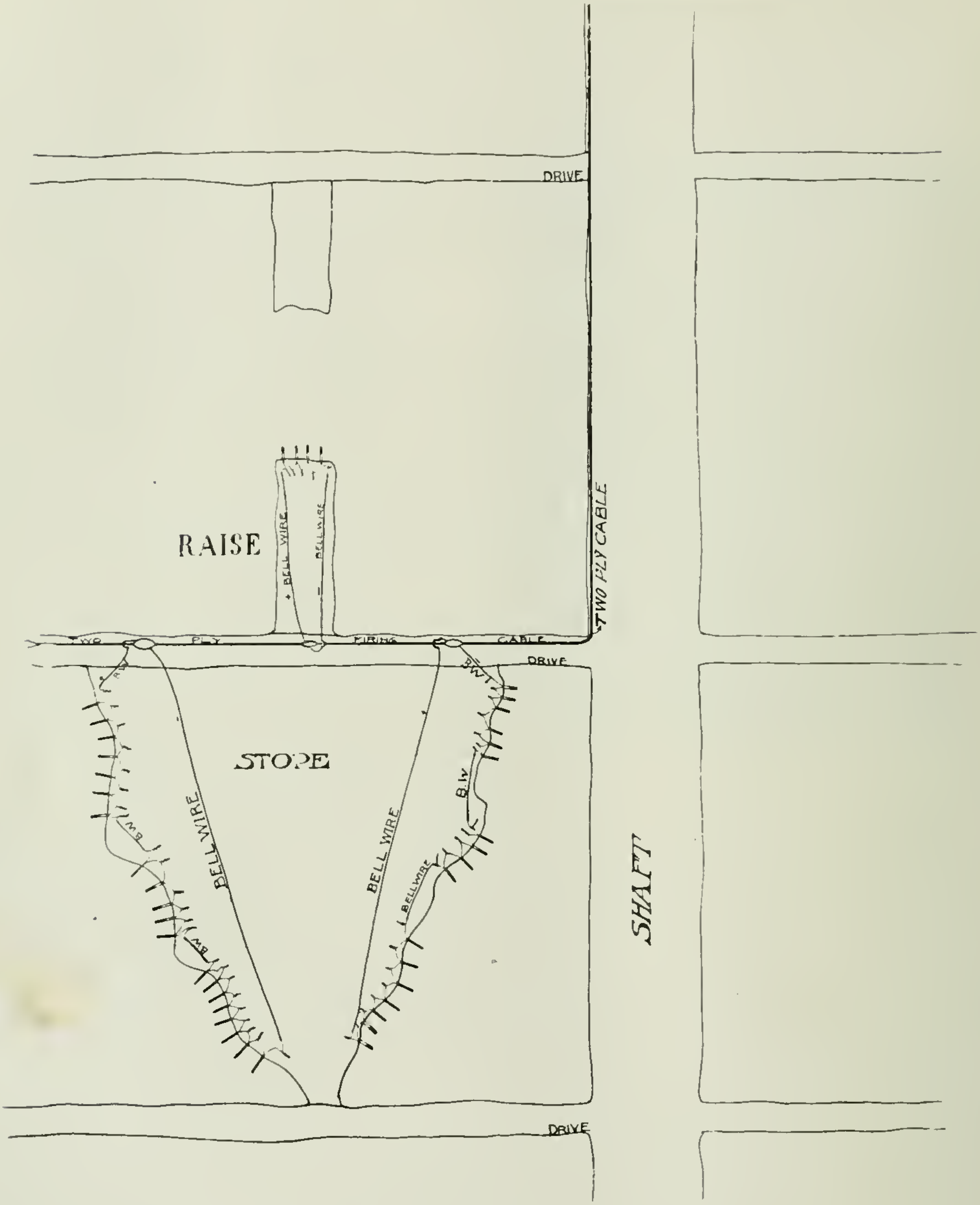


Illustration No. 5.—Connecting up in Parallel.

Now, the question arises, if with a pressure of 110 volts, the number of copper wire fuses in a single series under the most favourable conditions must be limited to about 80, how can it be arranged to fire a number exceeding that figure, from a similar voltage supply? The answer is that they must be arranged in two or more parallel circuits. This leads us to the further question of how many parallel circuits can be connected to one voltage supply. Assuming again the pressure to be 110 volts and the number of fuses in each series to be within the limits already mentioned, the number of parallel circuits still depends on the amount of power available. For instance, if the power is supplied by a transformer at the surface with an output of 10 amperes and about 110 volts, to a particular level through wires of such thickness that when tested at any one point in the level, it shows substantially 110 volts, when a current of 10 amperes is flowing, then, assuming as before that 1 ampere is required for each series, ten series can be connected to and fired through these wires. We know from theory, as well as from practice, that this can be done, but we strongly recommend that even a larger safety margin than the above conditions would afford should be adopted.

At the Meyer and Charlton, where a greater reserve of power than we have mentioned is available, the number of fuses in any one series is so limited that the resistance does not exceed 75 ohms. With a large safety margin like this, the number of circuits in parallel may be increased with little danger of overtaxing the capacity of the system. It also reduces the chances of misfires in the event of any excessive resistance being inadvertently introduced through defective connections, and covers any voltage drop in the feeding wires.

To sum up this important part of our paper, we may say that the number of fuses in series is governed by the value of the voltage, and the number of circuits which may be connected in parallel is limited by the magnitude of the power available.

Relative Merits of Copper and Iron Wire Leads.—It will be gathered from what has just been said that on account of the relatively low resistance which copper offers to the passage of electricity, from the mining point of view, it is preferable that the wire leads should be made of that metal, rather than iron, for it not only simplifies connecting up, but reduces the risk of misfires.

From the metallurgical point of view, however, iron has the advantage that there is no danger of its interfering with the process of treatment of the ore, nor does it affect the quality of the fine gold. It is well-known to metallurgists that at present a certain amount of copper from the detonator tubes finds its way into the bullion, although not to any serious extent, and, so far as we know, copper from this source has never been held responsible for any troubles connected with the metallurgical process. We are of the opinion that the greater part of the wire leads of electric fuses could be picked out of the ore before it reaches the stamps, but it is undoubted, that if copper wire leads are employed the amount of copper which gets through the battery screens will be considerably increased. Whether this increase will seriously affect the process of treatment of the ore or the quality of the final product, is at present, uncertain, but if ultimately it should be found detrimental to an appreciable degree, iron wire leads will require to be used exclusively, or at least, to a considerable extent.

Making up Charges: Tamping and connecting same.—The manner of connecting an electric fuse to the cartridge is practically the same as for ordinary fuse and detonator and requires no description here. With regard to tamping, it may be said at once that electric fuses are rather more liable to damage by the tamping rod than the ordinary fuse, and miners should be cautioned to exercise care when carrying out this operation.

Method of making Contact between Fuses.—Illustration No. 6 shows how the bare ends of the wires should be twisted together in order to ensure good contact and firm attachment.

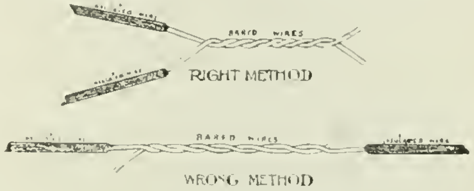


Illustration No. 6.—Method of Connecting Fuses Together.

It is always advisable in order to avoid loss of current, that care should be taken to prevent uninsulated joints coming in contact with the ground. This is particularly advisable in the gold mines in the Transvaal where the water is often slightly acid, for, as is well-known, slightly acid water is a much better conductor of electricity than neutral water. If this should be impossible, as for example in wet shafts, it is absolutely necessary that the joints should be protected by insulated tape.

Portable Exploders.—We come now to the apparatus required for firing electric fuses. The most common method is by means of a portable exploder, and where work is not on a very large scale, this apparatus is quite suitable and safe. There are several types of exploders, but, generally speaking, all act on the same principle, which is that of the dynamo, in fact an exploder may be looked upon as merely a minature dynamo.

In one type of portable exploder the current is generated by rotating the armature by means of a small handle at the side, and when the maximum speed has been obtained, a button is pressed which closes the circuit, connecting the current supply with the firing line.

Another and probably the most common type is known as the "Rackbar" exploder. With this type the armature is rotated by means of a vertical plunger acting on a set of cogwheels.

When the plunger reaches the bottom of its stroke, it automatically closes the circuit, and the full pressure of the current is discharged into the firing line. The voltage produced depends greatly on the force with which the plunger is pushed down and to ensure maximum voltage it is important that the plunger should be pushed down with as much force as possible.

Exploders can be obtained of different sizes according to the voltage required, from the small pocket type, capable of firing only one or two fuses, to the very powerful "Atlas" type, which we have just illustrated, and which is capable of firing up to 170 low tension fuses fitted with copper wire leads and connected in series.

Illustration No. 7.—Atlas Exploder.

Firing Cables.—It is obvious that for safety reasons the exploder must be placed at a considerable distance from the charges, and the current is conveyed through two copper leads. Where no permanent system of wiring has been installed, a portable "twin wire" cable is generally used. The leads of a cable of this type are carefully insulated from each other and are bound together in one cover which protects them from outside influences.

Bell Wire from Cable to Fuses.—When using a "twin wire" cable, the ends near to the charges, are necessarily close together, and in order to join them to the two ends of the series of charges to be fired, which are often a good distance apart, two separate lengths of wire are required. Cheap insulated "bell" wire is generally employed for this purpose. The "bell" wire leads should always be long enough to ensure that the more expensive "twin wire" cable is far enough away from the charges to prevent its being damaged by flying debris.

Cable Box and Reel.—When portable cable is used it is, of course, necessary to run the cable out just before firing and wind it up again before work is resumed. Under these circumstances the cable is best carried on a reel as shown in illustration No. 8.

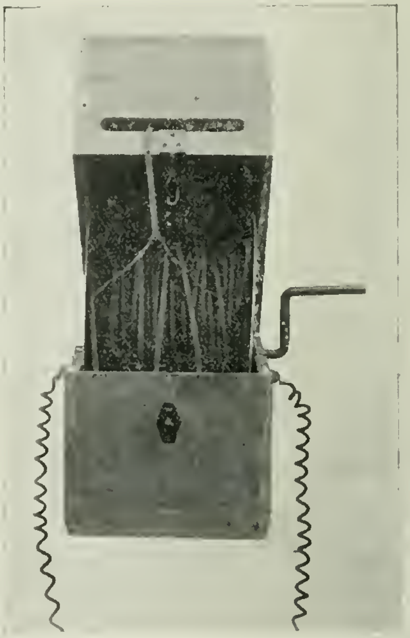


Illustration No. 8.—Cable Box and Reel.

Permanent Cables.—Portable exploders can also be used in conjunction with permanently fitted cables, and in shaft sinking and gold mining in general where electric blasting becomes the regular practice, permanent cables will be found most convenient. This matter will be dealt with in more detail under our next heading.

Permanent Electrical Installation.—Permanent installations are used to a certain extent in Europe and America for electric blasting, and in our opinion, this is the only practical method for large scale operations in Rand gold mines. This naturally raises the question whether firing should be done from the surface or from the shaft stations of the different levels. On certain large mines it may be found extremely difficult to adopt the surface method of firing, but where possible, it should be installed, mainly for the reason that the whole mine can be fired by one individual, after all the workers have been brought to the surface. Surface firing has also the advantage that precautions arranged with a view to preventing premature explosions can be made more effective.

We will now give a brief description of the system in use at the Meyer and Charlton Mine, and indicate how it is operated The system was devised by Mr. F. W. Girdler Brown, General Manager, and Mr. F. Chamberlain, chief electrician of that mine. Four levels have been fitted up for electric blasting. The current is carried down the shaft in a seven core highly insulated cable, the wires being 7/20 S.W.G. and is taken through each level by means of a "twin core" cable, the wires again being 7/20 S.W.G. highly insulated and armoured.

The miners charge up their holes and leave the wire leads of the fuses unconnected. After all the day-shift workers have been hauled to the surface, three or four specially trained men go underground to connect up the charges.

Each operator, having connected up a given series of charges, tests the series with a galvanometer in order to find out whether or not there are any defective connections. If the galvanometer indicates any defect the operator must trace it and put it right.

Illustration No. 9 shows a circuit being tested.

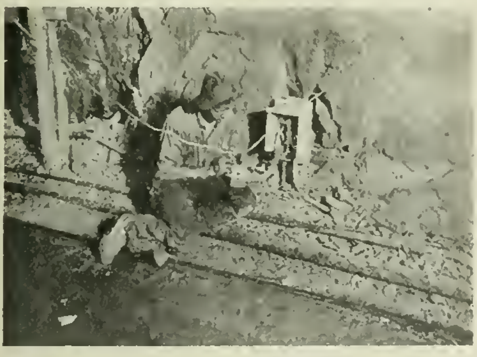


Illustration No. 9.—Testing a Series of Charges Underground.

When everything is in order he connects this circuit with the firing cable of the level. All other circuits are dealt with in the same manner. Illustration No. 9 also shows one of the boxes in which connection is made to the firing cable. These boxes are fixed on the cable at intervals of 60 ft. along each drive.

Further operations will be more easily followed by referring to illustration No. 11 and in order that this illustration may be more readily understood, we reproduce a photograph of the switchboard in the firing cabin at the surface.

Illustration No. 11 shows the underground arrangement for two levels only; the other levels are equipped in the same way.

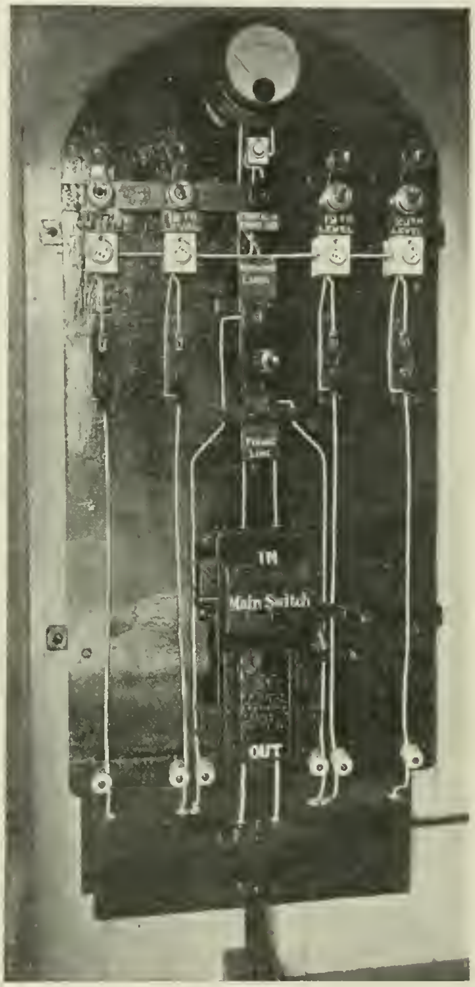


Illustration No. 10.—Switchboard in Surface Firing Cabin at Meyer and Charlton G.M. Co., Ltd.

Having, as already stated, connected every circuit to the firing cable, the operator makes sure that everyone is out of the level. He then closes and locks gate E (of which we will re-produce a photograph later on) in the cross cut, and proceeds to the shaft station. He then unlocks the box containing the double pole switch (F), and puts the switch in the "on" position, thereby establishing connection between the firing cable of the level and the main cable in the shaft. When everything is in order, he is hauled to the surface. The act of closing the gate operates on an automatic switch, which closes a break in the indicator line at that point.

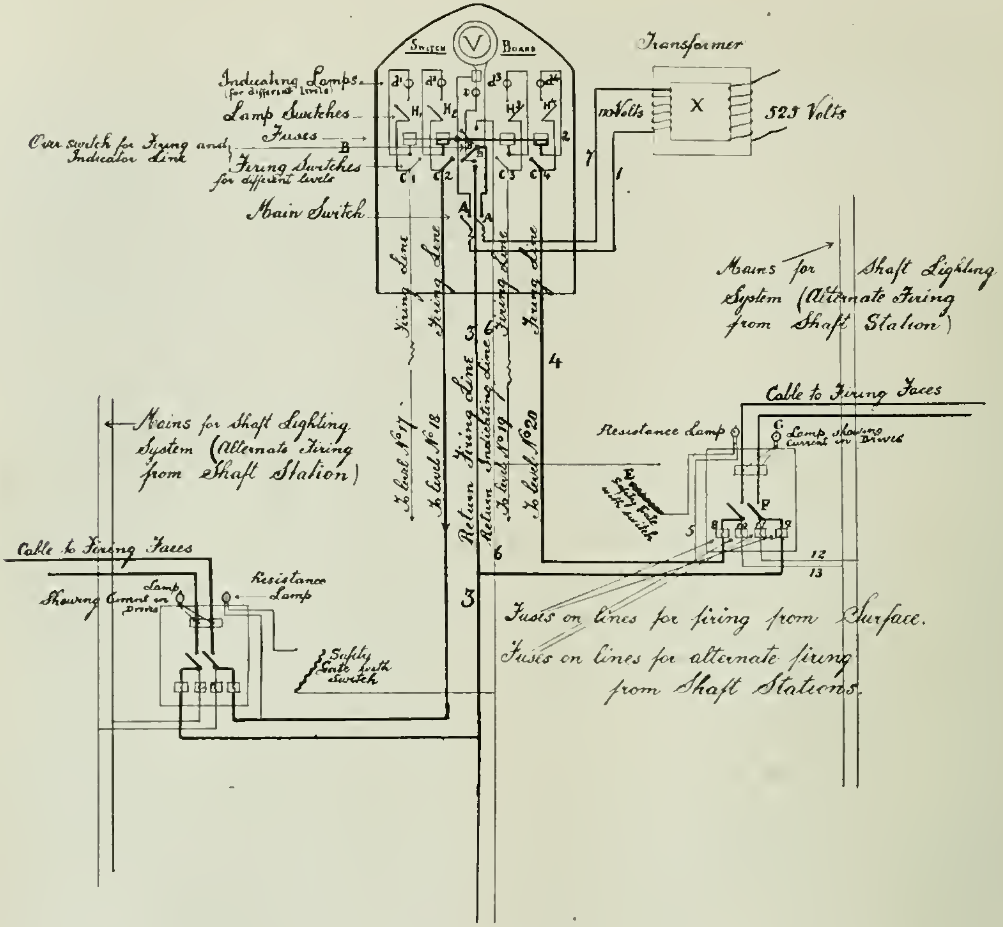

Switch Board
V
Transformer
X
110 Volts
525 Volts
Indicating Lamps (for different levels)
Lamp Switches
Fuses
Over switch for Firing and Indicator Line
B
Firing Switches for different levels
Main Switch
d1
d2
D
d3
d4
H1
H2
H3
H4
C1
C2
C3
C4
A
A
1
2
7
Firing Line
Firing Line
Firing Line
Firing Line
To level No 17
To level No 18
Return Firing Line 3
Return Indicating Line 6
To level No 19
To level No 20
4
3
6
Mains for Shaft Lighting System (Alternate Firing from Shaft Station)
Cable to Firing Faces
Lamp Showing Current on Drives
Resistance Lamp
Safety Gate with Switch
Mains for Shaft Lighting System (Alternate Firing from Shaft Station)
Cable to Firing Faces
Resistance Lamp
G
Lamp showing current on Drives
E
Safety Gate with Switch
F
5
8
9
12
13
Fuses on lines for firing from Surface.
Fuses on lines for alternate firing from Shaft Stations.

Illustration No. 11.—Sketch of Electric Blasting Installation at the Meyer and Charlton G.M. Co.

Operations similar to those we have described have been going on in the other levels at the same time. We may say that the operator in each level is the only one who possesses keys of the double-pole switch and the gate in the cross cut.

When the connecting up operations underground have been completed, the mine captain proceeds to the firing cabin on the surface, of which he alone has the key, and opens it. He then unlocks the double-pole switch (A), of which again he possesses the only key, and places it in the "on" position. If current is on, it will be indicated in the voltmeter (V). He then puts the single-pole "over switch" (B) in the upward position and in so doing switches the current into the indicator line. After this he closes lamp switches (H_1, H_2, H_3, and H_4) and waits. As soon as the gate (F) in the cross-cut is closed, lamp d_4 will glow indicating that all is ready in that level. After all four lamps d_1, d_2, d_3, and d_4, have become incandescent indicating that al levels are ready for firing, the mine captain opens the lamp switches and puts the "over switch" (B) in the downward position thereby switching the current off the indicator line and into the firing line. When current is in the firing line the small red lamp (D) on the switchboard will glow.

He then closes switches C_1, C_2, C_3 and C_4, in turn, and in doing so fires the charges in each level. It is, of course, advisable to leave each switch "on" for a definite interval of time, say, 3 sec., and also to allow time for all charges in one level to explode before switching the current on to the next level. The reason for this will appear later. Having fired all the levels, the mine captain puts all switches on the switchboard in the cabin in the "off" position, locks up the double-pole switch and then locks the cabin. Next morning, before work is resumed, the double-pole switch at the shaft station of each level is put in the "off" position and locked up.

If the person carrying out this duty should notice that the red lamp (G) situated near the switch box is glowing he knows that the switches in the firing cabin have not been put in the "off" position. In other words, the current is still on the firing line and he must report this irregularity at once. This lamp constitutes a very important safeguard, because every level gets the same warning, and some one is sure to report the matter. It will be observed from Illustration No. 11 that arrangements have been made to enable firing to be done from the lighting current of the mine if desired, but in order to use this arrangement it is necessary to do the firing from the station of each level separately. When not required, the current on this auxiliary system is cut off by removing the fuses on both the outgoing and return wires, which connect the wires of the lighting system to the firing cable of each level. It has been put forward as an objection to firing from the surface that it will be difficult to tell whether all levels have been fired, and impossible to count the shots. The first objection has been met at the Meyer and Charlton by establishing a telephone system between the firing cabin and the different levels, and by this means it is possible to hear the blast proceeding. To make this arrangement effective, it is, of course, necessary to wait until all charges have exploded in one level before switching the current on to the next, hence the interval of time already referred to, which is necessary between the firing of the different levels. As for counting the shots, that is impossible under any conditions.

We now come to consider the question of the dangers attendant on the system of firing by means of a permanent installation, such as we have just dealt with. First there is the danger of premature explosion through accidental closing of the circuit. A glance at the sketch we have just been dealing with, will show that when the current is not required it is cut off at the surface by means of a locked double-pole switch in the firing cabin. If this should be accidentally left "on," there is the single-pole "over switch," but if that also should be left in circuit, there are the small switches, one on the firing line of each level. Then if it should so happen that all of the switches in the firing cabin are left in the "on" position, there is the double-pole switch at the shaft station of each level. Finally, this switch, as has already been said, operates a red lamp, which acts as a danger signal in the case of all switches being left on. There is very little chance of all of these safeguards being neglected at the same time.

We now come to deal with the danger of premature explosion through current leakage. Illustration No. 12 will serve to show what we mean in this respect.

Referring to Figure 1, (A) is a single-pole switch in the "off" position. At (B) the uninsulated ends of two fuses, which are connected together, have been stuck into the wet ground, while at (C) the bare wire of the firing line has been exposed. Assuming that the current is left accidentally on the firing line while the work of connecting up the charges is in progress, it is possible that under certain conditions, even when switch (A) is in the "off" position—or, to go further, even when the circuit is also broken at another point, as shown in the sketch—sufficient current to fire some of the charges can flow through the earth from (B) to (C).

If, however, as shown in Figure 2, both the outgoing and return wires on the firing line are broken by means of a double-pole switch, or to go further, as has been done at the Meyer and Charlton, if both wires are cut at two separate points by means of double-pole switches, the danger of premature explosion through current leakage is avoided. Nevertheless, our examples show the necessity for the use of double-pole switches in an installation of the kind we have been describing. Before leaving this subject, we should like to say that we consider the scheme now in use at the Meyer and Charlton reflects great credit on the officials concerned, especially when it is remembered that the scheme of firing practically the whole mine from the surface was in actual operation four weeks after the first steps were taken towards the introduction of electric blasting.

Admirable as the scheme is, we are able to state that the officials referred to, are quite alive to the fact that it is not perfect. They have been handicapped in one or two respects through lack of suitable material, and they have in view certain alterations which will further increase the safety factor. One of these alterations will result in the gate in the cross-cut at the shaft station of each level forming a part of the firing line of that level, so that even if all switches should be left in the "on" position, no firing will be possible in any given level until the gate of that level is not only closed but locked.

We desire to express our great indebtedness to the Managing Director of the General Mining and Finance Corporation and to the General Manager and various officials of the Meyer and Charlton Gold Mine for their courtesy in supplying details of the method of working and results obtained at the above-mentioned mine. We should like to place on record our appreciation of the services rendered by Mr. G. F. Ayers, one of our colleagues, in assisting to collect the data necessary for the preparation of this paper.

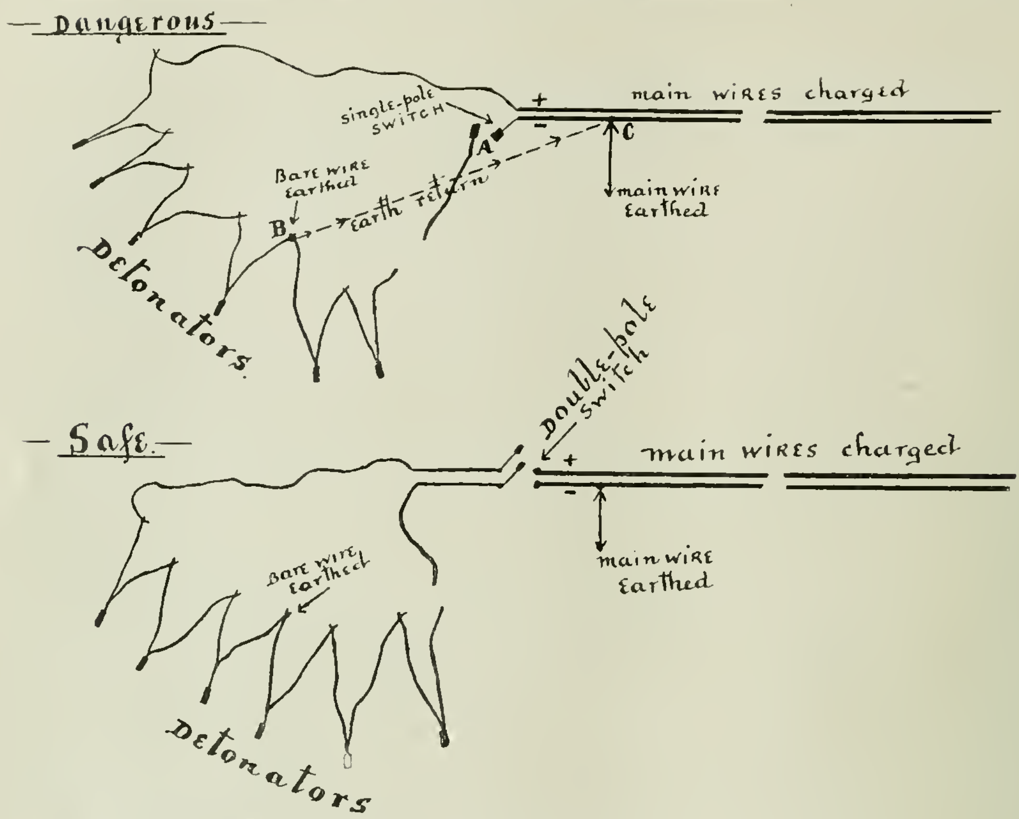


Illustration No. 12.—How to Avoid Premature Firing Through Current Leakage.

In conclusion we may say that our endeavour has been not only to state the case for electric blasting, but to put forward some information regarding its practical application which will be of assistance to those directly concerned in its use. We are aware that in the course of our remarks we have touched on several points which leave ample room for discussion, and we hope members will come forward and give their views on this highly important subject.

Illustration No. 13 (a).—Gate in Cross-cut (Open).

APPENDIX "A".

Permanent Electrical Installation at Meyer and Charlton G. M. Co. Ltd.

Path of Current.—The transformer supplies alternating current so that it does not matter from which point we start to trace its path. The arrangements for all levels are similar, therefore, it is only necessary to deal with one level. Referring to illustration No. 11: (Level No. 20).

Illustration 13 (b).—Gate in Cross-cut (Shut).

Indicating Line.—(That is the line carrying current to the indicating lamp on switch board at the surface. When the lamp glows firing may be proceeded with in the level).

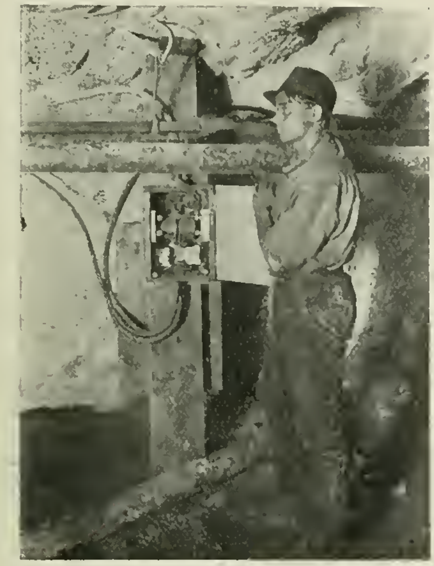


Illustration No. 14.—Firing Box at Shaft Station.

Leaving the transformer the current flows through line 1 and main switch A : into the fuses on switch board : through line 2 and indicating lamp d_4 : through switch H_4 into firing line 4 : branches into line 5 and then through resistance lamp and switch on gate E (gate being closed) into the return indicating line 6 : through "over" switch B (which is in upward position) and back through switch A and line 7 to transformer.

Firing Line. — Again leaving transformer through line 1 the current flows through switch A into the fuses on switchboard : through switch C_4 (closed) into firing line 4 : through fuse 8 (at shaft station) and switch F ; into firing cable in the level : through electric detonator fuses ; back to switch F : through fuse 9 : into return firing line 3 : then through "over" switch B (which is in downward position) : back through switch A and firing line 7 to transformer.

When firing from shaft stations by means of alternate arrangements connected with shaft lighting system, the current flows through line 12 into fuse 11 : through switch F along firing cable in the level : through electric detonator fuses back through switch F and fuse 10 : through line 13 to lighting main.

Mr. E. Pam *(Member of Council)*: It is a great pleasure to me to propose a vote of thanks to these three gentlemen for a most interesting paper on a subject which has come to the front again quite recently. I think the order in which the authors have put the various points, that is, health, safety and economy, also reflects a very sound feeling which is at present existent on the Rand. I think it is clear that the reason why we have not adopted electric blasting so rapidly as the coal mines have, is that it is obviously not so vital on metalliferous mines, and I think it is only fair to put on record that the majority of the metalliferous mines in England still use ordinary fuse. With regard to the manufacture of fuse, a statement is made that the burning is not quite regular, but I have so far been absolutely unable to prove any case of so called running fuse, and from enquiries from Inspectors of Mines and Explosives am lead to believe that the danger is non-existent.

Before criticising the question of economy, I should mention that I agree at once that it is a small matter and does not compare in importance with health and safety ; but I would call attention to the fact that in the calculation the cost of the extra blasting gang has not been taken into account. Apparently three or four men are used on the Meyer and Charlton for four levels, so that the cost cannot be neglected.

Another point I note is a discussion on the question of blasting a development face in one shift. The final conclusion appears to be to blast the cut early and then go back and blast the round. I am afraid that is not very satisfactory, because the men are asked to go back into the smoke—the very thing we are trying to avoid. Where it is possible to give a man two places, there is no difficulty. There is a possibility that by the more efficient blasting of the cut electrically, it may become practicable to blast the whole cut and round together, and if this can be proved it will be a big advantage both in safeguarding health and in economy. I think we should note also that although we are all agreed that there is a lot of smoke and dust about at blasting time, there are, comparatively speaking, a small proportion of our men suffering from that. Very serious efforts have been made as the authors mention along other lines. I took out some figures on our mine*, and find that less than 10% of our underground employes can be harmed by smoke and dust due to blasting, that is to say a very large majority of them come up down cast shafts, and of those working on the up cast shafts a proportion of them again are already on the surface by the time blasting commences. Nevertheless 10% means a very considerable number and they also should be protected. One other point is that when electric blasting is adopted on the Rand, it will mean a large capital expenditure, and probably will not be used very eagerly by all parties, so that it might be worth pointing out that a very large proportion of the maximum of good can be done if the mines adopt a system by which the bottom levels of the up cast shafts are immediately equipped leaving the rest of the mine for further consideration. There is no question that the bottom three or four levels where the connections are few and where the boys or men may have to walk up a level occasionally are far from healthy at blasting time and it is on this portion of our mines that our attention should first be focussed.

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*Geldenhuis Deep, Ltd.

I should like to propose a sincere vote of thanks to the authors for an exceedingly interesting and instructive paper.

Prof. J. Cellier (*Member*): I have much pleasure in seconding Mr. Pam's vote of thanks to the authors of this paper, Messrs. Cullen, Donaldson and Waters.

I feel that our knowledge of electric blasting has been brought entirely up to date. In addition to the expense of the wages paid to the firing party, as mentioned by Mr. Pam, the initial outlay on cables, etc. is to be considered, and further, the maintenance on portable cables, bell wires in stopes, etc.

In metalliferous mines the application of electric blasting will probably always be more limited than in collieries, as in the latter the mineral being softer the whole round in headings can more readily be blasted at one time than in the case of harder rock. In breaking down the mineral also the charges are not so heavy as there is usually some form of undercutting, and further the area over which the holes are placed is not so confined, comparatively speaking, and therefore more shots can be fired simultaneously.

As regards exposure of blasters to fumes and dust during blasting time, much can be done by properly regulating the times of blasting in different parts of a mine, as also mentioned by Mr. Pam.

Mr. E. Roberts (*Member*): I would like to ask the authors of this paper if they could give us any statistics with regard to the number of misfires in vertical shafts. I have been informed that on a mine where electric blasting is used for shaft sinking, numerous misfires have occurred—sometimes none of the shots explode, and occasionally 8 or 9 misfires are found. I should like to know if this is true.

Mr. W. Waters (*Associate*): I may say that we have no definite particulars of the misfires at the mine in question (not the Meyer and Charlton) but I know that on several occasions misfires have occurred. The mine captain went down one day and found that out of a total number of 38 shots, only 2 had been connected up, so that 36 had no chance of exploding. The manager has informed me that practically all the misfires have been due to faulty connections on the part of the miners, and cannot be attributed to any fault in the electric system.

Mr. E. Roberts (*Member*): Is that instantaneous blasting?

Mr. W. Waters (*Associate*): They are using both instantaneous and delay action fuses. They started first of all with instantaneous but found the concussion too great, and consequently are using a certain proportion of delay action.

Mr. W. S. V. Price (*Member*): In the event of tamping with the delay action fuses, would it not be very likely to break the connections? Say, for instance, the round is charged up in a drive, and the end of the delay fuse is just at the collar of the hole, in the way of the tamping stick. Is there not a danger that the connection would be broken?

Mr. W. Waters (*Associate*): When using electric detonators, and also explosives generally, the usual care must be exercised. There should be no reason for connections being broken during tamping operations if ordinary care be taken. The figures given in our paper prove that at the Meyer and Charlton, there have been fewer misfires with electric blasting than previously with safety fuse. This system has only been used for about two months, and consequently there were only a few trained men to begin with. As time goes on and when all the miners have become accustomed to electric blasting, misfires should, and I think will be reduced, or possibly eliminated altogether.

Mr. E. M. Weston (*Member*): I have listened with interest to this paper. The middle portion of it while useful as a summary of the methods employed can be found in any modern text books and calls for little criticism. It is apparent from the remarks of Mr. Roberts that the somewhat lengthy history of its attempted introduction into South African metal mining has certain unwritten episodes about it which might be of great interest. We must turn to the claims made on behalf of this system. Regarding health conditions, it must be admitted that burning fuse does give off CO and CO_2; but I venture to say that in comparison with other sources the amount is negligible and no one ever suffered any great injury from it. I should be pleased to learn when it was proved that smoke from either safety fuse or any other source had the power to "naturally hold in suspension a large quantity of very fine dust." These experiments must have been very interesting and it is a pity the Society has not had the benefit of them before. The authors state that electric blasting will remove the necessity of men having to travel through dust and fume laden drives and stopes; but this, as the authors acknowledge later on, it will not do, as men will have to return through them just the same after blasting their cuts in development work. I regard the presence of smoke from fuses as a very good indication of the presence of noxious gases and a danger signal, which with electric blasting may be absent, tending to cause more gassing accidents. Regarding the question of safety, there is, as the authors admit, much to say on both sides; but I would ask them to remember that there are more fools and irresponsible persons per square yard underground on these fields than anywhere else in the world, and that it will take a good many big accidents to show where the dangers lie under these conditions. Where there are powerful currents for lighting, haulage and ventilation purposes being led about underground there is always grave risk of leakage, and a recent death of an electrician on a station shows that such leaks do occcur. I should scarcely like to claim "that accidents from premature explosions should be practically impossible." I do not endorse the authors' suggestion that electric blasting should be made compulsory for shaft and winze sinking, and if the authors had first obtained the opinions of a few practical men on this subject, or had themselves practical experience in this work, their opinions would carry more weight. If the Mining Inspectors of the Rand had to write a report on blasting accidents on the Rand I am sure they could also write "few, if any of these accidents would have occurred, if the regulations had been observed." The trouble is that precautions are not observed. When an accident occurs with fuse in a stope or drive, one hole may go; but with electricity there would be a bombardment from which no one would have a chance to escape. Regarding economy, the authors conveniently forget to include the amortisation and interest on the capital cost of all these electrical installations spread over the life of the mine. Nor do they make any allowance for the immense wastage of return cables such as shown down the stopes in their diagram. The blast would destroy 20 to 150 feet of wire along the levels and in many stopes after each firing. Then again the method undoubtedly means the employment of much skilled labour in maintenance of the system generally. That also is left out. Regarding their contention about cut-hole firing, personally I attach no value at all to their claims. If a cut has been properly drilled in ordinary Rand ground it will come if only one fuse be put in it. If the easers be loaded it will generally put them off also with no sign of reduced power, and cases have occurred where the whole round has come. The authors state: "we might add that when cut holes are drilled too far apart at the base it sometimes happens that more than one report is heard and that with safety fuse the miner expects and finds trouble." I might add that such a cut is no cut at all and that electric blasting would not make it a good or a better one. If the miner could be trusted to always put in a proper well thought out cut the problem of avoiding travelling in smoke could be solved just as well with fuse as with electric blasting. There are a few miners who can be trusted to do this to-day. One of my miners recently sank 800 feet of an inclined winze 10 x 7 feet and in doing so fired 150 to 200 rounds without first blasting the cut and had only 2 to 3% of his whole rounds require reblasting. The authors say that the necessity under either system of blasting the cut first "will not appreciably affect the general result." I venture to say that this matter of blasting cuts is the crux of the whole matter, and if we could solve this problem without a serious decrease in efficiency the phthisis problem would be brought a long way towards solution without any reference to electric or fuse firing. As Mr. Pam has shown, it is to-day, with fan ventilation easy to arrange that the great majority of the men are not subjected to fumes and smoke after blasting, and no doubt the $8\frac{1}{2}$ hour bank to bank law has tended in this direction also. There will doubtless always be found isolated areas in mines where electric blasting might be adopted with advantage to render any long travel through smoke areas unnecessary. I cannot follow the reasoning of the authors that the adoption of this system gives any longer working time in the faces. Just the same economy would be gained by employing a blasting gang for fuse firing, and where two shifts must be employed and the work rushed the electric system would cause delay. Most of the advantages claimed for this method are the advantages of single shift working as against double shift working and have nothing to do with firing methods. Regarding misfires, the authors, I am afraid, do not know that the causes of misfires are mostly quite unconnected with the system of firing employed. I have yet to learn that the Society is in any need of drawing conclusions from statistics showing 0·3% differences over a limited period and founded on "best of belief" evidence. In conclusion, though I believe the authors' claims are most exaggerated and founded on lack of knowledge of underground conditions, I consider that electric blasting may in certain mines be found a useful auxiliary to the system at present employed, but that to endeavour to hasten its adoption by over-stating the case is a mistake.

The President: In view of the fact that the subject of electric blasting is arousing considerable interest locally at the present time, this instructive paper is particularly welcome. There is no doubt that it is highly desirable, for health reasons, that no men should be underground when, and immediately after, blasting takes place; and the use of electric blasting holds out every hope that such an improved state of things will be brought about.

I notice that we have with us Mr. Warington-Smyth, the Secretary for Mines. I take it that his presence here this evening is a proof of the keen interest which the Government is taking in all safety measures.

MINING COPPER ORES AT MESSINA.

By J. Allan Woodburn, M.Inst.M.M. (Associate).

DISCUSSION.

Mr. A. L. Chambers (*Member*): As a former Manager at Messina, I have read with interest Mr. Woodburn's paper describing the methods of mining there, and I would like to congratulate him on his most interesting record.

Having had considerable experience in the Northern Transvaal and of the occurrences of copper there, I feel that perhaps a few remarks resulting from my personal observations may be acceptable, as there is very little information available about that district and practically no data regarding the base metals "which should some day be a considerable source of value to the country."

Occurrences of copper have been located at different points over the area extending along the southern bank of the Limpopo River for about 100 miles, and south about 35 miles, approximately. Molybdenite has also been found in this area, while coal measures have been discovered, with coal of very good quality occurring in places; gold and diamonds are also reported to have been discovered.

The country, however, has never been thoroughly prospected, it is covered with thick dense bush, mostly of thorn trees, through which it is often necessary to cut one's way. Water is very scarce and the places where it can be found frequently very far between. Both lions and leopards are at times met with, and sometimes prove a source of trouble and danger. In addition to the above little obstacles, the difficulty of keeping up supplies for a prospecting party is a very serious and expensive question. The natives living in that part of the country are of a very low type physically and mentally, due principally to the frequent failure of their crops owing to the excessive heat, drought and poorness of soil, so that they frequently live just on the verge of starvation. They are very antagonistic to the white man in this way, that they will not give any information regarding paths and trails, places where water can be obtained, and such general information as the engineer or prospector requires, or, if they do give any information, it is generally found to be wrong or misleading. The reason for this is that the district is a very safe retreat for native bad characters, and many of them have not paid hut tax or obtained a pass for years. Their huts, or, more correctly speaking, sun shelters, are hidden away in most inaccessible places, excepting in the case of known and established kraals where the outlaw type leave their small possessions in security in the care of their brothers or friends, so that they are free to roam the country, free from all fear of being caught or rounded up by the Police patrols, which occasionally pass through. Constant watch is kept, and immediately a stranger or white man is observed, they all clear off into the bush, where, owing to its dense thorny nature, it is quite impossible to follow or track them. They have a series of main paths or kafir highways from various points, which it is easy to find, but in many instances there are concealed paths through broken country, and exceedingly dense bush which enables them to easily get about and avoid detection; at the same time news travels almost as quickly amongst them as by the white man's telegraph. An interesting person who certainly deserves mention, is the native doctor, who seems to possess a remarkable knowledge of herbs. They cure fever, dysentery and various ailments, they will also heal up in a short period wounds caused by tigers, such wounds as are generally considered incurable, and from which gangrene almost invariably results, even with skilful medical attention. They also possess a poison, which, on occasion, is given to a person with whom they wish to do away. It is supposed that they place it in the person's food in the evening, no indication of it being apparent until the person is found dead in the morning, when it is reported amongst them that, to translate literally their expression, "he was afraid in the night." Investigations into cases of this nature have so far failed to determine or detect the nature of the poison used.

The area under review is extremely interesting from a geological aspect. The principal features from the north-west part of the area being the gneiss formation, the oldest known formation in Africa, called by Dr. Voigt "The Fundamental Gneiss,"* consisting principally of mica gneiss, hornblend gneiss, epidote, and garnet gneisses. To the south or south-east of these the surface is covered with large areas of sandstone outliers of the Waterberg series, which have apparently been detached from the main belt, represented here by the Zoutpansberg range of mountains, through the intrusion of extensive ultra-basic dikes, which can be traced for many miles, striking north-east to south-west. The intrusion of these dikes appears to be the first of a series of extensive earth movements, which have taken place over a very large area, creating intense pressure, the result of this being the metamorphism of the original formation; very interesting instances of this, both in the nature of contact metamorphism and regional metamorphism being met with, while, from the same cause, shearing and fissuring has resulted. The latter are due principally to intrusion of subsequent or later date dikes, also of very large extent, which have been the principal feature of the second stage of earth movement. The third stage being apparently that of contraction, when the cooling or settling down of the disturbed area took place, resulting in large segregations of white vitreous quartz and the filling of the dike created fissures with a gangue or matter; at the same time mineralising gases and solutions from the magma ascended and by their agency the copper ores have been deposited.

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*See Proceedings "Geological Society of South Africa," Sept. 4, 1905.

The occurrence of, or deposits of, copper are found under various conditions, such as: the pink or red gneissoid rock, where crushing and shearing due to the first movement occurred, and where the later date dikes have opened or created fissures; these fissures have become filled with a gangue in which the copper occurs in the form of irregular lumps and lenticular masses, sometimes accompanied by secondary quartz and hematite. The gangue of these fissures is of a soft nature, very much altered and as it appears to-day is evidently the product of metasomatic replacement, while a curious feature of the deposits is that lumps of iron, in appearance like cuprite, often form part and parcel of masses of rich sulphide ore, the one merging into the other without any line of demarcation. At the water level, where the water fluctuates between the wet and dry seasons, the copper values are frequently found entirely leached out and the remaining gangue or fissure filling takes the appearance of a kaolin; sometimes the leaching has been so perfect that not even a copper stain has been left. In developing or opening up these copper lodes, it is necessary to establish a theory as to the possible and probable direction and extent of fissuring certain dike intrusions would produce in bursting through the earth's crust, the origin of the fissure filling and source from which the mineralising solutions came. I mention this on account of the peculiar nature of copper, which, being soluble, is by the action of circulating descending waters, in this type of country, which has been so much fractured, frequently carried in solution along the lines of least resistance and re-deposited in cavities and along planes which have no direct connection with the lode or true lines of fissuring along which the ascending mineralising solutions came. Without a theory to guide one, a theory which is very easy to prove the correctness of quickly, it is quite possible that considerable sums of money may be spent in useless work, without any sound results being achieved.

Another type of occurrence is that of the nature of igneous sheets, where secondary infiltration of quartz, with sulphides of metal has taken place. Copper has also been found to occur in fissures and cavities in the overlying sedimentary rock, in which cases mineralization also seems to be governed or due to the fissuring caused by extensive dikes. An interesting feature of this type of occurrence is native copper, which has been found in small lenticular masses.

In all the copper occurrences so far reported there does not seem to have been found any instance with an oxidised or carbonate zone, the sulphide or zone of secondary enrichment being found at, or quite close to the surface.

In the early stages of the development of the copper occurrences where the sulphide ore occurs in lenses or lenticular masses, it is possible, by careful mining, to procure an output, which very materially helps towards paying expenses, before sufficient data has been obtained in order to determine the type and capacity of the concentrating or reduction plant most suitable for the future. In this type of mining only the richest ore is stoped, rough sorting takes place underground, and the selected ore is then bagged and sent to the surface, where it is finally sorted, cobbed and bagged for shipment, while the fine from the stope and hand-sorting is screened, and by means of hand jigs a rich concentrate is produced ready for shipment. Scattered through the district are found groups of ancient workings, varying in size from small pot-holes to several hundred feet in length. Frequently the groups of workings cover considerable areas, thereby conveying the impression that an extensive copper mining industry must have at one time been carried on in that part of the Transvaal, for in a number of cases these old workings are for that type of mining by no means shallow, instances being known where they have gone down to nearly 80 ft. vertical depth. In any case there is abundant evidence that many thousand of tons of ore have been mined, therefore the quantity of copper removed must have been considerable and with such slow methods of mining there is no doubt but what to-day is a little known district was at one time a busy and largely populated mining area.

The method of working employed was that of chipping out the ore by means of iron gads and stone hammers. Such methods of mining being possible owing to the soft and peculiar nature of the gangue, which, on exposure to the air, disintegrates rapidly; for instance a face, which is apparently hard to-day, will by exposure overnight, soften and yield quite a lot of ore to a gad or small pick next day. The ore was then collected in baskets made of reeds, and removed to the surface, where sorting took place, the rich copper ore being collected and sent to the smelters while the gangue was used for stope filling. It is quite an open question as to whether the discarded gangue was used for stope filling while work was in progress or not; possibly it was to some extent, but a peculiar feature of these old workings is that as we find them to-day, they are in appearance elliptical depressions with a small raised bank of debris conforming to the sides of the old open stopes, and showing every evidence that they were filled in by hand. Native tradition says that many mines existed in various parts of the country when their forefathers came into it, and that when the white man came, their kings and chiefs passed a law that these mines were all to be filled in, which was done. Further, any native disclosing these ancient mines to a white man would be put to death by starvation and torture, their idea being that if the white man found these mines, they would remain, and if they failed to find them they would go away. The truth of the above traditions is strongly confirmed by the fact that even to-day natives are extremely reluctant and almost refuse to disclose the position of ancient workings, and in many instances in the Transvaal crude methods of obliteration are only too apparent. Personally, I have visited ancient workings whose existence has been disclosed by the caving of a piece of ground leaving a small opening into which game has bolted, thus attracting attention to what would otherwise not have been noticed. In one case, a stope, over 100 ft. in length, and over 50 ft. in depth, with openings between pillars leading to what appeared to be an additional extensive stope, was exposed, and yet there was not on the surface the least evidence to lead one to anticipate the existence of such extensive mining operations. In some instances circular shafts have been sunk to one side of the ore body mined, and where these have been cleaned out, or have become exposed by the subsidence of the débris filling, the old notches or holes where the workers placed their toes in going up or down, are clearly visible.

In carrying on the mining operations for copper, that industry also entailed others, such as mining iron, smelting and making the implements required, also smelting the copper ore into ingots and bars, the making of copper ornaments for the neck, wrist and ankles. Owing to the soft nature of the copper, it was drawn into wire by passing it through holes in plates made of iron, gradually reducing it to the required size; some of these old plates for wire drawing were in existence among the natives until quite recent years, and also specimens of their work in copper ornaments, while the iron gads and stone-hammers are occasionally found at the bottom of the old stopes, the head of the gads being burred over, similar to the drill heads in use to-day. The copper produced by smelting was first cast into ingots similar in shape to the old oblong cowbell, having along its apex a series of rows of spikes, the number of spikes denoting its value for trade purposes, so that in a sense it represented a crude species of coinage or the equivalent of such. For the purpose of making ornaments these ingots were again cast into bars or rods about $\frac{3}{4}$ in. dia. and about 16 in. to 18 in. long and these again were drawn out or worked according to the various ornaments required.

There is no doubt, judging from the size and extent of the old workings, that their origin is of considerable antiquity and it is very probable that it dates back to the period of the occupation of the country by those people who built the Zimbabwe, and the metal copper occurring in such quantities as it does in a soft gangue, and itself of such a soft ductile nature, the industry was, after the departure of the originators, to some extent carried on by the natives and gradually died out. The idea that these ancient workings are purely of native origin is, on account of their size, and to those who know the African native, absolutely untenable; further, they must have entailed a certain amount of organisation in the mining and the manufacture of the iron implements, which took place many miles away from the copper mines. The organization of an industry on such a scale as the mining of the copper ore and the smelting operations, the Kafir is incapable of. Their antiquity may be considered further established by the fact that slow growing trees of considerable size are found growing in the debris in the old workings themselves. A curious feature regarding these old workings is that no matter how large an area they cover, there are no indications left as to the type of housing accommodation which was used, and no foundations of any sort, such as one might expect from such an extensive mining industry. From the days of this early industry to the present time is a very far cry, and no doubt much that is interesting could be written if only the controllers of the former industry had left behind them assay plans and records, but I question if even these could be as valuable as the paper in which Mr. Woodburn so ably describes the modern methods employed.

The President: I had the pleasure of paying a visit to the Messina mine a few weeks ago, and through the courtesy of the management was enabled to study the many interesting technical points that the mine presents. I can, in a small way, corroborate what Mr. Chambers says about the difficulties of the country, having seen the fresh spoor of a lion at the Company's pumping station and having incautiously rubbed shoulders with a wacht-een-beetje thorn bush. No doubt, when the railway reaches Messina, towards the end of this year, an impetus will be given to prospecting; and the copper industry of the Northern Transvaal will then begin to expand from the very promising beginning it has made at Messina.

VENTILATION OF THE MINES OF THE RAND: THE PROBLEM OF OBTAINING HEALTHIER CONDITIONS.

(Read at September Meeting, 1913.)

G. H. Blenkinsop, M.Inst.M.M. (Associate).

DISCUSSION.

Mr. James Whitehouse *(Associate)*: We are greatly indebted to the author of this paper for again promoting discussion on what we all realise is a most important subject. Lest, however, his paper may create some misunderstanding in the minds of those not conversant with underground conditions, I would like to draw attention to several of his remarks.

The author states "It is scarcely necessary to say that the current of air through mine workings, especially when it is charged with moisture, does not mean decreased temperature or decreased impurity." This statement is obviously wrong, since, if the air in a mine is moving displacement must be taking place, and fresh air must be entering the mine to replace that already present which in turn is discharged into the atmosphere. With regard to the influence of artificial ventilation it would be interesting to know whether the author of this paper has any information as to what its effects are in a mine, and what conditions exist before and after installing a fan. If he will refer to my paper published in the *Journal* of the Society in May 1911, he will then see an instance of extraordinary improvement on the installation of a fan in the mine.

The second point to which I would like to draw attention is the author's statement that "the temperature of the air passed through the mine workings by means of fan ventilation will on the Rand be above the limit of safety." I should like to know what is the limit of safety? The mines of the Witwatersrand as a whole are very much cooler than those of other parts of the world, being considerably lower in temperature than the English collieries, and very much cooler than the tin mines of Cornwall. The author further states that "the minimum increase of rock temperature is easily put at 1° F. for every 220 ft. of workings traversed, but to that in the case of the Rand mines must be added heat produced by oxidation of pyrite and decay of timbers." I would again refer the author to my paper of May 1911, where he will find that the increase in rock temperature is 1° for every 254 ft. I should also like to ask the author what allowance he proposes to make for the oxidation of pyrite in the mines of the Rand

Again the author states that "we shall be making a very moderate allowance if we place the increase of temperature at 1° for every 200 ft. of mine workings traversed." Well, if this statement is correct, I should like to know what the author would predict for a current of air which might be travelling many thousands of feet. Are we to assume that in this case the temperature of air would be so raised that it would burn the hair off a man's head? I can hardly think he is serious in this, as one assumes that if the temperature is raised beyond what the rock temperature is at that depth, that heat would immediately be transmitted from the air to the rock, and the former thereby cooled. It is difficult to understand from where the air is going to derive its heat. The temperature of the rock at 4,000 ft. is 80° F. and assuming that the air is heated up by some artificial means, which the author can perhaps explain, the moment the temperature was raised above 80° the heat would, of course, be transmitted to the rock. If the author's figures are correct the temperature of the air leaving the Village Deep to go into the Village Main Reef would be 110°, but as a matter of fact it is 76°, which I fear proves the author's statement to be absolutely incorrect.

Finally, I should like to ask whether the author has had any experience in conveying to different working points fresh currents of air through pipes as suggested in his paper. On the

Village Deep Mine the fan circulates 260,000 cu. ft. of air per minute, and I should like to know what size pipe the author considers would be necessary to carry this quantity of air into the mine, and what the water gauge would be? Having delivered to the working places his fresh supply of air, what becomes of it when it leaves that place? He will, I think, discover that it will pass into other stopes on its way to the surface unless he provides a diver's equipment for every man in the mine. For the ventilation of isolated workings it is a common practice to carry air to the faces by means of air pipes, but this has always proved a troublesome arrangement, since the heavy blasting operations often flatten the pipes throughout their whole length.

Quite apart from the cost of carrying out this scheme, I would suggest that it is impracticable, and a thing that we could not for one moment consider for the purposes of ventilating the mines on the Rand. I disagree with the author when he says that it is impossible to supply fresh currents of air to working places by means of a a fan. He apparently assumes that one current of air traverses the whole mine, which is, of course, not correct. Where a fan has been installed it is possible to supply the working place with fresh currents of air by splitting. By this means one is sure of getting good ventilation. By using pipes I should be quite sure of not getting it.

Mr. A. J. Walton *(Member)*: I think the most striking point in Mr. Blenkinsop's paper is his statement that the underground temperature of air increases 1° F. for every 200 ft. of mine workings traversed. This is not a statement which is borne out by fact, the temperature of the air at the bottom of an up-cast shaft being practically constant and apparently independent of the temperature of the air entering the mine. To prove my point, I had the temperatures taken at the bottom of No. 2 shaft of the Rose Deep (which is up-cast), on the 16th inst, when the temperature was 73° F., and the surface shade temperature 69° F. At this shaft the ventilation is induced by a Rateau fan which delivers 80,000 cu. ft. per min. against a $1\frac{1}{2}$ in. water gauge. The air is coursed by means of doors, but even if all the air short circuited the minimum distance which would have to be traversed from down-cast to up-cast is 5,600 feet, which, on the author's "moderate allowance," would give a temperature at the up-cast shaft of 97° F., or 24° F. above that actually recorded. As a matter of fact the actual distance traversed by the air is over 9,000 ft., which would give a temperature of 114° F. or 41° F. higher than that recorded.

The author makes a big point of the heat generated by the oxidation of pyrite and decay of timber, etc., but does not make any allowance for the evaporation of water, which I notice is one of his objections to fan ventilation. Possibly some of our chemical members may be able to throw some light upon the amount of heat likely to be generated in the former process, but I am of opinion that the evaporation of water will more than absorb all the heat generated in the process. The simile which the author makes of fan ventilation to fanning one's self in a theatre is bad, as, with the fan ventilation, fresh air is being drawn from the surface all the time, while the other operation is simply disturbing a more or less stagnant atmosphere. If fan ventilation feels cool then one object is attained as less fatigue results to the worker than when working in a place which feels hot.

I should like to ask if the author is aware that several fan ventilating systems are installed on the mines of the Rand, and that where such are in use the mines are generally acknowledged to be cooler and the conditions better than before the systems were installed? The author, with many others, accuses fan ventilation of raising dust, which is probably the soundest argument against the employment of fans, but at the same time it is not one which presents serious difficulty. A liberal use of atomisers in the airways will easily produce a sufficient degree of humidity in the atmosphere to prevent the raising of dust, especially if the roads are treated with some substance which will absorb moisture, such as molasses. I have used a solution of calcium chloride with success for a similar purpose in coal mines at Home. There is produced in every mine a quantity of fine dust at blasting time, which remains in suspension in the air for a long time even if there is no ventilating current. This dust is probably the worst we have to contend with from a health point of view and can only be removed by exhausting the air from the mine. This means that sufficient fresh air should be sent into a mine between shifts to fill all the cavities which have been formed in years past by mining and which are open to the ventilating current. To give some idea of the quantity of air required, I will assume the case of a mine which has been in existence 10 years and has produced a matter of 360,000 tons per annum. The cavaties thus formed will have a capacity of approximately 48,000,000 cu. ft., but some will probably be caved or otherwise closed, so that it will be reasonable to assume that 24,000,000 cu. ft. of fresh air should be put into the mine in 3 hr.—the time between shifts—or 133,000 cu. ft. per min. This with a fan is a simple matter and would in most cases only mean the expenditure of a small amount of capital, while the actual running cost would be low the horse-power required at a W.G. of 1·5 being about 50.

The author states in one part of his paper that fan ventilation is able to reduce the very high temperature which obtained in the mines of the Rand when they depended solely on "natural ventilation," and in another place says "The temperature of the air passed through the mine workings by means of fan ventilation will on the Rand be above the limit of safety." The two statements appear to conflict, but it will be interesting to know what is meant by "the limit of safety." (Mr. Blenkinsop: 75°.)

The author further states that fan ventilation does not give a supply of fresh air to all working places, but the same objection can be urged against his method of utilising Root's blowers. He may ventilate the end of a drive with fresh air or even some stope faces, but the air must pass out in some way, and the natural tendency is to rise through the stopes so that many of the rock getters and all the tramming boys will be supplied with foul air. The author assumes a mine with 2,000 workers underground, and estimates the cost of the plant at £10,000, but says nothing at all about the cost of running. The old Mining Regulations specified 70 cu. ft. of air per person per minute as the minimum requirement, or for the mine under consideration 140,000 cu. ft. per min., and this is where the matter begins to get interesting. He does not state what pressure he proposes to work the blower at, but for the purpose of calculation I will take a 6 in. water gauge which appears to be the usual pressure for blowers on smiths' hearths. I will also assume a very moderate depth of shaft, viz., 1,000 ft., and that one half of the pressure is absorbed in sending the air down the shaft, and for the sake of comparison that the quantity is the same as that taken in the former example for fan ventilation and so determine the size of pipe required in the shaft.

Now the usual accepted formula for the friction of air in mines is :—

$$h = k\,\frac{L \quad P \quad V^2}{A}$$

where h = water gauge in inches.

k = the co-efficient of friction.

L = length in feet.

P = the perimeter of the air way.

V = velocity in thousands of feet per min.

A = area in square feet.

k is the variable quantity depending upon the shape and uniformity of the air way, and for this purpose can be given a value of 0·0004, so that if a pipe of circular section is used we get :

$$3 = 0{\cdot}0004\left(\frac{1{\cdot}000 \times 3{\cdot}1416x}{0{\cdot}7854x^2} \quad \frac{\times \left(\frac{133}{0{\cdot}7854x^2}\right)^2}{\phantom{0{\cdot}7854}}\right)^2$$

where x = diameter of the pipe, or a required diameter of approximately 7 ft. This taken in conjunction with the size of pipes which would be required on the levels is quite impracticable and would run to a cost of several thousands of pounds for pipes alone and the horse power required would be about 180, any decrease in the size of pipes would increase the power required enormously.

The author has probably based his estimate on a much smaller quantity of air, but a sufficient quantity must be supplied not only for men actually working at the faces, but also to reduce the quantities of noxious gases throughout the mine to the limits set by the regulations, and though the quantity will vary on different mines it is not safe to calculate on less than 70 cu. ft. per person per minute.

The ideal system of ventilation is one where the gases and dust produced by blasting are removed in the shortest possible time and where practicable the mine should be thoroughly swept by fresh air while no one is in the workings and the current reduced while men are at work. This can only be done economically by a fan installation and not by any method utilizing innumerable small pipes which cannot, except by working at comparatively high pressures, deliver a sufficient quantity of air to the mine.

Mr. J. H. Veasey *(Member)*: The Mining section of the Society owe Mr. Blenkinsop a debt of gratitude for having brought forward that important subject "Ventilation" in a form which at once invites and demands discussion. Coal mining, on account of the dangerous gases given off from coal, at an early stage demanded efficient ventilating systems for safety of life and property. Metalliferous mines, on the other hand, cared little about ventilation so long as a candle could be persuaded to burn. Fire damp was absent, and in an emergency they could fall back—although at enormous cost—on the contents of the compressed air mains. These condition are now changing, thanks to legislation, the Phthisis Commission, and the conscientious resolve on the part of employers and officials to improve underground health and sanitation. The author's scheme of blowing fresh air down the mine in pipes to be liberated at the working faces is ideal. The miner would be supplied with pure fresh air under the worst conditions—even a coal mine full of carbonic acid gas—in much the same way as a diver, but at normal pressure. His scheme is independent of natural ventilation, and consequently does not admit of assistance from natural sources. The author's pipes are the "main air intake," the rest of the mine becomes the "return air way." This is good for the miner, but bad for others. The author estimates the cost of his scheme at £10,000 for 2,000 underground workers, but does not mention (1) operating cost or (2) maintenance. This latter item cannot be neglected where the air pipes—probably as light a guage as possible—are exposed to rough treatment in the mine. It would be extremely helpful to the valuable records of this Society if accurate and detailed estimates of these three items of cost could be supplied either by contributors to the discussion or by the author in his reply. The proposal for "one or more blowers at suitable places on the surface" does not seem to accord with Rand ideas of centralisation, and would be impossible at mines like the Cinderella Deep. Taking the author's figures, and assuming an allowance of 200 cub. feet of free air per min. for each person, the problem is 400,000 cub. feet of air per min. through 6,000 feet of main column in the shaft and branches to the working places. Let the author be assured that his proposed initial pressure of, say, half to threequarters of a pound will be insufficient to pass this quantity of air through the given length of pipe of any practicable diameter. I mean his pipe would have to be so big as to render the scheme impossible. Men who started life in older countries may recollect cases in which the whole cross sectional area of a circular shaft 16 feet in diameter was necessary to pass sufficient air for the mine at a pressure of several inches of water.

The larger the airway the less the power required to produce ventilation. The whole area of a circular shaft gives us the largest possible airway. When we put the same amount of air current into a pipe the velocity has to be increased inversely as the square of the diameter to deliver the same quantity, except for the amount that the pressure applied to produce the velocity assists by increasing the density of (compressing) the air. The power required to produce this pressure increases at a rapid rate, for example:—

Case 1.—Ordinary Colliery Practice.

Brick-lined circular shaft 14′ 0″ diameter, sectional area = 154 sq. ft., circumference 44 ft.

The author's data: Depth 6,000 ft.; 2,000 persons.

Different authorities allow from 100 to 200 cub. ft. per person per min. Taking the minimum of 100 we require 200,000 cub. ft. of air per min. and the velocity of current $= \frac{200,000}{154}$

= 1,300 ft. per min.

The formula generally used is:

$$P = \frac{KSv^2}{a}$$

when

P = pressure lbs. per sq. ft.

K = Atkinson's coefficient = ·00000001.

S = rubbing surface = 264,000 sq. ft.

v = velocity of current ft. per min. 1,300.

a = sectional area of airway 154 sq. ft.

Substituting figures—

$$P = \frac{\cdot 00000001 \times 264000 \times 1{,}300 \times 1300}{154}$$

= 29 lbs. per square foot.

and $\frac{29}{5 \cdot 2} = 5 \cdot 5$ inches water gauge.

The horsepower is given by

$$\frac{\text{Quantity cub. ft. per min.} \times \text{press. sq. ft.}}{33{,}000.}$$

$$= \frac{200{,}000 \times 29}{33000}$$

= 176 h.p.

Assume for a moment that it is practicable to introduce into a shaft an iron pipe 6 feet in diameter—though same result might be obtained at less expense by bratticing the pump and ladder compartment thoroughly air tight, and entering them at surface and stations through air locks—and try one calculation for a 72″ pipe.

The formula previously used is adapted for mine airways and would give too high a result for a smooth pipe.

Taking from "Kent's Pocket-Book" a modification of Hawksley's formula

$$h = \frac{Q^2L}{\cdot 39d^5}$$

when

h = inches water gauge.

Q = cub. ft. per sec. $= \frac{200000}{60} = 3{,}334$ in thousands.

L = length feet = 6,000.

a = diameter inches = 72.

Substituting

$$h = \frac{3{,}334^2 \times 6{,}000}{\cdot 39 \times 72^5}$$

= 88·38 inches W.G.

$$\text{lbs. per sq. in.} = \frac{88 \cdot 38 \times 5 \cdot 2}{144}$$

= 3·2 lbs. sq. in.

Horsepower required

$$= \frac{\text{Quantity cub. ft. per sec.} \times \text{press. per sq. foot}}{550}$$

$$= \frac{3{,}334 \times 3 \cdot 8 \times 144}{550}$$

= 2,785 h.p.

a startling result.

On the Rand to-day there are several mines not working. Apparently the necessary capital to work them is not forthcoming. If this is so, it means that to-day mining has not sufficient inducement for the capitalist, and in support of this a critical examination of monthly and quarterly returns shows that some mines of lower grade have a very small margin of profit per ton. For these reasons technical men must "hasten slowly" with any schemes which involve increased operating expenses in addition to capital outlay My experience differs from some of the author's generalised remarks. A downcast current is *not* usually allowed to circulate through worked out places. In steeply inclined metalliferous mines it has long been the practice, where there is no steam pipe present, to make the main shaft the downcast. Air is thus taken to the bottom workings first, and returns through the worked out portions to the surface. In a flat mine if a main air current has to pass a worked out place, that place is generally packwalled off to preserve the air current if for no other reason. For sanitary reasons our Regulations No. 9 and No. 158 section 1 (*f*) require abandoned places to be adequately fenced off. Penalties are provided for polluting workings, and where employees work loyally with officials and bossboys are made to understand their responsibility, in spite of the language difficulty, one phase of underground sanitation is kept well in hand. There are mines on the Rand perfectly ventilated by natural means. Certain natural laws attached to ventilation must not be lost sight of, the most elementary being that heated air tends to rise, cold air to fall. A current will always take the shortest route, and these short routes must, for the benefit of the rest of the mine, be provided with stoppings or regulators. Bear in mind that airways must be of fair size to avoid loss by friction, and that when ventilating with air *not* under pressure ordinary small pipes are of no practical use. Any heat imparted to the air at the bottom of the upcast shaft improves the ventilation. Before the advent of fans, furnace ventilation was general colliery practice. Are there not circumstances in which it could be economically used to improve ventilation to-day? Take a vertical shaft and provide it with a few sprays or jets of water at the collar. If there is another outlet to the workings which can become an upcast, we find a strong downcast current produced. Why? First, the falling drops of water produce a downward current on the principle of the "trompe"; second, some of the atomised water is vaporised and taken up by the air, thus considerably reducing the temperature of the air and increasing its weight. So that this simple arrangement, costing very little, increases the downcast current in two ways, not to mention the beneficial effect of the cool moist air underground. So far my remarks have referred to general mining. Where long development drives are carried ahead we generally depend, for fresh air, on the exhaust from the machines. and the drive becomes a return airway. Many argue that this is insufficient; perhaps as many say that the air is quite good, and that there is no necessity for either a brattice or an air flume, and the subject seems worthy of further discussion. It is very wasteful to take compressed air direct from the main for ventilation only. If a dead end requires ventilation, and an air flume can be placed in position, a blower made of a small jet of compressed air in the throat of a venturi tube connected to the air flume at either end will use the costly compressed air to a better advantage and set up a fairly large circulation. Some power stations produce their induced draught with a jet of compressed air at the throat of a funnel, shaped after a venturi tube. To-day on most of our levels we have pipelines of water under pressure. In some cases this water will be cheaper than compressed air, and a jet of water can be used to produce the same effect. There is on record* a test where "with a jet $\frac{1}{200}$ sq. in. area, a pipe discharging $1\frac{2}{3}$ gallons of water per minute at 165 lbs. pressure per sq. in. a circulation of 850 cub. feet of air per minute was produced at the end of a level. . . ." Underground conditions to-day are better than ever before and will further improve. The thanks of the underground section of the industry are due to those who brought about the universal use of water for the laying of dust, and to the foresight of those who have placed the development of their mines so far ahead that blasting takes place only once in 24 hours, leaving natural ventilation about 15 hours in which to replace the atmosphere. It is in the power of all underground employees to contribute to the solution of the problem brought forward by the author—that of "obtaining healthier conditions"—by working loyally with the officials and keeping *all* the regulations at present in force instead of, as in many cases, trying to avoid them.

Mr. E. M. Weston *(Member)*: Mr. Blenkinsop appears to specialize in theories and proposals that take one's breath away. He paints a terrible picture of underground conditions which may or may not be true; but it is not true that we work in red hot air. His remedy has been tried before. In Australia they have thoroughly tested this method and others. Members might be interested to learn of one strange expedient adopted in alluvial mining. The deep alluvial mines work old river beds where deposits sometimes spread out over a width of half a mile. The gravel is somewhat like Rand banket, and is in some cases, even cemented with iron oxide and silica. Owing to overlying geological conditons CO_2 gas is very prevalent and, as in coal mines, it accumulates in the worked-out and collapsed areas and also in the looser gravels. When the barometer falls it flows out into the workings. Most mines were worked from one shaft and the whole mine was often rendered unworkable. In several mines an air-lock system was tried in order to remedy this. Air-tight double doors were placed below near the shaft and powerful Roots' blowers were used to force air at a pressure into the mine where the pressure was kept several inches above that of the atmosphere, and relief valves let any surplus air escape. The CO_2 gas was thus forced back into the goaf. Needless to say working conditions were very bad, but work was actively carried on on a large scale under these conditions. Regarding the application of the author's proposal here, I would merely say that it would be practically impossible to instal it, as there would be no room for pipes of the diameter required. If it were installed it would be useless within a week owing to damage from rock falls, trucks and blasting, etc., and any attempt at its maintenance would absorb any profits the mines made while it was there.

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*Encyclopædia Brittanica, 11th Ed., Vol. VI., page 588, line 56.

It is not required because, in conjunction with the system of fan ventilation in use, we have to-day a much better high pressure supply from the exhaust of rock drills just where required. This does supply cool air, which his proposed system emphatically would not do.

The meeting then terminated.

Contributions and Correspondence.

NITROUS FUMES FROM FIRING STICKS.

With reference to Dr. L. Heymann's reply to discussion on his paper on "Cheesa-stick" fumes,* I wish to deal with the last paragraph thereof, in which I am accused of issuing an 'unfounded' statement.

In the first place, Dr. Heymann states that he has challenged me to produce evidence for my statement that his method had been used in Western Australia before 1911. This is scarcely a correct representation of the facts. What happened was that a friend of his called on me, and I asked (on Dr. Heymann's behalf) to be shown Mr. Mann's pamphlet on blasting-gases. I discovered that I had mislaid my copy, and, although we ransacked the Seymour Library and the Mines Department Library, we were unable to find one. This happened many months ago, so Dr. Heymann had ample time to get a copy of the pamphlet from Australia or elsewhere before replying. As to the facts at issue, I now find from the pamphlet (kindly lent me by Messrs. Kynoch) that Mr. E. A. Mann estimated nitrous fumes (collected after blasting) by shaking for several hours with water, whereon "the amount of nitrogen was estimated by conversion into ammonia with a zinc-copper-couple and nesslerising," and calculated into NO_2.* This, I think, is sufficiently like the process described by Dr. Heymann to justify the remarks I made at the time the paper was read.

The method of reducing nitrates and nitrites to ammonia by means of zinc is of course extremely old, dating in fact from 1861 (see Fresenius, 1902, 397-399). When applied to nitrous fumes it has nothing like its original accuracy, because HNO_2 is lost when the zinc is first added, being carried away by the current of hydrogen (HNO_2 is a weak acid and therefore volatile): the correct procedure is to oxidise the alkaline solution of the fumes with $KMnO_4$ before acidifying it. It may be noted also that Gmelin and Martin, also Rose-Finkener, consider the results worthless unless the reduction is done in *alkaline* solution, whereas Dr. Heymann used sulphuric acid. I may add that Mr. E. A. Mann found it necessary, as a result of experiment, to add a correction of 25% to his results when dealing with small quantities of nitrous fumes.

In conclusion, I might suggest that it was Dr. Heymann's duty to deal with the serious criticism made by Mr. Crosse and myself that his method is wrong in principle, since what is wanted is not *total nitrogen* but *total acidity* (*free* HNO_2 and HNO_3).

JAMES MOIR.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RAPID SILVER ESTIMATION IN MILL SOLUTIONS.—"A rapid volumetric method for estimating Ag in cyanide mill solution. The cyanide solution is filtered slowly through a thin layer of Zn dust the Ag being completely precipitated. The precipitate is dissolved in HNO_3 and titrated with standard thiocyanate solution, using ferric alum indicator."—G. H. CLEVENGER, *Eng. Mining Journal*, 95, 892 3; *Chemical Abstracts*, Vol. 7, No. 15, Aug. 10, 1913, p. 2531. (J. G.)

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*This *Journal*, Sept. 1913, p. 118.

*"Investigations into the Composition of the Gases caused by Blasting in Mines," by E. A. Mann, 1911, p. 15: Parliamentary Paper, Western Australia.

COMPARISON OF REDWOOD'S WITH ENGLER'S VISCOSIMETER.—"Detailed comparison of Engler viscosimeter Nos. 2807 and 2809 (S. & R.) of the Reichsanstalt with Redwood viscosimeter No. 1303 (Baird & Tatlock) and Nos. 1404 and 1409 (Griffin) of the Nat. Phys. Lab., England, showed that the relation of Redwood degree (time for 50 cc. oil divided by time for 50 cc H_2O) to Engler degree at 20° differs slightly from that at 50° and that the formula given in their former publication expresses the mean relation between these temperatures. The ratio Redwood degree to Engler degree at 20° may differ by 11% from 1 and at 50° is about 6% greater than at 20°. This varying ratio at different temperatures is probably caused by temperature differences between different parts of the instruments; it has not yet been determined whether this influence is greater in the Engler or in the Redwood instruments."—W. MEISSNER, Phys. tech. Reichsanstalt, etc.; *Chemical Abstracts*, Vol. 7, No. 15, Aug. 10, 1913, p. 2697. (J. G.)

PRECIPITATION OF GOLD BY MANGANOUS SALTS.—"Some interesting occurrences of gold associated with manganese dioxide led to the suspicion that manganous salts, under certain conditions, might react with solutions of gold salts to bring about mutual precipitation; the gold in the free state and the manganese as hydrated manganese dioxide. A search for literature on such reactions was without avail, and experiments were undertaken with a view to ascertaining if such reaction is possible, and, if so, under what conditions it can take place. The reaction in question is in a sense the reverse of the series of reactions involved in the well known 'chlorination process' of extracting gold from its ores:—

$$MnO_2 + 4HCl = MnCl_2 + Cl_2 + 2H_2O$$

and

$$3Cl_2 + 2Au = 2AuCl_3$$

The reaction is doubtless much more complex than the summary equations, but it will be seen that a reversal of the series would lead to the formation of gold and manganese dioxide.

Gold-Chloride Solution.—Auric chloride solutions of varying concentrations were mixed with solutions of manganous chloride with concentration ranging from 0·5/*N* up to saturation, but no reaction was detected even when the mixture was boiled for several minutes. The addition of a small amount of alkali to the mixtures, in the cold, caused an immediate precipitation of a dark brown mass resembling manganese dioxide in the hydrated form commonly precipitated. The precipitate was collected, carefully washed to free it from the gold solution, and then treated with standard oxalic acid, containing a small amount of sulphuric acid. The solution thus obtained was divided into equal portions: one was analyzed for manganese and the other titrated for loss of oxalic acid. The results showed that for every equivalent of manganese an equivalent of oxygen had been taken up by the oxalic acid, proving the precipitate to be manganese dioxide. In a check experiment the precipitate was taken into solution with the standard oxalic-sulphuric acid mixture, which was titrated for loss of acid. The manganese in solution was then reprecipitated as hydrated manganese dioxide by a standard method, and the precipitate was again treated with the standard acid mixture. The same loss of acid as before shewed the original precipitate to be manganese dioxide

Gold was left by the solution of oxalic acid and was readily recognised as such.

Gold chloride solutions are notably acid in their reaction, due probably to hydrolysis in part, and in part to the ionization of an addition-product with water. Hittorf and Salkowsky showed by electrolytic experiments that gold chloride solution is ionized as follows:—

$$\overset{+}{2H} + \overset{-}{Au}Cl_3\overset{-}{O}$$

Solutions of gold chloride show marked acid properties toward indicators, and apparently the acidity thus developed is sufficient to suppress the reaction,

$$2Au^{+++} + 3Mn^{++} = 2Au + 3Mn^{++++}$$

If one postulates the intermediate formation of $MnCl_4$ in minimal amounts, one is justified in assuming that the salt would be largely hydrolyzed, since tetravalent manganese is a very weak base. This hydrolysis would be suppressed by the presence of acids, but on reducing the acidity, hydrolysis might become effective, forming the very slightly soluble hydrated manganese dioxide, and with this removed from the equilibrium system by precipitation, the reaction might proceed until concentration were diminished to equilibrium conditions.

It is of interest to note that the precipitation began long before the solutions were entirely neutralized as shown by indicator tests. This was shown in another way by placing a crystal of iceland spar in the mixture of gold and manganous chlorides in solution. A slight effervescence occurred, and after a few hours the crystal was covered with a a brown coat of manganese dioxide in which flakes of gold were plainly visible. The precipitation of gold was practically complete when an excess of $MnCl_2$ was employed, though this solution had at least the acidity of saturated carbonic acid.

An interesting reaction of similar nature was found to take place between auric chloride solution and manganous carbonate. The precipitated carbonate (doubtless amorphous, in part, at least) reacts at once and is turned to the dark brown hydrated dioxide. The crystallized carbonate, that is, the mineral rhodochrosite, reacts slowly, but, after a day, a crystal was coated with dark brown and flakes of gold were plainly discernible.

Summary.—If we suppose manganese tetrachloride to be an intermediate product, present only in very minute traces, we may find in it some basis of explanation. Suppose

$$2AuCl_3 + 3MnCl_2 \rightleftarrows 2Au + 3MnCl_4$$

Since tetravalent manganese is an exceedingly weak base it should be little ionized, but it may be subject to two other sorts of dissociation, namely:—

1. A molecular dissociation,

$$MnCl_4 = MnCl_2 + Cl_2$$

2. Hydrolytic dissociation,

$$MnCl_4 + 4H_2O = Mn(OH)_4 + 4HCl$$

Theoretically we should have the hydrolysis constant

$$\frac{[Mn(OH)_4] \times [HCl]^4}{[MnCl_4]}$$

assuming hydrolysis according to the above equation. It is obvious that the presence of acid would tend to cut down the hydrolysis and allow the tetrachloride to become sufficiently concentrated to make its molecular dissociation appreciable—the greater the concentration of acid the greater this tendency—and we may, in this way, easily obtain conditions under which free chlorine is liberated, as in the chlorination process.

If, however, the acid concentration is reduced below a certain figure the concentration of the tetrahydroxide will reach saturation. Any further reduction of acidity must result in the precipitation of the hydrated dioxide. It will be seen that such an adjustment may be very delicate indeed—and this seems to be supported by the lack of success in preparing the tetrachloride subject, as it would be, to both hydrolytic and molecular dissociation."—A. D. Brokaw, *Mining and Scientific Press*; from *American Chemical Society*, July 26, 1913, p. 149. (A. McA. J.)

Removal of Fusions in Alkaline Carbonates from the Crucible.—"Dropping a little powdered KNO_3 into the crucible while the contents are still liquid, renders the mass porous on cooling. Solution in water is then complete in 20 min. or less."—R. Howden, *Chem. News*, *107*, 232; *Chemical Abstracts*, Vol. 7, No. 15, Aug. 10, 1913, p. 2530. (J. G.)

Cyanide from Sugar-Mill Residues.—"Germany has made the greatest advancement in this regard. Of a total production of 400,000 tons of beet molasses in this country, about 55% is desaccharified for sugar production, about 30% is used as cattle food, about 10% is fermented into alcohol, and the remaining 5% is utilised in miscellaneous ways. Among the last may be mentioned the use of molasses for manufacturing dye stuffs, shoe blacking, yeast, moulds and briquettes, and numerous other commodities.

The residues from the desaccharification factories, best known under its German name of molasses *schlempe*, has been a subject of special study from the standpoint of utilization. From 1,000 kg. of best molasses are obtained about 350 kg. of concentrated *schlempe*, containing about 30% of mineral matter, mostly potash salts, some 20% or more of nitrogenous substances, and a remainder of acids, gums, caramelization products, and other organic residues.

Molasses *schlempe* contains 12% to 15% of potassium and 4% or more of nitrogen, and its conversion into derivatives of these elements constitutes at present the chief method of utilization. The *schlempe* is first heated in retorts, by which means it is decomposed into a mixture of volatile products consisting of carbon dioxide, carbon monoxide, hydrogen, nitrogen, methane, ammonia, methylamine, methyl alcohol, water, and other substances. The volatile decomposition products escape from the retorts at a temperature of about 400° C. and are led through a system of tubes heated to a temperature of about 1,000° C. The effect of this heating is to convert the volatile nitrogenous compounds into ammonium cyanide, the gas after cyanization, containing about 7% NH_3 and 7% HCN. After leaving the hot tubes, the gases, which are always kept under reduced pressure, are cooled, freed from tar, and then washed over sulphuric acid to break up the ammonium cyanide, the ammonium sulphate, which is formed, being recovered. The hydrocyanic acid is then absorbed in water, and the residue of combustible gases is led back to the furnaces for heating the retorts. The hydrocyanic acid is then distilled, and absorbed in sodium hydroxide; the solution of the latter, after evaporating and crystallizing yields solid sodium cyanide

By the above method, about three-fourths of the nitrogen in molasses *schlempe* is recovered as ammonium sulphate and sodium cyanide, the remaining one-fourth escaping as gaseous nitrogen. A small amount of pyridine is also obtained by this process, in connection with the ammonia. The residue of mineral matter in the retorts, after distilling the *schlempe*, is worked up into potash, of which some 15,000 tons is made annually in Germany from this source.

Two factories in Germany produce annually, by the process of distillation described, about 5,000 tons of ammonium sulphate and 5,000 tons of sodium cyanide, with a commercial value of about $1,750,000. The sodium cyanide thus manufactured is nearly all exported to the Transvaal, where it is used for extracting gold by the well known cyanide process.

Chemists in Germany are making further efforts toward improving the utilization of molasses *schlempe*. By the present methods of distillation, about one-fourth of the nitrogen is lost, and this is wasteful from the standpoint of highest economy. It has been felt by some chemists that efforts should be made toward removing some of the valuable organic constituents of the *schlempe* before making the distillation."—C. A. Browne, *Mining and Scientific Press*; from *Columbia School of Mines Quarterly*, Aug. 2, 1913, p. 186. (A. McA. J.)

Uses of Cast Silicon.—"According to *Metallurgical and Chemical Engineering*, xi, 103, silicon is used in the form of pipes for the conveyance of acid gases at a high temperature from stills to condensers, also in the construction of the condensing batteries themselves, proving more efficient than stoneware owing to its high thermal conductivity and ability to withstand sudden changes of temperature. Silicon pipes are also used for the transportation of hot liquid sulphuric and nitric acids, but it cannot be used for hydrochloric acid unless the discoloration resulting from the slight solubility is immaterial. It is used as a lining in centrifugal pumps, acid valves, and pipes for the elevation of corrosive liquids by compressed air, and for the construction of ploughs in ore roasters it is said to have proved efficient. Cast silicon is also being employed in the form of shallow pans and pots for the concentration of zinc chloride solutions, supplanting the enamelled stoneware vessels which are rapidly destroyed. Cast silicon ware is said to be produced in practically all shapes required by the chemical industries, such as, for example, pipes, evaporating vessels, receivers, tiles, alembics, crucibles, and pump parts."—*Journal of Industrial and Engineering Chemistry*, v., No. 3. —*Chemical News*, August 8, 1913, p. 72. (J. M.)

Tantalum as a Substitute for Platinum.—"The rise in the price of platinum which has now been going on for the last ten years or so, has brought tantalum to the front for crucibles and dishes, and other chemical apparatus which has to stand high temperatures. Tantalum has its faults, however, as well as platinum, but if the two chief ones are borne in mind, viz., that it cannot resist hydrofluoric acid, and that it oxidises readily if heated with ready access of air, something may be done in the way of using tantalum for making crucibles and dishes. Tantalum seems also to have a field in electrolytic work as a substitute for platinum electrodes. Tantalum cathodes answer perfectly, but tantalum anodes, by virtue of the affinity for oxygen of the metal which has already been alluded to, soon become coated with dark blue tantalic oxide. This being a bad conductor soon stops the current. The only remedy in this case is to cover the anode with a thin coating of platinum. Even then money is saved, as the coating takes much less platinum than is needed for an anode all of platinum. The use of tantalum for cathodes has many advantages as compared with platinum. It does not alloy with zinc or cadmium as platinum does. Hence no coppering or silvering of the electrode is required, and the insolubility of tantalum in aqua regia makes it particularly suitable for the regeneration of the metals from precipitates containing gold and platinum. Tantalum is lighter than platinum, so that a less weight of it is required for electrodes of the same mass and surface, and its price is less than half that of platinum. Moreover, it is much stronger than platinum, so that thinner electrodes of it can be used to resist dangers due to rough handling."—*Chemical Engineering and the Works Chemist*, Aug. 1913, p. 243. (J. W.)

ALCOHOL AS FUEL IN SOUTH AFRICA.—"According to the London correspondent of the *Birmingham Daily Post*, considerable attention is being given by public authorities and influential capitalists in South Africa to the problem of fuel alcohol. It is recognised that the cost of transport of petrol for long distances into the interior restricts the use petrol-driven vehicles and agricultural machinery. In some territories, especially those scourged by the tsetse fly, East Coast cattle fever, and the like, mechanical power is an imperative necessity if development is to proceed to its fullest possible extent. It is also being realised that the future lies with fuel-alcohol of local manufacture, and overtures have already been made, directly or indirectly, for Government facilities to undertake this new industry. A difficulty which does not present itself in Europe arises from the fact that the standard of impotability recognised by white people does not suffice to render the denatured alcohol impossible as a beverage for the native races, and the powerful interests which concern themselves with the welfare of the natives are at work to prevent the removal of the present Excise restrictions until it has been shown that no danger to their constituents would ensue."—*The Oil and Colour Trades Journal* and *Birmingham Daily Post*, Aug. 23, 1913, p. 664. (J. W.)

VOLUMETRIC DETERMINATION OF CHROMIUM IN PRESENCE OF IRON.—"The solution is treated with an excess of bromine water, then made strongly alkaline with potassium hydroxide, heated, with frequent agitation, for half-an-hour on the water-bath, cooled, made up to 250 cc., and filtered through a dry filter. 100 cc. of the filtrate are neutralised by adding dilute sulphuric acid until the colour of bromine becomes permanent, 20 cc. of 30% potassium bisulphate solution are added, and the mixture is boiled vigorously for 5 min. Chromate is determined iodometrically in the cooled liquid, but should any odour of bromine be observed after boiling (thus indicating undecomposed bromate), a further 10 cc. of bisulphate solution must be added and the boiling repeated before titration. The residual ferric hydroxide is dissolved and titrated in the usual way."—A. KURTENACKER, *Journal Society of Chemical Industry*, July 15, 1913, p. 713. (F. W. W.)

A REACTION OF IODINE-IONS.—"The production of the reddish-brown substance formed in the Nessler test for ammonia may be used for the detection of iodides. The reagent is prepared by dissolving 10 gm. of mercuric nitrate in 50 cc. of water acidulated with 5 cc of nitric acid, and then adding 60 cc. of concentrated ammonia solution. 1 cc. or more of this solution is added to the solution under examination. The reaction is more delicate than any of those hitherto used for the detection of iodides. Chlorides give a white precipitate which does not interfere with the test. Bromides may give a white or yellow precipitate according to the concentration, but if the mixture of the solution and the reagent be heated, the yellowish or brownish-red colour due to iodide becomes apparent."—R. CIUSA and A. TERNI, *Gas. Chim. Ital.*, 1913, *43*, II., 86; *Journal Society Chemical Industry*, Aug. 30, 1913, p. 843. (J. A. W.)

SEPARATION OF IRON FROM CHROMIUM,—"The ignited oxides are gradually heated from 200° C. to 650 C., and then maintained at the higher temperature for 3—4 hr. in a very slow stream of chlorine carrying sulphur chloride vapour from the liquid kept at 30°—40° C. If the chromium is likely to be more than 40% of the total metal, the oxides are first mixed with their own weight of ammonium sulphate, and the mixture is gradually heated in a stream of chlorine only till all the sulphate is decomposed and volatilised; the porous mass remaining is then exposed to the mixed chlorine and sulphur chloride. The boat, and the portions of the tube containing any volatilised chlorides, are then treated with cold water, which dissolves the ferric chloride completely, and leaves the chromic chloride undissolved. From the weight of chloride that of the chromium sesquioxide is calculated, whilst the iron is precipitated by ammonia from the solution of ferric chloride."—A. DESHAYES, *Journal Society of Chemical Industry*, July 15, p. 713. (F. W. W.)

DETECTION OF HYDROCYANIC ACID.*—"The test proposed depends on the coloration or precipitate which is obtained when copper acetate, potassium cyanide, and benzidine acetate are mixed together in solution. The reaction is also given by other substances, and is only characteristic of hydrocyanic acid under the following conditions. After the removal of metals from the solution under examination, the latter is boiled with sodium carbonate, filtered, and carbon dioxide is passed into the filtrate; this is then added to a mixture consisting of one drop of a 3% copper acetate solution, 1 cc. of a 10% disodium phosphate solution, and four drops of a saturated benzidine acetate solution. The test will detect 0·007 mg. of hydrocyanic acid in 10 cc. of solution."—C. PERTUSI and E. GASTALDI, *Chem. Zeit.*, 1913, *37*, 609-610; from *Journal of the Chemical Society*, No. dcix, July 1913, p. 637, (F. W. W.)

THE ESTIMATION OF MERCURY AS METAL BY THE DRY METHOD.—"Hg compounds when heated with CaO are decomposed according to the equation, $2HgX + 2CaO = 2CaX + 2Hg + 2O$. The method is applicable to ores and minerals but is experimentally difficult. Penfield's apparatus (*Am. J. Sci.*, [3] 48, 31) for the determination of H_2O in minerals obviates the latter objection. Bulbs are blown at the closed end, and near the open end, of a glass tubing 20 cm × 5 mm. The cleaned and dried tube is filled by means of a drawn-out glass funnel, with a suitable quantity of the powdered sample, followed by mixed iron filings and CaO to 8 cm. of the tube. An asbestos plug is then placed midway between the bulbs, and the open end of the tube drawn out to a fine capillary. The terminal bulb is heated in an iron pipe, and finally directly, until all Hg is removed and condensed in the second cooled bulb. This receiving bulb is then drawn off, H_2O;

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*See also Moir, this *Journal*, Vol. X., April, 1910, p. 342 Moir, Vol. XI., Oct., 1910, p. 155; and Vol. XIV., Sept., 1913, p. 152.

removed by a current of dry air, and the bulb and the contents weighed. The Hg is then poured out, and completely removed by air drawn through the heated tube, and the latter weighed empty. The method is applicable to sulphides, if $PbCrO_4$ is added. Five-tenths to 1·0 gm. of the sample should be heated with 2·5 to 3·0 gm. of $PbCrO_4$ and the Fe and CaO mixture."—A. C. Cumming and J. Macleod, *Jour. Chem. Soc.*, *103*, 513-6; *Chemical Abstracts*, Vol. 7, No. 14, July 20, 1913, p. 2366. (J. G.)

METALLURGY.

The Preservation of Iron.—"Iron plates coated with various liquid vehicles and paints, were exposed to the action of air and water respectively, for 9 to 10 months. The results showed that (1) the surface of a paint film is not always a true indication of the extent of corrosion; (2) a little paraffin wax in the paint reduces corrosion in air, but appears to assist corrosion in water; (3) fine pigment (rouge) is more protective than coarse, but good commercial grades are finely enough ground for all practical purposes; (4) double coats of paint may be less protective than single coats; (5) slightly rusted plates, if quite dry when painted, are less liable to corrode than plates freed from rust before painting."—J. N. Friend, *Journal of the Society of Chemical Industry*, June 30, 1913, p. 660. (J. A. W.)

The Corrosion of Iron and Steel.—"It is now six years since the electrolytic theory in its developed form was offered as a basis for the explanation of the corrosion of iron and steel, and it seems worth while at this time to enquire into the accuracy of some of the conclusions to which it has led, as viewed in the light of present experience.

In the first place we have found that the factors controlling the rapidity or extent of corrosion are by no means so simple as they were at first thought to be. Many conditions which were considered of little or no importance have been found to exert a profound influence upon the reactions involved. For example, samples of iron and steel which exhibit marked differences in corrosion exposed in the normal condition in which they come from the mill, fail to show any difference upon exposure, when they are first planed to a uniform surface. Apparently the mechanical strain to which the samples are subjected in the planer, masks or neutralises the difference in corrosion inherent in the normal material. It is not surprising, therefore, that many conflicting results have been obtained and published from the investigators now interested in this work. Only those tests which have been carried on under identical conditions of surface finish, temperature, access of oxygen and moisture, general atmospheric conditions, etc., should be given any weight, and even when the greatest care is taken, generalisations must be drawn with caution.

One of the conclusions reached by a consideration of the electrolytic theory of corrosion which has proved misleading, is that homogeneity in the material insures protection, while heterogeneity leads to rapid attack. While this is a corollary which may be logically drawn from the electrolytic theory and is doubtless in itself true, there are evidently other factors which superimpose themselves upon those due to differences in structure, producing a final effect contrary to that predicted. The iron of the old chain bridge at Newburyport, Mass., has withstood corrosion in a truly remarkable manner for the last 98 years; and yet it is conspicuous for its heterogeneous structure. Large areas of perfectly pure iron, free from both carbon and slag, are mixed up with areas showing at least two kinds of slag, and very high carbon; yet all withstand atmospheric corrosion. On the other hand Burgess has shown that iron free from all contaminating elements which could segregate or produce a lack of uniformity, does not withstand rusting so well as the same iron to which has been added a little manganese or copper or nickel. This behaviour is observed also in the case of the so-called pure irons made in an open hearth furnace, and which are relatively free from carbon, manganese, sulphur, and other constituents prone to segregation, which have come to the writer's notice. While theoretically a very pure iron should withstand rust, there are apparently some factors present which more than offset any advantage inherent in purity. Obviously conditions affecting the surface of the material so soon as rusting has started are important causes which have largely been overlooked and which demand more thorough investigation.

The most important advance in this field made in recent years, is a knowledge of the effect of the addition of small amounts of copper to normal open hearth or Bessemer steel. This result is most strikingly brought out in the paper of D. M. Buck published in full in a recent issue of the *Journal of Industrial and Engineering Chemistry*. The present writer has within a week inspected with much interest the test roofs here described and can testify to the remarkable effect shown. While panels of Bessemer and open hearth steel containing the ordinary amounts of metalloids have entirely failed, the corresponding panels made from these same heats of steel but to which a small amount of copper was added are in a remarkable state of preservation. A panel of the so-called pure iron very low in metalloids, but containing one-half the amount of copper carried by the less pure steels is less attacked than those steels containing no copper, but is much less resistant than are those to which 0·2% of copper was added. Had copper been omitted entirely from this iron it would doubtless have succumbed even earlier. While these tests do not show that any steel, however poorly made, will, with the addition of copper, withstand atmospheric corrosion, they prove that it is the copper, and not the absence of manganese and the other 'impurities' which is the controlling factor. Several hypotheses have suggested themselves as explaining this marked effect of copper in causing steel to resist atmospheric corrosion, but as yet none is sufficiently tangible to afford a working theory.

Rapid progress has been made in acquiring that knowledge of the relation of pigments and finished paints to corrosion which is necessary to a better protection of iron and steel. Predictions founded on theory that a basic paint, or one containing a chromate pigment would inhibit rusting, while one made up from lamp black or graphite would accelerate rusting, have, in the main, been found correct. The effect of the pigment upon the character of the oil film making up the paint, however, has shown itself also to be very important. Many basic pigments, such as basic lead carbonate or zinc oxide, which in themselves inhibit, do not withstand the weather; lamp black and graphite, on the other hand, make a very impervious and highly resistant paint film. The logical conclusion in protecting iron is, therefore, to use a basic priming coat, a second coat of a mixture of a basic pigment and a little lamp black, and when well dried out to apply a lamp black or graphite finishing coat. Experience has shown also the importance of brushing the paint well on to the iron; a good paint may fail on account of poor application.

Careful tests with galvanised work show that an even coating of zinc is the all-important factor. The common practice of clean wiping galvanised ware is fatal to durability, since the protecting layer is not metallic zinc, but a thin deposit of a zinc-iron alloy. While therefore much has been accomplished in the way of making a more resistant base, there is still a necessity for a uniform substantial coating of spelter over the surface."—William H. Walker, *Journal of the Society of Chemical Industry*, June 16, 1913, p. 584. (F. W. W.)

Concentration of Telluride Ores of Gold.—"The ores consist of a hard silicious gangue with sylvanite which is present in very soft blade-like monoclinic crystals. Gravity separations have been unsuccessful because of the loss due to the floating away of the fine telluride particles. The proposed method is to crush the ore dry to 60-mesh or finer, when 50% of the sylvanite can be recovered by flotation. If the tailings are then treated on a Wilfley table, a further 20% to 25% is obtained. By regrinding the tailings from the table to 150-mesh, 66% of the remaining gold is recoverable by the ordinary cyanide treatment. Tables are given showing that about 80% of the gold, and an average of about 70% of the silver were recovered by this method, on an experimental scale."—H. E. Wood, *Journal of the Society of Chemical Industry* (from *Eng. and Min. J.* 1913, *95*, 885-886), June 16, 1913, p. 606. (F. W. W.)

A German Criticism of American Metallurgy.—"Dr. Heberlein, who recently visited a number of U.S.A. and Mexican lead, zinc and copper works, as a result of his observations read an interesting paper at the last meeting of the Association of German Metal Smelters. He was particularly struck by the fact that the smelters are, as yet, unprovided with coking installations, and that the coking works have not yet adopted efficient means for the recovery of by-products. He highly commends the small electric locomotives able to proceed by the sharpest curves into the narrowest corners of the works. He illustrated by photographs the working of American circular pumps for liquid lead, and be particularly mentioned the thawing sheds, where ores frozen during transport in railway trucks are thawed in order to facilitate unloading. He praises the high value placed in America upon well-equipped works' laboratories, as well as the careful composition and ingenious handling of the charges. After alluding to attempts made to neutralise the noxious effects of sulphurous acid and the recovery of potassium salts from phonolites, he shortly discussed American labour efficiency, which, notwithstanding much higher wages, he does not consider superior to the German standard. As a result of all he had observed, he stated his belief that the Americans are in advance of Europe only in the metallurgy of copper and in the treatment of poor lead ores."—*London Mining Journal*, August 9, 1913, p. 764. (A. R.)

Alloy of Zinc as Precipitant.—"The preparation and use of an alloy of zinc, lead, magnesium, aluminium, or other similar combination, is the basis of several patents granted to Mr. Charles W. Merrill, of Berkeley, Cal. The inventor claims that such an alloy is a more efficient precipitant than zinc alone. In order to prepare the alloy in a finely divided state, it is mixed with an abrasive before cooling, whereupon it can be readily reduced to powder by any form of comminuting apparatus. The use of the abrasive is essential to overcome the natural malleability of the metals which ordinarily prevents their comminution.

Another form of application of the precipitant is to make use of it as a liner and balls for a small tube mill, through which the solution passes. The resulting attrition is sufficient to liberate the metal and allow it to flow with the solution to the collecting press. The alloy has a galvanic action which makes it a more efficient precipitant than one metal alone. (1,063,568-69-70, June 3, 1913)."—C. W. Merrill, *Metallurgical and Chemical Engineering*, Vol. xi, No. 7, July, 1913, p. 414. (J. A. W.)

Applications of Ductile Tungsten.—"Less than 10 years ago tungsten was universally conceded to be a very brittle metal. Since the introduction of ductile tungsten (C. G. Fink, *Trans. Am. Electrochem. Soc.*, xvii., 229; W. D. Coolidge, *Trans. Am. Inst. Elec. Eng.*, xxix., 961), however, large quantities of drawn wire, flexible and strong, are being daily produced for the manufacture of incandescent lamps.

We have studied the physical and chemical properties of this new tungsten, and have obtained a number of very interesting results.

The ductile metal is practically insoluble in all of the common acids (W. E. Ruder, *Journ. Am. Chem. Soc.*, 1912, 387), its melting point is higher than that of any other metal (I. Langmuir, *Trans. Am. Electrochem. Soc.*, xx., 237), its tensile strength exceeds that of iron and nickel, it is non-magnetic, it can be drawn down to smaller sizes than any other metal, and its specific gravity is 70% higher than that of lead.

It was natural that a metal with such striking properties as these should soon find applications other than that for incandescent lamps.

Table of Physical and Chemical Properties of Ductile Tungsten.—Density, 19·3 to 21·4.

Tensile strength, 322 to 427 kg. per sq. mm.

Young's modulus of elasticity, 42,200 kgm. per sq. mm. (steel 20,000); *i.e.*, twice as elastic as steel.

Melting-point, 3177 (Langmuir) 3,100 ± 60 (v. Pirani and Meyer).

Boiling-point, 3,700° (?).

Thermal conductivity, 0·35 gm. cal. per cm. per sec., per 1° (Pt. 0·166) (W. D. Coolidge, *Trans. Am. Inst. Elec. Eng.*, xxxi., 870; *Journ. Ind. and Eng. and Eng. Chem.*, iv., 2).

Expansion co-efficient, $4{\cdot}3 \times 10^{-6}$ (Pt. $8{\cdot}8 \times 10^{-6}$).

Specific heat, 0·0358 (Weiss).

Resistivity (25°), hard, 6·2 microhms per cc.; annealed, 5·0 microhms per cc.

Temperature co-efficient of resistance, 0·0051 (0°—170°).

Hardness, 4·5 to 8·0 (Mohs scale).

Insoluble in HCl, H_2SO_4, HNO_3, HF, NaOH, KOH (aq.) $K_2Cr_2O_7 + H_2SO_4$ (C. G. Fink, *Trans. Am. Electro-Chem. Soc.*, xvii., 229; W. D. Coolidge, *Trans. Am. Elec. Inst. Eng.*, xxix., 961). Soluble in mixtures of HF and HNO_3, and in fused nitrates, nitrites, and peroxides.

The boiling-point of the metal has not yet been determined.

The Young's Modulus of Elasticity are determined with a wire 0·00648 cm. in dia. and 784·86 cm. long. The smallest weight (P) was 250 and the largest 1,125 gm. The elastic elongation was 0·35 cm. for the smallest weight and 1·65 cm. for the largest. The average for five different weights was 42,200.

The hardness values were determined with the scleroscope and the values translated into the Mohs scale.

Electrical Contacts.—Wrought tungsten has been substituted with success for platinum and platinum-iridium as contact points in spark coils, voltage regulators, telegraph relays, etc. (W. D. Coolidge, *Trans. Am. Inst. Elec. Eng.*, xxxi., 870; *Journ. Ind. and Eng. Chem.*, iv., 2). The service far exceeds that for platinum and platinum-iridium contacts due to the greater hardness, higher heat conductivity, and lower vapour pressure of tungsten as compared with platinum.

Tungsten Furnaces.—These furnaces are of two types. The type recently described by Winne and Dantsizen (*Trans. Am. Electro-Chem. Soc.*, xx., 287) consists of an alundum tube wound with tungsten (or molybdenum) wire. To prevent oxidation the tube is encased in an air-tight box with an inlet and outlet for hydrogen. This furnace is admirably well suited for laboratory experiments. Temperatures of 1,600°—1,800° C. can be easily maintained for hours, whereas platinum at these temperatures would rapidly disintegrate.

A second type of tungsten furnace (U. S. Pat. 1,006,620) is constructed on lines similar to those of the Arsem vacuum furnace. A tungsten metal tube takes the place of the helical carbon resister. The tube is surrounded by a screen and the whole enclosed in an air-tight compartment almost identical to that used by Arsem. The compartment is either evacuated or a small quantity of gas, such as hydrogen, is introduced. This furnace lends itself admirably for the study of reactions at very high temperatures, such as the production of artificial gems.

Tungsten Gauze.—We have used this gauze successfully for separating solids from acid liquors. We performed these experiments on a laboratory scale. However, this gauze could well be used on a commercial scale. For example, for the removal of sludge from copper refining baths, and for centrifugal apparatus, in general, whenever acid liquids or acid gases are dealt with.

Furthermore, it might be used in apparatus such as described by Cottrell (*Journ. Ind. and Eng. Chem.*, iii., 542) for the removal of sulphuric mist from gases. The Cottrell electrodes consist of three concentric cylindrical screens of iron wire, the inner and outer ones acting as discharge electrodes, while the intermediate screen and the outer leaded glass containing vessel act as collecting electrodes from which the deposited acid drains into a leaden pan below. Tungsten gauze is not attacked by sulphuric acid and would consequently give a much longer life than iron gauze.

Wrought Tungsten Targets for Röntgen Tubes.—This application has proved to be one of the most interesting. Tungsten is very well suited for targets or anticathodes, and the realms of application and efficiency of the Röntgen tube have been thereby greatly increased.

As has been shown by Coolidge, the high specific gravity, high melting-point, high heat conductivity, and low vapour pressure make tungsten a far more efficient target than any other metal.

Thermocouples.—We are investigating the thermoelectric properties of the couple, tungsten molybdenum. The electromotive force increases with the temperature up to about 540°, then decreases and passes through zero millivolt at about 1,300°. We have found this couple very convenient for high temperature measurements in the tungsten-hydrogen furnace.

Standard Weights.—A material suitable for standard weights must preferably be hard yet plastic, it must not be easily scratched nor marred, but still not be so hard that it will chip or break; furthermore, it must withstand the action of the atmosphere and finally it must be small in bulk. Now wrought tungsten can be made so hard that it will readily scratch glass and still be ductile; furthermore, the density is high (19·3 to 21·4) and it is unaffected by the atmosphere. Tungsten weights remain wonderfully constant.

Tungsten Cells.—We have taken up the study of the electrochemical behaviour of tungsten and have made up a series of cells and combinations. All measurements were made at 25° and compared with the calomel electrode as standard. Our readings for the cell tungsten, aqueous sodium hydrate, potassium chloride, calomel, mercury, are:—

5N NaOH, 0·68; 2N, 0·62; N, 0·57; $^{1}/_{5}$N, 0·525; $^{1}/_{10}$N, 0·50; $^{1}/_{20}$N, 0·48; $^{1}/_{40}$N, 0·455; $^{1}/_{160}$N, 0·445; $^{1}/_{320}$N, 0·380; and 0·0N, 0·06 volt. In the last cell the tungsten rod was immersed in distilled water.

The addition of small amounts of impurities to the tungsten metal causes the tungsten—sodium hydrate electrode to assume E.M.F. values that approach that of zinc in zinc sulphate.

The values for potassium hydrate are similar to those for sodium hydrate. The E.M.F. of the cell $Hg—Hg_2W_3O_{11} + Na_2WO_4$ solid—Na_2WO_4 sat. soln.—solid Na_2WO_4—W was found to be 0·505 volt and promises to be a good standard cell.

Miscellaneous Applications.—Besides the applications of tungsten cited above, many others have been but partly worked out and others merely suggested.

Owing to its chemical stability the finest sizes of wire down to 0 0002″ or 0·005 mm. in dia. are well adapted for galvanometer suspensions, and for cross hairs in telescopes. It has also been suggested to use these fine wires in surgical operations in place of the coarser gold and silver wires. A further suggestion is the use of the wire in musical instruments. The tensile strength and elasticity of tungsten wire are exceptionally high (see table).

It could be used to advantage in climates where steel is readily corroded.

We are investigating the formation of hydrocyanic acid gas by passing over heated tungsten wire mixtures of nitrogen and acetylene or methane. (Compare Berthelot and Lipinski, *Zeit. Elektrochemie*, xvii., 287).

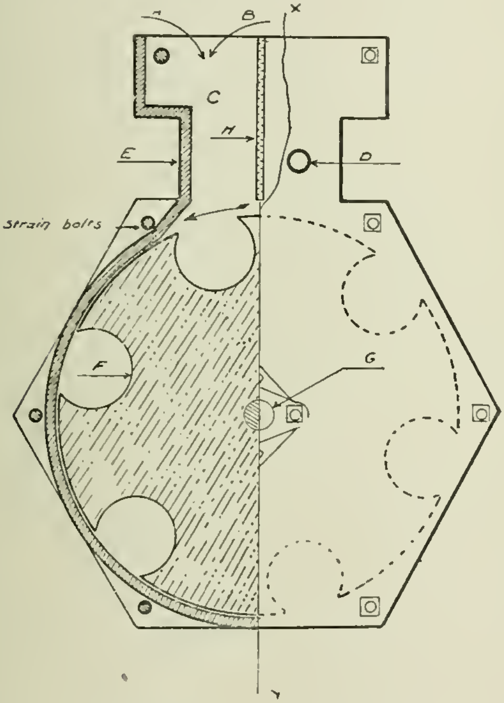
We are making acid proof dishes and tubes out of tungsten. Furthermore, tungsten wire recommends itself as a unit resistance, since it can be made absolutely pure, can be easily duplicated, and is not corroded.

Since tungsten is non-magnetic and elastic it is being tried in electrical meters, replacing the phosphor-bronze springs.

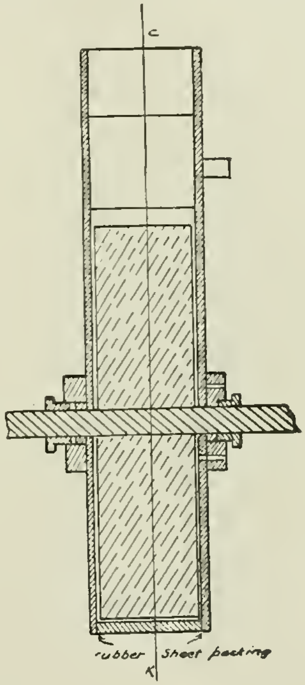
Similarly watch springs could be made which would never become magnetised. Finally we might mention: tungsten pen points, tungsten drawing dies, tungsten knife blades, tungsten reinforced asbestos curtains, and fire-proof coverings, etc."—C. G. Fink, *Chemical News* (VIII. Internat. Congress Appl. Chem., New York, 1912), June 13, 1913, p. 282. (J. M.)

Zinc Dust Precipitation Equipment.—"Many of the departments of modern cyaniding have become practically standardized in the course of widespread construction of mills in the past five years, but among them all there exists a considerable divergence of opinion on the method of feeding zinc-dust to the solution pumps for precipitation according to the Merrill system.

When this method of zinc-dust precipitation was first put on the market, the accepted form of zinc-dust feeder was a slowly moving continuous belt, which carried the dust and which was operated by a series of pulleys and cords attached to floats in the solution tanks. This feeder spilled the dust, in the required amount, into a shallow cone, which was filled by a small steadily flowing stream of solution, and the whole was emulsified by a light jet of air. The emulsion flowed from the upper edge of the cone to the suction of the solution pump supplying the zinc presses. While apparently simple, this method was not altogether satisfactory because of slippage of pulleys, spilling of zinc, and the inevitable clotting and incrustation of zinc-dust, wherever moisture reached it, before entering the pump suction. Furthermore, it was not a positive method of feed, and therefore not absolutely under control.



Half Section. Elevation.



Section on x-y.

Various other zinc-dust feeders have been described, for example, those used at Zambona in Sonora, and Mocorito in Sinaloa, differing from each other and from the standard type above mentioned. Apparently realizing the defects of the earlier apparatus, the Merrill company has supplied a new device, which relies upon positive methods of feed, and, with some slight changes, it has been found satisfactory at the plant of the Tigre Mining Co., S. A., in Sonora.

Essentially, it is a covered hopper which holds the zinc-dust and discharges it through a slot in its lower side into a horizontal tube. This tube is provided with a central auger or screw, which, revolving slowly, forces out the zinc-dust at the end of the tube at a given rate of discharge. The auger is keyed to a gear driven by a worm that is keyed in its turn to a shaft, upon the other end of which is fixed a large friction disc. Motion is imparted to this disc by a friction roll carried on a small shaft, which is belted to a line shaft. A considerable variation in speed is made possible through shifting the friction roller across the face of the disc.

The only change found necessary in the machine as supplied to the Tigre company was a replacement of the auger which forces out the zinc-dust, and the tube containing it, as the original tube was too small for the requirements of precipitation, if the friction disc was to be run at any reasonable rate of speed. The tube as supplied had an available diameter of about 1 in., and was changed for one of 1$\frac{3}{4}$ in. The original machine would doubtless do good work on a solution derived from the treatment of gold ore, but on silver ore containing copper in solution as well, it is too small, especially when the solution pumps are steadily run at top capacity.

With the above described zinc-dust feeder was also supplied a new emulsifying device in the shape of a small tube-mill about 2 ft. long × 8 or 10 in. dia. The zinc-dust was to be emulsified in this small machine with a stream of solution and thence to flow to the pump suction.

This tube-mill was not found to be entirely satisfactory and it became necessary to devise a substitute for it. The idea of a tube-mill was considered good as a means of polishing and still further sub-dividing the zinc, if possible, as well as of emulsifying the solution. So a machine was built embodying this characteristic, though differing in some features from the original tube-mills.

The accompanying sketches show the essentials of the machine, though not a duplicate of the one built. The solution A and dust B drop into a small hopper

C, in which the emulsion always stands at the level of the outlet pipe *D*. The lower part of the machine, with which the hopper connects by the neck *E*, is a flat box, circular in plan, inside which revolves a disc *F*, of the shape shown in the sketch. This disc has round pockets cut in its margin, some of which contain small pebbles. Pebbles may be put in any or all of them at will. The counter clockwise motion of the disc, revolving on the shaft *G*, carries the emulsion at least once around the machine before passing out through the outlet pipe *D*. This is effected by a baffle of rubber belting *H*, which prevents the emulsion, when first formed, from passing directly to *D*.

This machine has been quite satisfactory in operation, the only difficulty arising in case the solution supply be cut off, when the machine would stick. This seldom occurs, however, and when it does is easily rectified.

One result of the operation of zinc-feeding and emulsifying may be seen in the amount of zinc present in the precipitate—7% in the main precipitation circuit, and a general average of 9% in both circuits recently. Several months average show a ratio of total zinc used to bullion produced of 1·042 to 1·000. On the score of cleanliness and avoidance of waste of zinc, it should be noted that all the zinc and solution fall into the hopper *C*, and is immediately washed inside, and there is much less chance for loss of zinc by clotting than in the hitherto prevailing methods adopted for feeding zinc-dust. This plan is especially well adapted to solutions which precipitate readily, and removes the necessity for the purchase of the comparatively expensive equipment required by the method of precipitating alternate tankfuls of solution, with the tilting box arrangement, first introduced, it is believed, at the Gold Roads cyanide plant in Arizona."—Donald F. Irvin, *Mining and Scientific Press*, June 7, 1913, p. 861. (A. McA. J.)

Gold Volatilisation Process.—"The new volatilisation plant on the Gwalia Consols is now treating 30 tons of ore per day of 24 hours, and, according to its inventor, Mr. Ben Howe, manager of the mine, the process is ideally simple and a high extraction is certain at a low working cost. The latter depends very much on the percentage of salt required to volatilise the gold. At first Mr. Howe said that 5 per cent. was necessary, but has since reduced the quantity to 2 per cent. The sulphide ore on the mine is very refractory and contains antimony and arsenical pyrite, and its successful commercial treatment by any hitherto known process has proved impossible. The lode on the property is about a chain wide and worth nearly 40s. per ton, and if Mr. Howe's process solves the problem of cheap effective treatment, the company and the whole district should have a prosperous career. In addition to the lode worked by the company there are many other similar lodes in the vicinity, all of which will be worked if a profit can be earned. A similar plant would probably prove effective on Lancefield ore, and enable that company to make a fresh start."—*London Mining Journal*, July 26, 1913, p. 709. (A. R.)

A Test for Coking Coals.—"In the spring of 1908, while studying the physical properties of coal, especially with reference to the colour of the streak and the powder, the writer observed that some coals, while being pulverised, adhered much more strongly to the pestle and mortar than others, and that the best coking coals were those which adhered most, whereas the non-coking coals adhered very slightly or not at all. This phenomenon gave promise of affording a simple, inexpensive, and yet rapid, method of distinguishing coking from non-coking coals, without the necessity of using an improved rick, or the better but more expensive way of sending the coal to a coking plant for a test in a regular coke oven. Accordingly the writer experimented systematically with all the coals in the study collection of the United States Geological Survey; a collection embracing 150 different samples ranging from lignite to anthracite, and including some of the best known coking coals of the country. Since then other men, particularly the coal geologists of the United States Geological Survey, have applied the test to many different coals, and the results obtained serve to substantiate the first conclusions that by this simple process coking and non-coking coals can be differentiated. Mr. M. R. Campbell, geologist in charge of the Section of Western Mineral Fuels of the United States Geological Survey, who applied this test to perhaps more coals than any other person, has not reported a single failure. Conversation with coal mine operators, mining engineers, and coal miners shows that this test is very little known by practical men, in view of which the author has concluded that a repetition in the presentation of the method is not only justifiable but desirable.

The method of conducting the test is as follows: Pulverize in a mortar a small quantity of the coal to be tested until it will pass through a 100-mesh sieve. Pour out the loose material, and note the amount that adheres to the mortar. With some coals the mortar and the pestle will be deeply coated with coal dust which adheres so strongly that it can be removed with difficulty; with other coals there will be only a thin film of coal dust adhering to the mortar and pestle; while with still others both mortar and pestle will be nearly as clean after the operation was completed as they were before it began. The degree of adhesion depends on the grade of the coal with reference to its coking qualities. If it adheres strongly the coal will make a good coke; if it adheres only partly the coal will make an inferior grade of coke, and if it does not adhere the coal is to be regarded as non-coking. Aside from the adhesion of this gummy soot to the mortar and pestle some coals exhibit a tendency to pack together or cohere. Although it seems to be more noticeable in coking coals, this tendency to pack was observed to a less extent in some of the non-coking type. Hence it is thought that this behaviour depends entirely on the moisture content of the coal, and has nothing to do with its qualities; however, sufficient work has not been done to prove this point. Moisture, when present in large quantities, interferes with this test, causing the coal to pack together and preventing its adherence to the mortar, hence the sample should be obtained in the dry part of the mine.

To make the test satisfactorily the sample should be as carefully prepared as one taken for chemical analysis, and should be taken in the same manner, *i.e.*, by cutting a channel from top to bottom of the part of the bed to be sampled. In sampling for coking tests, each bench of the coal should be taken separately, as here and there coal beds are found which are composed of benches, some of which will coke and others that will not. In case well-developed benches do not occur, the bed might be sampled 1 ft. at a time, keeping each sample separate from the others and properly labelled.

Each sample should be pulverized so as to pass through one-half inch mesh, and then quartered down to a convenient size for handling and transporting. About five years ago, when the author first published a description of the test, he expressed the hope that every geologist and mining engineer engaged in practical work would apply it with a view of proving or disproving, as the case may be, its value as a practical test for coking coals. Since then a number of geologists have applied it in the field and the laboratory on all kinds of coals, some of which are coked in actual practice, but not a single well authenticated failure has been reported."—M. A. Pishel, *Eng. Magazine*, September, 1913, p. 897. (J. A. W.)

Some Exact Melting-Points of Metals.—"The following are extracted from a paper bearing the title 'Melting Points of Refractory Elements': —Sb = 630°C: Ag = 960½: Au = 1063: Cu = 1083: Mn = 1260 (98% purity): Ni = 1452: Co = 1478: Fe = 1530: Cr = 1520 (98% purity); Pd = 1549: V = 1720 (98% purity): Pt = 1755: Ti = 1795."—G. K. Burgess and R. G. Waltenberg.—*Chemical News*, August 8, 1913, p. 67. (J. M.)

Sublimation of Metals at Low Pressure.—"Experiments were made with iridium (in nitrogen at 20 mm. pressure, copper (in nitrogen at 1 mm.), iron and tungsten (in a high vacuum, about 0·004 mm. at the beginning), and sensible volatilisation was observed in all cases at temperatures considerably below the melting points. In addition to volatilisation as ordinarily understood, there was evidence of straight-line emission of particles of metal. The data for a number of metals, for which information is available, are collected in the following table: the volatilisation was detected mostly at low pressures.

Metal.	Boiling Point.		Volatilisation detectable at:	Melting point at 1 atmos. *
	At 1 atmos.	*In vacuo.*		
	° C.	° C.	° C.	° C.
Mercury ...	357	160	—39	—39
Potassium ...	760	370	63	63
Sodium ...	880	420	97	97
Cadmium ...	778	450	160	321
Zinc	918	550	180	419
Bismuth ...	1420	1000	269	269
Lead	1525	1150	360	327
Silver ...	1955	1400?	680	961
Copper ..	2310	1600?	400	1084
Tin	2270	1700?	360	232
Gold	2530?	1800?	1370	1064
Iron	2450	—	950	1500
Platinum ..	2500?	—	1200	1750

—G. W. C. Kaye and D. Ewan, *Royal Soc. Proc.* 1913, A. *89*, 58; *Journal Society Chemical Industry*, Aug. 30, 1913, p. 830. (J. A. W.)

Specifications and Tests for Zinc Dust.—"The composition and physical characteristics of zinc dust are very variable and consignments, even from the same zinc smelter, may vary greatly. The cyanide worker who is unfamilar with zinc-dust precipitation will meet with many difficulties and disappointments until he adopts a scheme of testing every consignment and prepares a specification of requirements for the purchasing agent. Too many purchasing agents look mainly at the price per pound, but zinc dust cannot be purchased merely on a basis of cost.

Since the use of zinc dust has become of such importance in cyaniding an important market has been created for a high grade of that smelter by-product. Consequently chemical brokers have made special efforts to select and place on the market a zinc dust suitable for use in the precipitation of gold and silver from cyanide solutions. Between 3,000,000 and 4,000,000 lb. of zinc dust is now used annually in cyanide plants in Canada, the United States and Mexico.

The following chemical analyses give the composition of a number of different consignments of zinc dust in cyanide plants:—

	American. per cent.	American. per cent.	German. per cent.	British. per cent
Zinc	84·16	73·87	90·9	92·7
Zinc oxide ...	7·25	14·82	4·0	4·0
Lead	5·20	8·30	2·7	1·7
Silica	0·52	0·61	0·4	0·2

However, while analyses are important, comparisons between the precipitating value of zinc dust and the chemical analysis elicited the fact that a chemical analysis could not be wholly relied upon for determining the value of zinc dust as a precipitant in a cyanide plant. For example, the two American dusts, in spite of a considerable difference in the percentages of zinc, precipitated almost the same amount of silver per pound of dust. The difference was too small to notice in practice. The two foreign dusts were, however, of higher precipitating value, but as the cost was higher the commercial efficiency, that is, the pounds of silver precipitated per dollar's worth of dust, was in favour of the American product. Finally a test upon an exceptionally high grade of zinc dust, made in an 'atomizer' from good spelter, gave a precipitating value of less than half that of a common grade of smelter fume or dust.

The value of zinc dust as a precipitant in cyanide work depends upon its chemical composition and the fineness of the particles. Any specification that is drawn must recognise these two important points. In addition, an arbitrary method of determining the precipitating value of zinc dust has come into general use and is perhaps the most important test or requirement, since it combines in one criterion all the practical results of changes in chemical composition and physical condition of the dust.

The following data may be used in drawing up specifications for the purchase of zinc dust:—

Chemical and Physical Characteristics of Good Zinc Dust.—Specific gravity, 6·8 to 7·1.

Weight per cubic foot, 141 lb (2·25 gm. per cc.). Screen test: 0·5 to 1·0% on 100 mesh; 1·5 to 4·5% on 200 mesh.

Precipitating efficiency, 35% to 50%. (Pounds silver precipitated per pound of zinc) 1·15 to 1·65. Analysis: Metallic zinc, 75% to 90%; lead, 1·5% to 7·5% zinc oxide, 3% to 10%.

A good zinc dust should contain a minimum of oxide. Oxide of zinc is not only inert as a precipitant, but, being very soluble in cyanide, tends to foul the solution. Lead is valuable up to 5% or 6%, the zinc-lead couple hastening the precipitation of gold and silver from low-grade solutions. Of course, the main effective agency is the metallic zinc in a very fine state of sub-division. There should be no agglomerations and the residue on a 200 mesh screen should be free from cinder, particles of fire clay, etc.

A good zinc dust should be of a fine, uniform, light-gray colour.

A specification may be drawn up as follows:—

Precipitating Efficiency.—When shaken with a solution of silver-potassium cyanide, testing 0·15% KCN, 1 gm. of zinc dust should precipitate not less than 1·25 gm. of metallic silver.

Sizing Test.—Not more than 1% of the dust shall remain on a 100 mesh screen, nor more than 5% on a 200 mesh screen. Respective apertures of screen 0·0055 in. and 0·002 in.

Composition.—The zinc dust shall contain not less than 2% lead; the amount of zinc in a free metallic state shall not be less than 80%; the amount of zinc oxide shall not exceed 12%. The simple tests required by this specification are not beyond the scope of an ordinary mill laboratory, nor beyond the skill of the average mill chemist.

Sampling.—It is not advisable to open kegs of zinc dust and leave their contents exposed for long periods before using. A few kegs should be selected at random, opened and their contents sampled by means of an auger. The opened kegs should be used before touching the others. The samples from each keg should be thoroughly mixed by rolling on a rubber sampling sheet and the dust spread out thin. A smaller sample (100 gm.) is taken for analysis by 'spatting,' that is, by going over the thin bed of dust with a spatula taking a few grams from 20 to 30 different points. A larger separate sample can be taken for screen tests, say 200 to 300 gm. The small sample is placed in a clear, dry, wide-mouth glass-stoppered bottle.

Precipitating Test.—Making a solution of silver-potassium cyanide as follows: Dissolve 12·7 gm. of silver nitrate and 6·5 gm. of potassium cyanide in two separate quantities of water (about 450 cc. each). Slowly pour the solution of silver nitrate into the cyanide solution with constant stirring. Make up to exactly 1000 cc. This will be a 10% solution of silver cyanide. The solution should be absolutely clear and free from precipitate.

Weigh out to 0·5 gm. of zinc dust into a large beaker. Add a few cubic centimetres of water and mix, or emulsify, by stirring with a glass rod. Add 250 cc. of the silver-potassium-cyanide solution and stir for 5 min. Let stand for about 2 hr. with occasional stirring. Filter. Wash the precipitate thoroughly.

Assay the precipitate by scorification or crucible fusion. Cupel the lead button. The weight of the silver bead multiplied by two will give the amount of silver precipitated by 1 gm. of dust. The theoretical quantity of silver precipitated by 1 gm. of zinc is calculated from the equation:—

$$2AgKCy_2 + Zn = ZnK_2Cy_4 + 2Ag$$

That is 1 gm. of zinc will, under perfect conditions, precipitate 3·303 gm. silver. Hence, the simple rule, when using 0·5 gm. of zinc dust. The number of milligrams of silver multiplied by 0·0606 gives the percentage efficiency. It being difficult to get a thorough sample with so small a quantity as 0·5 gm. of dust it is advisable to make three tests and take the mean result.

Determination of Zinc Oxide.—Make up a solution of the following composition: 70 gm. ammonium chloride, 150 cc. ammonium hydrate (sp. gr. 0·90) and 250 cc. water. Weigh out 1 gm. of the dust into a beaker, add 25 cc. of the solution as above and stir with a glass rod for 5 min. Filter, washing thoroughly with hot water. Add a piece of litmus paper to the filtrate and neutralize with hydrochloric acid. Then add 5 cc. strong hydrochloric acid. Heat to 80° C. and titrate with a standard solution of potassium ferrocyanide, using uranium acetate as an indicator. The volume of solution to be titrated should be about 125 cc.

Standard Potassium Ferrocyanide.—Dissolve 21·6 gm. of pure crystals in 1 litre of water (1 cc. of this solution = 0·005 gm. zinc).

Uranium Acetate Indicator.—Dissolve 5 gm. in 100 cc. of water and add 2 cc. acetic acid. A sheet of glossy white writing paper may be used as a 'spot plate.' In titrating roughly divide the solution into two equal parts rapidly titrate one part with the standard ferrocyanide until a drop taken from the titrate gives a brown colour with the uranium acetate on the spot plate. Add the second half of the solution to the first and rapidly run in almost enough standard solution to finish the titration and complete the reaction by the addition of a few drops of the standard at a time; testing after each addition until the brown colouration is obtained.

To convert the amount of zinc found as above into terms of zinc oxide multiply by 1·245. The results are a little high owing to the slight solubility of finely divided zinc in ammonia; but they are accurate enough for all practical purposes.

Metallic Zinc.—Many methods have been proposed for the determination of zinc in the metallic state when contaminated with oxide, etc., but the most satisfactory scheme for cyanide work is to determine such zinc by the method of difference. That is determine the total zinc present as both metal and oxide and subtract from that total the amount of zinc present as oxide as determined above, The result will be correct within perhaps 0·25%. To determine the total zinc, dissolve 1 gm. of the dust in hydrochloric acid (dilute), make up to exactly 250 cc.; remove 50 cc. with a pipette. Add 5 cc. strong hydrochloric acid and 2 gm. of ammonium chloride to the solution taken for test; dilute to 200 cc. with water, heat to 80° C. and titrate with standard ferrocyanide.

Determination of Lead.—Weigh out to 10 gm. of dust and treat with 50 cc. dilute hydrochloric acid and a 1 in. square of sheet aluminium. When the zinc is nearly all dissolved wash by decantation. Remove the aluminium. Dissolve the residue in about 2 cc. dilute nitric acid. Evaporate to get rid of excess of acid. Add 10 cc. dilute sulphuric acid, and 30 cc. water. Filter and wash with cold water containing 25% of alcohol. Dry and weigh the lead sulphate in the usual way. The weight of lead sulphate multiplied by 0·736 gives the weight of lead.

Storage of Zinc Dust.—Zinc dust rapidly deterioates, especially in the damp atmosphere of a mill. The following test shows the extent to which zinc dust oxidizes in opened kegs:—

	Zinc. per cent.	Zinc oxide. per cent.
Fresh dust	91·2	2·5
After 3 months	82·1	13·9

The keg was covered loosely with the original cover. It need hardly be emphasized that zinc dust must be stored in a dry place. It should also be treated as an inflammatory substance. A separate store room should be provided.

General Remarks.—The precipitating power of zinc dust depends not merely on the percentage of metallic zinc, but also to a very great extent, upon the fineness of the metallic particles. The screen tests given are of no value in determining the relative fineness of a zinc dust sample, they are practically useful only to see that the consignment does not contain agglomerations and to show up the presence of chips of brick or wood, pebbles, etc. The particles of zinc that do the work are so minute as to be beyond the range of measurement by any screen now on the market. To measure the particles of zinc dust would require elaborate tests and apparatus far beyond the practical needs of a mill.

The precipitation test does not represent commercial conditions and in practice the amount of zinc used to precipitate gold and silver from a cyanide solution would be enormously in excess of that shown by the test. Two examples from Mexican practice will show the relationship between the test results and results in the mill: The zinc dust had a precipitating efficiency of 40%; that is 1 gm. of dust precipitates about 1·32 gm. silver.

Mill 1.—Solution contained $5\frac{1}{2}$ oz. silver + (1 kilo dust precipitated 1·1 kilo silver).

Mill 2.—Solution contained 0·4 oz. silver + (1 kilo dust precipitated 0·2 kilo silver).

The precipitation test is merely for the purpose of checking up zinc-dust shipments."—A. M. MERTON, *Mining World*, June 28, 1913, p. 1227. (A. R).

GRADING ANALYSES AND THEIR APPLICATION TO CYANIDING.—" The notable article, published by H. Stadler, on the efficiency of rock crushing machinery, has given rise to a large amount of discussion and has brought forward many varied ideas on the subject. The subject has been approached by John W. Bell, of McGill University, Montreal, who endeavours to deduce some facts which will be applicable and useful in cyaniding, using Stadler's paper as a basis.

The original paper has has been so often referred to and quoted, and the Ordinal Number so well known, that it will not be necessary to repeat it at this time. After showing how the Ordinal Number, indicated by N, is obtained for any screen of known square aperture, Mr. Bell proceeds to say that, if then a mill product is sieved on a single screen, the percentage of each size determined and multiplied by its corresponding Ordinal Number, and the quantities thus obtained added together, a composite figure is obtained which is the Ordinal Number of the product. If desired, the average volume of the particles in it can be determined by substituting this volume of N in the formula, $N = \frac{\log\frac{1}{V}}{\log 2}$ and solving for V. Mr. Bell has worked out the Ordinal Numbers for screens and grades used by Messrs. Caetani and Burt, in the test carried on by them at the El Oro mill, in Mexico, and Table I. shows the result of the calculation.

TABLE I.—Ordinal Numbers of El Oro Screens and Grades.

Mesh.	Aperture. Inches.	Ordinal No.	Grade.	Ordinal No.
25	0·0292	15·3	−25 / +40	16·5
30	0·0203	16·9	−40 / +60	18·9
40	0·0167	17·8		
60	0·0098	20·0	−60 / +80	21·0
80	0·0063	22·0		
100	0·0044	23·6	−80 / +100	22·8
150	0·0035	24·6	−100 / +150	24·1
200	0·0026	25·8	−150 / +200	25·2
			−200	28·0

In the series of screen analyses and sand indices, which mean gold-silver extraction, available and at hand in the Caetani-Burt paper, the data needed to study the relation between volume and extraction are found. By obtaining the Ordinal Numbers of the El Oro screens and grades, it was a simple matter to calculate the Ordinal Numbers of each of the various products made at the El Oro mill, and knowing the corresponding gold and silver extractions obtainable, the results could be plotted on an Ordinal Number extraction basis.

It is stated that the Ordinal Numbers of nine products were calculated and plotted together with their corresponding gold and silver extractions. This established 11 points on each curve; it was at once apparent that the curve took the form of two straight lines, as shown in Fig. 1.

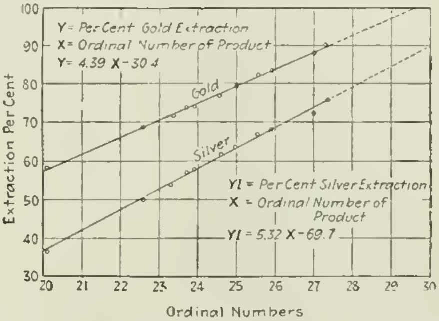


FIG. I.—Extraction curve based on ordinal numbers.

The conclusions were: First, that there is a close relation between volume and extraction. Second, that the extraction does not seem to be proportional to the increase of surface exposed, since the surface exposed increases rapidly from one grade to another, while the increase in extraction appears to be constant for a great interval. Third, that Ordinal Number extractions taken off the curve or calculated by the formula, check with the sand indices calculated by Caetani and Burt, which checked closely with the results actually obtained. Fourth, that a 100% gold extraction is indicated for an $N = 29·7$ product and complete silver extraction for an $N = 39·9$ product. These figures are, however, of merely speculative interest.

Ordinal number extractions may be compared with corresponding sand indices in Table II., in which

TABLE II.—Comparison of Ordinal Number Extractions Sand Indices.

Product.	Ordinal Number.	Sand Index Extraction, Gold.	Ord. No. Extraction, % Gold.	Difference.
Feed to No. 4-5 tube mills	20·1	58·4	57·8	+0·6
Stamp mill pulp	22 6	68·9	68·8	0·1
Spitzkasten 1, spigot 1 ...	23·3	71·5	71·9	−0·4
Feed to No. 3 tube mill ...	23·7	73·6	73·6	0·0
Spitzkasten 1, spigot 3 ...	23·9	74·1	74·5	−0·4
Spitzkasten 1, spigot 4 ...	24·6	77·2	77·6	−0·4
Discharge No. 3 tube mill	25·0	79·4	79 3	0·1
Spitzkasten 1, spigot 6 ...	25·5	81·6	81·5	0·1
Spitzkasten 2, spigot 3 ...	25·9	83·1	83·3	−0·2
Spitzkasten 2, spigot 4 ...	27·0	87·8	88 1	−0·3
Slime to agitation plant..	27 3	89·9	89·4	+0·5

Note. — The Ordinal Number extractions were obtained by the formula $Y = 4·39\ (X) - 30·4$, where $Y = \%$ extraction. $X =$ the Ordinal Number of the product.

they are set forth. The relation between volume, surface and extraction was further analysed by means of the data used by Caetani and Burt in calculating the sand indices. Volume extraction and surface extraction curves were plotted from the data in Table III. Commencing with the largest particle in Table III, the volume was halved at each step and the probable extraction for each volume in this series was obtained from the curve. The results obtained are tabulated in Table IV. Table V. illustrates how the gold extraction increases for equal increases in surface. Table IV. appears to indicate a more or less even increase in extraction each time the volume is reduced by one-half, and Table V. does not support the belief expressed by the authors of the El Oro paper that the maximum extraction must be proportional to the surface of the ore exposed by crushing. It is not known whether this relation between volume and extraction applies only to El Oro ore, or is of general application, but if it has general application, there would be substantial reason for introducing the Stadler series of grades into the regular milling practice. The simplification realised by being able to represent a complex screen analysis by a number, is worthy of the attention of the practical mill superintendent. It permits the use of simple charts showing the variations in the size of mill products and the resulting increases or decreases in extraction. In an all-sliming mill there is a classifier overflow of a certain ordinal number which, if maintained, will insure maximum profit. The mill chart for this product should be a straight line as nearly as possible, and any abnormal carelessness in the adjustment of the classifier would be indicated graphically. Occasional or systematic estimates of the ordinal numbers of the mill assay and samples might well repay the small additional expenditure of time and money required.

TABLE IV.—Probable Exraction for each volume.

Volume-billionths of a cubic inch.	Gold Extraction Per cent.	Increase Per cent.
6850	45	—
3425	51	6·0
1712	54·4	3·4
856	58·0	3·6
428	62·2	4·2
214	66·2	4·0
107	70·2	4·0
54	76·0	5·8
27	80·8	4·8
14	86·0	5·2
7	90·0	4·0
3	92·6	2·6
Volume reduced ½—11 times		11)47·6
	Average increase =	4·3

TABLE III.—Data for Plotting the Curve.

Product	+40	+60	+80	+100	+150	+200	−200
Average diameter grain-inches (d)	0·0190	0·0134	0·0079	0·0052	0·0039	0·0031	0·001
Volume-billionths cubic inches	6850	2400	492	140	59	30	4
Gold extraction	45	53	61	68	75	80	92
Silver extraction	17	28	42	51	59	66	79
Surface exposed	53	75	127	192	256	322	625

Note.—The ratio of the original surface to that exposed by crushing has been assumed to be proportional to the number of particles in unit volume multiplied by the square of the average diameter of a particle $= \frac{1}{d} \times \frac{d^2}{1} = \frac{1}{d}$

TABLE V.—Increase of Gold Extraction with Surface Increase.

Surface Square Inch.	Extraction Per cent.	Increase in Extraction for each Increase of 100 Sq. in., Per cent.
100	56·7	—
200	70·0	13·3
300	79·0	9·0
400	84·6	5·6
500	88·5	3·9
600	91·2	2·7

Another argument in favour of calculating the Ordinal Numbers of the products is the ease with which two or more can be compared with respect to fineness.

The simplification realised by being able to represent a complex screen analysis by a single number is valuable also in determining the efficiency of screen, classifier, etc. For example, the principal factors which affect the character of the overflow from conical desliming classifiers, are the diameter of the overflow rim, the volume of overflow in cubic feet per minute, the efficiency of the feed plate in destroying the entering velocity of the incoming feed, and the rate of withdrawal from the spigot, although this factor is closely related to the second mentioned.

It is said that experience has shown that the simplest way to study this problem is to sample and measure a number of overflows, determine their ordinal numbers and plot the results for a cone of given diameter.

It is noted by Mr. Bell that screen analyses made by hand are both antiquated and expensive. For a comparatively small outlay, machines can be obtained which will permit turning out more screen analyses per day than a muscular assayer will do in several days, and the machine screening will be far more reliable and accurate, not to mention the saving in assayer's time and energy."—J. W. BELL.—*Engineering and Mining Journal*, May 24, 1913, p. 1044. (H. A. W.)

MILL-SITES.—"Little is gained in general by extensive excavation, especially if it be in hard ground, since the improved methods of elevating and transporting render it in a measure unnecessary. The ideal metallurgical works should be so arranged that all parts may be accessible by a broad-gauge railway line. This necessitates laying-out the plant by the side of the tracks, with the longer axis of the buildings parallel to the rails, and not transverse, whereby the greatest convenience in the matter of dumping room is secured. The plant, if at all extensive or complicated should be divided into departments or sections, so as to have good ventilation with freedom of each section from the smoke, gases, and flying particles from the others. By this disposition, the cost of buildings, and particularly of grading and of retaining walls, may be much diminished. Where, on the contrary, the different parts of a plant are huddled together, not only is the cost of working greater, but that of subsequent additions to the works will also be greater. Further, the very compactness of a plant, which is so much valued by some designers, will invariably prevent the orderly and logical additions of parts that may be subsequently found necessary.

Most metallurgical works are built upon hill-sides in order to get the benefit of gravitation, by which the substances undergoing treatment are assisted on their way. The practice of so building the works has become fixed by the habit of generations, and with some builders has become almost second nature. But with the improvement in machinery, by which a wealth of appliances for hoisting, transferring, and lowering the heavy and bulky materials of metallurgy is now available, the arguments and reasons that formerly led to the hill-side have mostly lost their force, and a rival school, advocating the so-called level site, has appeared. Their arguments relate chiefly to smelting, but have an important bearing upon other metallurgical processes.

A sloping or hill-side site is made up virtually of a number of terraces, upon each of which some part of the plant is placed; there may be two, three, or more of these terraces, the result of excavation into the hill-side. The single-level site, which ostensibly is composed of but one level, virtually has quite as many, if by 'level' is meant the elevation on which work is performed. For there must be, first, a lower dumping ground on which the waste is deposited, and also several floors of greater or less extent, upon which the men stand to work at their tasks. Each of these floors may be entitled 'levels' with as much propriety as if they were the generally more extensive areas embraced in the hill-side site. Thus there must be a space about the head as well as the base of cupola furnaces for their proper working; the use of ore-bins implies at least two levels; and so for the usual arrangements of metallurgical works of all kinds. The conflict of opinion as regards one-level *versus* terraced sites was largely based upon a misapprehension, and there is really not such a great difference between the two opposed plans of construction as has been generally imagined. The greater differences arise from the method of supporting the levels and their relative positions about the works. In the hill-side plan they are laid out in an orderly sequence from top to bottom; while in the one-level plan they are scattered about the terrace according to convenience or the views of the designer. In the former they rest upon the solid or filled ground: in the other they are supported, for the most part, on framework of wood or metal. The question, therefore, is narrowed to a consideration of the relative advantages of floors or levels supported by posts, or by earth or stonework. It is necessary to consider the matter of installing and using the different classes of elevating and transferring machinery made advisable by the one or other plan, together with the possibility of future enlargements and alterations—a matter that should always be kept in mind.

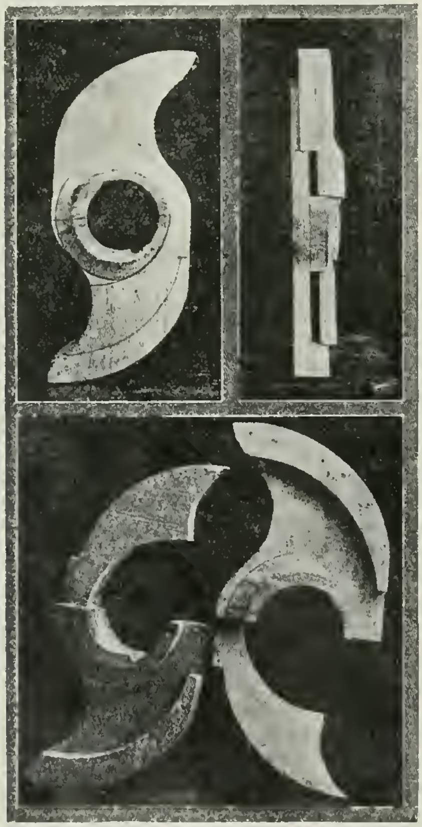
There is no question that the one-level site, with buildings somewhat widely separated, is the most favourable to enlargements; and while the question of convenience and economy of working to be settled by the engineer on the spot, Mr. Lang is convinced of the superior advantages of the one-level plant in those respects also. Doubtless, most builders will agree that ventilation is better, and that as a general thing the danger from the spread of fire is less. As a rule, there is greater economy of space in the one-level plan, as the room directly beneath the upper floors is not wholly occupied by the supports, and because the stowage spaces can be better utilised. Ore, for example, which it would be out of the question to transfer for storage from one level to another, may be moved on the same level to wherever convenience may dictate. Thus any given space on the terrace may be used successively for several purposes, while on the hill-side plan the engineer is rigidly held to one use for one space.

In the case of stamp-milling where the course of the ore is constantly downward, and where there are no materials of any moment that require to be returned from lower to higher levels, there is little opportunity to argue against the hill-side construction. But in processes like cyaniding, where the re-elevation of the liquids forms an important part of the process, and which cannot be obviated, there is less reason for seeking a site upon sloping ground, as the pumping of the liquids from vat to vat cannot be much forwarded by differences of levels between the vats."—HERBERT LANG. *Mining and Scientific Press; Mining Magazine*, August 1913. (A. R.)

A NEW USE FOR COBALT.—"From an article in the *Ironmonger* it appears that cobalt may be largely used in steel manufacture. Steel experts in Sheffield are experimenting with a new grade of high-speed steel which contains a small proportion of cobalt in addition to tungsten, vanadium, and other usual ingredients. The effect of cobalt is to intensify the characteristic qualities of tungsten and vanadium, and to impart to the tools increased speed and durability. Cobalt steel is already made in America and Germany, and some is offered in England at very high prices. The Sheffield steel makers will not offer this new steel to the engineering world till experiment has demonstrated beyond doubt its permanent commercial value, and the particular purposes for which it is adapted. A small quantity has been made in Sheffield, and some manufacturers have been asked to supply it. Cobalt is found in Ontario, and at present the supply is limited, but there is plenty of ore which could be mined if a regular demand arose. The current price is $2,500 to $3,000 per ton. Cobalt steel will be more expensive than the best vanadium steels, and it is not thought in Sheffield that it will supersede the present high-speed steels to any considerable extent, although it will be advantageous for various purposes. Experiments as to the effect of cobalt on carbon steel mixtures are being made."—ANON, *Brass World*, ix., 6, 217; *Journal of the Franklin Institute*, Vol. clxxvi., No. 2, August, 1913, p. 218. (J. G.)

LEAD SALTS IN CYANIDING.—"The following example of the effect of lead salts on the extractions by cyanide of the gold and silver in an ore which I was testing recently may be of interest. The ore, assaying about $7 Au and 5 oz. Ag, consisted of a mixture of clean oxidized quartz and vein material (mostly calcite), the last carrying some of the metal. An analysis of the ore revealed no constituents prejudicial to treatment by direct cyanidation, except the presence of traces of manganese, which might possibly retard the dissolution of the silver. Agitation with solution produced a 95% extraction of the gold in 20 hr., but after 60 hr., even with solution titrating as high as 0·25% KCN, the silver extraction reached only 58%. The addition of lead acetate (0·5 lb. per ton of ore) at the start of the agitation caused the gold extraction to drop to 65%, and that of the silver to rise in 24 hr. to 75%. By agitating for 20 hr. and then adding the lead acetate and continuing the agitation in the same solution for a further 24 hr., the gold extraction reached 96% and the silver 89%. No appreciable increase in extraction of either the gold or silver was obtained by decanting the solution after the 20 hr. and replacing it with fresh solution, before adding the lead acetate. Finely ground litharge had exactly the same effect as the lead acetate. The tests were conducted with precipitated solution from a plant where lead acetate, at the rate of 0·6 lb. per ton of ore, is added to the solutions."—VYVYAN C. BENNETT, *Mining and Scientific Press*, July 26, 1913, p. 154. (A. McA. J.)

THE BERRILL CAM.—"The present battery cam has been developed after numerous trials with various types during many years. Among those types that will be remembered are the plain long armed cam; split type; Spiers' dove-tailed; the bed-post cam, in which the arms fit into the boss similarly to the angle iron of a bed; and the ordinary keyed style, which is still used in many mills. The first five cams were keyed to the shaft. Later came the Blanton patent, which most millmen consider to be the last word in cam design. Then followed the improved Blanton, the cam-shaft of which looks, in section, like a circular saw, this cam wedging in ten places instead of one, as in the original. Following this type came the African cam, designed by Hans C. Behr. This cam raises the stamp slowly, the lifting being accelerated toward the end of the arm. A design for a sectional cam, to allow of easy removal from the shaft without disturbing the others, has often been attempted, and about the most ingenious I have seen is the Berrill patent divided or split type. As a cam is a difficult piece of apparatus to describe, and this one in particular, the illustrations will help to explain its construction. The Berrill cam is made in two parts or sections which, when fitted together, form a complete cam. As is seen in the illustration, there is an opening in both parts of the cam, which allows it to be fitted on the shaft, or removed at will. A Blanton key is used for holding the cam fast on the shaft, and a copper rivet is driven through the two sections, near the centre of the blades, and riveted up. This is simply to keep them from chattering. As the weight of the stamp comes on the cam, the scarfed parts are jambed together, and being a good fit, and also held by the rivets, there is no movement of the two parts, and a rigid cam is formed. The opening in each section is made ⅛ in. larger than the shaft to be used; and when fitting the parts together, there is plenty of room between any two stamps to tighten them.



Showing Wooden Model of Cam. (1) Complete Cam; (2) Fitting Cam Together; (3) Cam in Halves, Showing Construction.

Advantages of this Type.—The advantages of this cam might be stated as (1) it is unnecessary to remove other cams to replace a broken one; (2) if one part of the cam is broken or chipped, it may be discarded and replaced by a new one; (3) in replacing a broken cam-shaft, the new shaft can be put in place, and the cams fitted afterward; and (4) a saving in time.

On the whole, there are not many cams broken in a battery, yet the Berrill device has points to commend its use. The Ivanhoe mill of 100 stamps breaks 19 cams yearly: Alaska Treadwell mill of 540 stamps breaks 42 yearly; and Homestake cams last from two years up.

A cam as described, was made in Melbourne at a a cost practically the same as for the ordinary cam. This cam worked for two years at the Hannan's Central mill, at Kalgoorlie, lifting an 1100-lb. stamp 105 times per minute; and in the Horse-Shoe mill a similar time, lifting a 1250.lb. stamp 112 times per minute. The mill managers at both places were satisfied with its behaviour. It was designed by N. H. Berrill, of Kalgoorlie, Western Australia."—M. W. VON BERNEWITZ.—*Mining and Scientific Press*, June 28, 1913, p. 995. (A. McA. J.)

ROTARY VACUUM FILTER.—"A rotary vacuum filter for slime pulp or for the separation of a valuable liquid from a finely divided solid has been patented by Mr. Thomas Breakell, of Wirksworth, England. The main object of the invention is to filter slime pulp continuously and at the same time provide means for keeping separate the valuable solution and the wash solutions. The invention is along the line of the well-known Oliver and Portland types of vacuum filters, but differs from them in being a closed drum in which a vacuum is maintained and into which the several solutions are drawn and collected in different compartments.

As shown in Figs. 1 and 2 the filter consists of a closed drum or barrel *a*, having a filtering surface *b* and arranged to rotate in a suitable frame. Within the barrel is a receptacle *d* divided by a vertical partition into two compartments *f* and *g*. A space *h* is left beneath the receptacle. On the inner periphery of the drum are arranged numerous ribs *l* which serve to divert the filtered fluid into the proper receptacles. A slime feed-box is placed at *n*, the lower edge of which, *m*, serves also as a knife to scrape the filtered slime from the face of the filter. Suitable pipes and distributing rollers for applying wash solutions are placed at *s* and *t*, and *q* and *u*. Pipes for withdrawing the filtered liquids under reduced pressure enter the drum through the bearing at one side of the drum, and extend into the several receptacles for solution. In the drawing these pipes are numbered 2, 3 and 4. Their outer ends are connected with a vacuum pump.

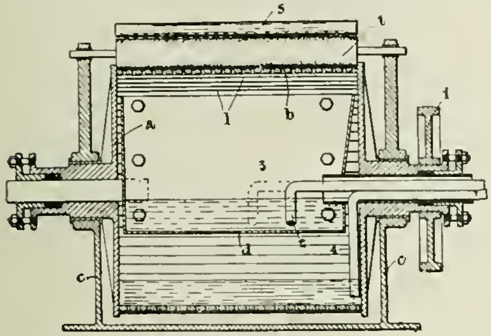


Fig. I.—Vacuum Filter.

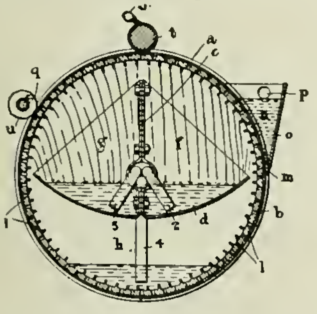
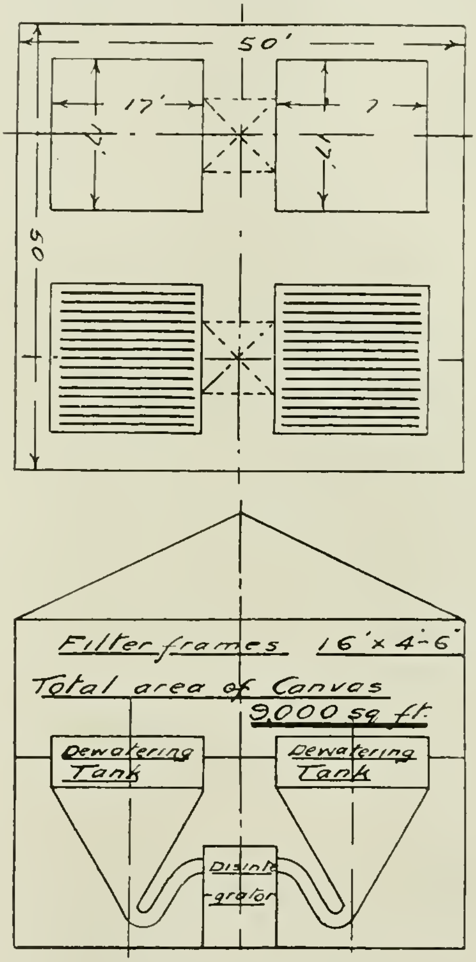


Fig. 2.—Vacuum Filter.

In operation the drum is rotated and the vacuum applied. Slime is delivered into the receptacle *n* and thus brought in contact with the filtering surface on the rising side of the drum on which it forms the usual cake. The solution passing through the filter is diverted by the inner ribs into the receptacle *f*. At the top of the drum a wash of barren solution may be applied from the pipe *s*, being distributed over the surface of the cake by means of the roller *t*. This solution, on passing through the filter, is diverted by the inner ribs into the receptacle *g*. Similarly, on the descending side of the drum, a wash of clear water may be applied from the pipe *q*. within a gauze distributer *u*. Part of the solution filtered at this point may pass into the receptacle *g* and the balance into the space *h*. The several solutions are removed by the pipes 2, 3 and 4.

It is not apparent that this filter offers any advantage over the better known types of drum filters either in the method of feeding the slime or in the manner of keeping the solution separate. The use of the rollers in connection with the wash solution, may be advantageous in getting a thorough distribution over the cake and in filling slight cracks that may appear. (1,064,702 June 17, 1913.)"—*Metallurgical and Chemical Engineering*, August 1913, p. 594. (A. McA. J.)

Dewatering Tank.—"The Waihi Gold Mining Co., Ltd., of New Zealand, completed a plant in 1908 for dewatering about 700 tons of slime per day. The plant works well, and gives a thick discharge of slime, at an extremely low cost. The plant consists of four conical bottomed tanks, each 17 ft. square, and the cones of each pair of tanks are connected up to a disintegrator. The dewatered slime is churned up with weak cyanide solution in the disintegrator, and from there it is elevated to agitators. Each tank contains 15 Moore filter leaves, each leaf being 16 ft. long and 4 ft. 6 in. deep. The total filtering area in the four tanks is about 9,000 sq. ft. The dewatering operation is simply a matter of collecting and discharging cakes. Care has to be exercised in regulating the outflow of the thickened slime, otherwise the thin slime, in the upper part of the tanks, may break through into the disintegrator."—Charles A. Banks, *Mining and Scientific Press*, July 26, 1913, p. 154. (A. McA. J.)



Dewatering Plant used by the Waihi Gold Gold Mining Co.

HARDINGE *v.* CHILEAN MILLS.—"In view of the prominence which the conical mill has attained in the fine-crushing field within the few years since its introduction, the following comparison with its more mature forerunner, the Chilean mill, based on extensive tests, is submitted in the interest of the milling profession.

The conical mill used in these tests is the 8 ft. Hardinge pebble-mill, having a cylinder 22 in. long. The cylindrical part of this mill is lined with cast iron liner plates, and the conical extensions with silex bricks bound together by cement. Each liner plate carries a projecting lifter, the function of which is to increase the height of drop of the lifted material. Danish No. 5 pebbles, obtained from the coast deposits of Denmark, are used for the grinding charge. The Chilean mill used is a fast-running, 3-roller, 6 ft. Saturn mill, with screens of 0·037 in. opening. The feed to these mills is the oversize of Callow screens having 0·029 in. openings, which follow rolls crushing to ½ in.

Advantages of the Conical Mill.—For the ore of this mine, a moderately hard but fissile schist, impregnated with finely disseminated granular chalcocite, the conical mill has proved itself superior, metallurgically and economically, as a fine-grinding machine. This superiority is attained by a combination of commendable characteristics, namely, smoothness and steadiness of operation, delivery of a product enabling better extraction, more economical water consumption, a lower operating and maintenance cost, and a low rate of depreciation.

Steadiness in operation, of paramount importance in plants of such large capacities, is effected in this type of mill by its simplicity in principle and the consequent simplicity in construction. Discharge screens, dies, and mullers are eliminated, and in their place more desirable crushing equivalents are substituted. Thus the screen of the Chilean mill is replaced by a perpetual device; the dies by linings which have long life; and the mullers by flint pebbles which are replaceable without interruption to operation.

Delays with these two types of mills in this plant have been found to be as follows :—

Chilean Mills.	Per cent.	*Hardinge Mills.*	Per cent.
Screen delays ...	0·57	Re-lining delays...	0·71
Repair delays ...	1·54	Repair delays ...	0·58
Total delays ...	2·11	Total delays ...	1·29

From the above it is seen that the delays of the Hardinge mill approximate 60% of those of the Chilean mill, and that relining constitutes more than one-half of the total delay. This is mainly due to the fact that when a mill is relined it must be idle about 48 hr., so as to give the cement used for binding time to set. This delay, however, can be materially reduced by means of shell stands, so that a newly relined shell will always be ready to be replaced by an overhead crane when a worn-out shell is to be removed. Allowing 2 hr. for this replacement, the delay from this cause will be reduced to about 0·05%. Thus the necessary delays with this mill are reduced to those of repairs to bearings and pinions, lifting out a shell with worn lining and reinstating a new one, and the occasional replacement of feed scoops, which approximate a total delay of about 0·6%.

Labour Charge.—Less actual attendance is required by the Hardinge mill. This makes it possible to reduce the operating cost in plants where the duty of the attendant can be so distributed as to include the supervision of other apparatus. Occasional pebble feed, lubrication, and a lookout for obstructed discharge boxes, are the only services required. By their adoption in this plant the operating labour cost of fine crushing has been reduced about one-half.

For the reduction practised at this plant, the conical mill has proved itself to be a much superior fine-grinding machine. Because of the generally granular character of the chalcocite of this ore, it is the aim to make a product of such size as will liberate a maximum mineral content with as small a production of ultra fines as possible. It has been found that a product which contains a maximum percentage between the sizes of 60 and 200 mesh is best. Below is given a typical screen analysis of feed and product for both types of mills. From this it is seen that the Hardinge mill yields 37% of the desired size of material, or about 50% more than the Chilean mill, and with a smaller production of slime.

Mesh.	Chilean Mill. Feed.	Chilean Mill. Product.	Hardinge Mill. Feed.	Hardinge Mill. Product.
+ 4 ...	13·9	...	12·9	...
+ 10 ...	47·5	...	47·3	...
+ 20 ...	22 9	2·3	26·8	0·2
+ 30 ...	5·2	11·8	5·0	3·2
+ 40 ...	0·9	6·7	0·8	4·9
+ 60 ...	1·0	11·4	0·8	13·8
+ 80 ...	0·5	6·7	0·4	10·4
+100 ...	0·4	5·4	0·3	8·6
+150 ...	0·5	6·3	0·3	8·0
+200 ...	0·7	7·2	0·5	10·0
−200 sand ...	1·2	10·6	0·8	10·0
−200 slime ...	5·3	31·6	4·1	30·9

Power.—The Hardinge mill also consumes less power. At this plant, a 150 h.p. induction motor operates three 8 ft. Hardinge mills or two 6 ft. Chilean mills, and the power consumption for the above reduction is as given below. The consumption is given on the basis of both crude-ore tonnage and actual feed tonnage, the latter being approximated at 70% of the former.

	Crude-ore tonnage. H p.-hr. per ton.	Actual feed tonnage. H.p.-hr. per ton.
Chilean mill ...	7·5	10·7
Hardinge mill...	6·7	9·6

A striking feature brought out by the comparative operation of these mills is the difference in the duty exacted of the cone tanks. The units of the plant operated with Hardinge mills have shown an average reduction of nearly 75% in the solid feed and a 40% reduction in the water feed to these tanks, as compared with the Chilean mill. This is to be attributed to the combined result of the smaller quantity of water fed to the grinding mill and of the smaller production of slime. For plants where the production of extreme fines is not desired, the opportunity is thereby offered of lessened outlay for dewatering equipment.

Cost of Maintenance.—The maintenance costs of Chilean and Hardinge mills are shown in the accompanying tables. The cost for each mill is based on crude-ore tonnage, so that the cost per ton of actual feed would be 40% greater than the cost shown, as in the reduction practice of this plant about 70% of the crude-ore tonnage passes through the fine-grinding mills.

The figures presented below cover a wide range of operation, and, going into detail as to maintenance, afford an interesting comparison.

Chilean Mill.
(Tons milled, 826,000.)

		Cost per ton.
Driving mechanism :		
Shafts, pinions, and gears ...	$0.00230	
Spindles, muller bushings, etc.	0.00275	
Miscellaneous	0.00042	
		$0.00547
Crushing mechanism :		
Dies (12,910 tons per die) ...	$0.01079	
Tyres (7,310 tons per tyre)..	0.00987	
Screens (183 tons per screen)	0.00893	
Miscellaneous	0.00176	
		$0.03135
Total supplies		$0.03682
Repair labour		0.00841
Shop expense		0.00543
Total maintenance cost ..		$0.05066

Hardinge Mill.
(Tons milled, 450,000.)

		Cost per ton.
Shafts, pinions, and gears ...		$0.00036
Lining :		
Liner plates and lifters ..	$0.00403	
Silex	0.00300	
Cement	0.00050	
		$0.00753
Pebbles (2·51 lb. at $0.013) ...		0.03262
Miscellaneous		0.00012
Total supplies		$0.04063
Repair labour		0.00268
Shop expense		0.00214
Total maintenance cost ...		$0.04545

From this it is seen that the Hardinge mill, for the practice of this plant, shows a maintenance cost of about 0.5 c. per ton less than the Chilean mill. It is to be noted, however, that pebble consumption constitutes 70% of this cost, and the freight on pebbles comprises approximately 50% of their expense in this locality. The item of 'shafts, gears, and pinions' is probably somewhat low, in that these mills have not been operated sufficiently long to obtain a true average. Nevertheless, this part of the cost is small, since these gears have a long life, and, constituting but a small percentage of the total, is inconsequential.

Depreciation.—However, the decisive factor of the lower cost of the Hardinge mill is the low rate of depreciation. The life of its shell, if proper care is taken that the lining is not allowed to wear through to it, seems comparatively infinite. Six Chilean mills have shown an efficient life of 825,000 tons, making the rate of depreciation, inclusive of transportation and installation costs, about 3 c. per ton. Allowing a life of 10 years for the Hardinge mill, its depreciation cost would be less than 0.5 c. per ton.

Summarizing these factors, the net gain in cost by operating with the Hardinge mill, for the practice of this plant, shows as follows :—

	Costs per ton.
Operating.	cents.
Labour	0.50
Power, 0·6 kilowatt-hour...	0.75
	1.25
Maintenance	0.50
Depreciation	2.50
Saving	4.25

To the above are to be added other advantages the more conspicuous of which are : greater capacity by reason of lower power consumption and lower delays ; superior product enabling a better extraction to be made ; smaller water consumption ; and for minimum slime practice requires less dewatering equipment.

Furthermore, this mill is not yet out of the experimental stage, and there are possibilities of still better performances. For instance, by lengthening the cylinder of a 6 ft. middling re-crushing mill from 22 to 38 in., it was found that the capacity of the mill was doubled, the power consumption lessened, and the pebble cost decreased to nearly one-half. It would seem, however, that this idea can be carried too far, for the more the cylinder of this mill is lengthened, the more it tends to approach a tube-mill, and so become a slimer. For re-grinding middling, however, this variation in dimension is a step in the right direction, in that the liberation of occluded mineral necessitates a fine product. Also large percentage variation in the sizes of feed seems to have a considerable influence on the consumption of pebbles. Thus it was found in a test in which all the feed was sized through 2·5 mm., that the pebble consumption was 1·85 lb. per ton of actual feed as against a consumption of 3·60 lb. with the oversize feed shown in the table.

Mechanical Crushing Efficiency.—An interesting comparison which, while based on rather ideal assumptions, is so decisive in result as to be given credit, is the mechanical crushing efficiency of these machines determined by the method of calculation discussed by Algernon Del Mar in his article on 'Mechanical Efficiency of Crushing.'* These calculations, as shown in the accompanying tables, are based on Rittinger's law that 'the work done in crushing is proportional to the reduction in diameter.' This assumes that all surfaces exposed give the same unit resistance to crushing, whereas it is to be inferred that there are some surfaces which, by reason of inherent fissility of the ore, offer a lower unit resistance than surfaces not so favoured. However, in view of the large number of surfaces produced, it would seem reasonable to assume that an average unit resistance to crushing will prevail in a not unduly long test. Furthermore, since in these calculations both machines are treated equitably with regard to the practical variations which do not enter into the law, the comparative results can be considered fairly reliable.

Table I. shows the crushing efficiency without regard to quality of product made, from which it is seen that the units of reduction performed by the Hardinge mill exceed those of the Chilean mill by from 18% to 23%, depending upon the degree of accuracy attained in the assumptions made in the calculations, and considering 5% as a safe limit. This evaluation proves that the Hardinge mill converts more of the power consumed into reduction of the charge than does the Chilean mill.

Table II. shows the comparative crushing efficiency with regard to size of product made. From this it is seen that in amount of work performed on the various sizes of the feed, the Hardinge mill exceeds the Chilean mill in all cases, again showing that this mill converts more of the power taken by it into actual work done. Furthermore, the excess work done is mostly expended on the grades of product desired, thereby proving that this mill more efficiently fulfills the duties assigned. It is here that the cone comes into play. This geometrical device

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*Eng. and Min. Jour., Vol. XCIV., No. 24, p. 1129 (Dec. 14, 1912).

Table I.—Mechanical Crushing Efficiency, Hardinge v. Chilean Mills.

On basis of law that 'the work done in crushing is proportional to the surface exposed in crushing' and therefore 'nearly proportional to the reduction in diameter' or 'nearly proportional to the reciprocals of the diameters crushed to."

Hardinge Mill.

Mesh.	Reciprocal of average size.	Feed, per cent.	Relative surface in feed.	Product, per cent.	Relative surface in product.
+ 4 ..	4·1	12·9	53	...	..
+ 10 ...	7·2	47·3	341	...	...
+ 20 ...	18·3	26·8	490	0·2	4
+ 30 ...	37·7	5·0	189	3·2	121
+ 40 ...	58·4	0·8	47	4·9	286
+ 60 ...	83·6	0·8	67	13·8	1,154
+ 80 ...	138·0	0·4	55	10·4	1,435
+ 100 ...	163·0	0·3	49	8·6	1,402
+ 150 ...	220·0	0·3	66	8·0	1,760
+ 200 ...	303·0	0·5	151	10·4	3,030
− 200 ...	400·0	4·9	1960	40·9	16,350
		100·0	3468	100·0	25,552

Chilean Mill.

Mesh.	Reciprocal of average size.	Feed, per cent.	Relative surface in feed.	Product, per cent.	Relative surface in product.
+ 4 ...	4·1	13·9	57	...	..
+ 10 ...	7·2	47·5	342	...	.
+ 20 ...	18·3	22·9	419	2·3	42
+ 30 ...	37·7	5·2	196	11·8	445
+ 40 ...	58·4	0·9	53	6·7	391
+ 60 ...	83·6	1·0	84	11·4	953
+ 80 ..	138·0	0·5	69	6·7	925
+ 100 ..	163·0	0·4	65	5·4	880
+ 150 ...	220·0	0·5	110	6·3	1 386
+ 200 ...	303·0	0·7	212	7·2	2,182
− 200 ...	400·0	6·5	2600	42·2	16,880
		100·0	4207	100·0	24,084

Summary:

	Hardinge.	Chilean.
Units of work in product ..	25,552	24,084
Units of work in feed	3,468	4,207
Units of work done by mill uncorrected for capacity ...	22,084	19,877
Units of capacities of 2·50 tons and 2·25 tons per horse-power day, respectively	55,210	44,723

Excess units of work done by Hardinge mill, 10,847.

Excess efficiency, assuming method of calculation correct, 23·45%.

Excess efficiency, assuming 5% as the limit of error, 18·45%.

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serves the function of adjusting the crushing energy expended so as to be proportional to the force required to reduce the particles to a given size. This is effected by two principles that are inherent with the operation of the mill. First, through the continual displacement of the larger particles of the charge upon the smaller, there takes place a segregation of the particles in the cone according to size—the larger assuming positions at the greater diameter and the smaller receding toward the smaller end of the cone of the mill; second, through the combined action of this segregation and the diminishing action of centrifugal force toward the apex of the cone, varying intensities of energy are imparted to the pebbles—the larger receiving greater inertia by reason of greater mass and greater lift, and the smaller less and less inertia by reason of the smaller mass and lower lift.

Grinding Zones.—There exist within the mill an orderly arrangement of zones of ore particles, each requiring a certain amount of impact to be reduced to a given size, and a series of zones of forces so arranged as to impart impacts that tend to be proportional to the crushing energy required by the ore particles upon which these forces are exerted. For these reasons the production of slime is minimized and the accumulated forces are utilized to best advantage, whereas in the Chilean mill the crushing

Table II.—Screen Size Crushing Efficiency.

Hardinge Mill.

Mesh.	Reciprocal of aperture.	Feed (cumulative percent.)	Relative surface in feed.	Product (cumulative per cent.)	Relative surface in product.	Relative surface produced.
4	4·9	87·1	427	100·0	490	63
10	13·3	39·8	529	100·0	1,330	801
20	29·4	13·0	382	99·8	2,934	2,552
30	50·5	8·0	404	96·6	4,878	4,474
40	66·7	7·2	480	91·7	6,116	5,636
60	115·0	6·4	736	76·9	8,844	8,108
80	147·0	6·0	882	66·5	9,776	8,894
100	182·0	5·7	1037	57·9	10,538	9,501
150	272·5	5·4	1472	49·9	13,598	12,126
200	333·0	4·9	1632	40·9	13,620	11,988

Chilean Mill.

Mesh.	Reciprocal of aperture.	Feed (cumulative percent.)	Relative surface in feed.	Product (cumulative per cent.)	Relative surface in product.	Relative surface produced.
4	4·9	86·1	422	100·0	490	68
10	13·3	38·6	513	100·0	1,330	817
20	29·4	15·7	462	97·7	2,872	2,410
30	50·5	10·5	530	85·9	4,338	3,808
40	66·7	9·6	640	79·2	5,283	4,643
60	115·0	8·6	989	67·8	7,797	6,808
80	147·0	8·1	1191	61·1	8,982	7,791
100	182·0	7·7	1401	55·7	10,137	8,736
150	272·5	7·2	1962	49·4	13,462	11,500
200	333·0	6·5	2165	42·2	14,053	11,888

Comparison of Efficiency.

(With units corrected for capacity.)

Size.	Hardinge Mill, energy units.	Chilean Mill, energy units.	Diff. in favour of Hardinge.	Distribution, per cent.
At 4 mesh	158	153	5	0·02
At 10 mesh	2,002	1,838	164	0·57
At 20 mesh	6,380	5,423	957	3·32
At 30 mesh	11,185	8,568	2617	9·08
At 40 mesh	14,090	10,447	3643	12·64
At 60 mesh	20,270	15,318	4952	17·18
At 80 mesh	22,235	17,530	4705	16·32
At 100 mesh	23,752	19,636	4116	14·28
At 150 mesh	30,315	25,875	4440	15·41
At 200 mesh	29,970	26,748	3222	11·18

forces are uniform and disadvantageously expended upon a mixed aggregation of coarse and fine particles.

Thanks are due to J. Parke Channing, consulting engineer and vice-president of the Miami Copper Co., for permission to publish these data. I am also indebted to B. Britton Gottsberger, general manager, for his kindness in placing at my disposal the metallurgical data of these tests.

The Hardinge ball-mill has also been tested in this plant as a substitute for rolls, for intermediate crushing on ½ in. material. This mill, however, was soon discarded, since it was found that the desired product could only be obtained at too low a capacity, and the consumption of steel balls was too great to be economical."—Robert Franke, *Mining and Scientific Press*; from *Bull. Amer. Inst. of Mining Engineers*, 9th August, 1913, p. 223. (A. McA. J.)

Slime Settlement.—"During an inspection of the chief mining districts of the world, I, being much interested in the subject of slime settlement, made a point of visiting mills where anything new in this direction was being tried or used. For several years considerable attention has been centred on assisting the rate of settlement of fine particles suspended in liquids by means of baffles in the containing vessel. This system has been brought before the public by a number of writers. R. H. Richards, in his book on 'Ore Dressing,' speaks of it as being in use in Mexico.

The underlying principle is that currents are set up between adjacent baffles, the downward current, which is along the upper side of the baffles, causes the slime that settles to slide downward, whereas the upward current, which travels against the underneath side of the baffles, is not strong enough to disturb the settlement. In the United States, one company has designed machines, claiming greatly increased settling capacity per square foot of area, which is theoretically brought about by the use of a large number of baffle-plates set at an angle of 60° across the flow of pulp in the tank. These are said to retard the flow of the light solids and to provide a settling space free from disturbing currents. In the case of these machines, however, it has been clearly demonstrated that no advantage is gained, and I, knowing this, was anxious to study a number of instances of which I had heard and read, where considerable benefit was claimed by the use of baffles set in tanks of different kinds

About 18 months ago a detailed article by B. L. Gardiner, dealing with the subject, appeared in the *Chamber of Mines Journal*, Kalgoorlie, Western Australia, giving the advantages derived from the use of corrugated iron baffles, called accelerators, set in a nest of spitzkasten. Extra efficiency was claimed for the corrugated iron as against ordinary smooth baffles, as the corrugations were said to deflect the flow into narrow streams. This article was copied by a number of the leading technical journals, as the author stated that the method as adopted at the Sons of Gwalia mine was not only a complete success, but that an improvement in the density of the thickened pulp was obtained, varying from 26% to 41%.

I closely investigated this matter when in Western Australia, and was surprised to find that the inclined baffle idea had been completely discarded, owing, so I was informed, to the impossibility of preventing the slime building up on the baffles, and then sliding off in cakes when sufficient had accumulated. This action caused disturbances in the spitzkasten, which naturally retarded settlement. In South Africa, two attempts to use this system of accelerated settlement came under my notice. Both, however, had been discarded. In one case, after months of experimenting, the reason was given, that no benefit could be found, and that, if there was any, it was counteracted by the building up and sliding off of cakes of slime, as in the case in Western Australia The trial in South Africa was made in the following manner: Sheet-iron plates inclined at an angle of 45° were set in a 50 ft. slime-collecting tank, with vertical sides, so that the baffles extended from near the centre to the side. As no success was met with, practically all the plates had been discarded at the time of my visit. Another attempt to work out this problem led to the fitting up of a large number of cone tanks with 45° curved baffles, inclined parallel to the side of the cones. These, however, gave no apparent benefit, and to-day the slime plant is shut down. In my opinion, the principle involved of assisting settlement of fine particles suspended in liquid, by the use of baffles, has given no practical results, notwithstanding the fact that increased rate of settlement can be demonstrated where a glass cylinder containing slime and water is placed at an angle of 45°, as against one standing vertically."—H. N. Spicer, *Mining and Scientific Press*, June 21, 1913, p. 954. (A. McA. J.)

MINING.

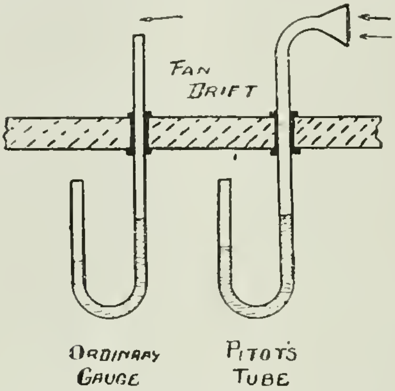
Freiberg Mines to Close.—"Many mining engineers in South Africa who were educated at Freiberg will regret to hear that the mines of that place were to be closed down in September. It is thought probable that one mine will be kept open for the purposes of the Bergakademie and the Bergschule. The Freiberg mines were first opened in 1168, and in the next century they were actively exploited. In 1572 they made their maximum production. In 1791, they supported a population of 5,187 souls. In 1878 there were 38 mines in operation, employing a technical staff of 250 men and about 6,000 miners and labourers. The fall in silver in 1893 affected these mines severely, and since then their production has been steadily dwindling. From beginning to end they have yielded over five million kilograms of silver. The final closing of the famous old mines will be quite an historic event in mining."—"Pyrites," *Sunday Post*, Johannesburg, 13th July, 1913. (J. W.)

The Water-Gauge.—"Much has already been said and written about the water-gauge or manometer and its uses, and yet it seems that many of the persons who habitually use the instrument are uncertain as to whether they use it as it ought to be used.

The manner of its use would seem to be determined by the person using it; but if the purpose for which the manometer is used is clearly understood, there ought to be no doubt whatever as to what its form should be, where it should be placed, and how it should be placed.

The water-gauge is used to measure the difference of pressure between the ends of a ventilating system—that is, the pressure which produces a flow of air through the workings of a mine from the downcast shaft to the upcast shaft. If the velocity of the air passing the water-gauge is low, an ordinary water-gauge may be used; but should the air-current be passing along with a high velocity, the gauge should be modified in shape and placed so that the air-current impinges on the end of the tube, which should be cone-shaped, preferably.

But one might ask, why should the form be determined by the velocity of the air-current? The drift leading from the upcast shaft to the fan may be short and turn abruptly from the shaft. In such a case, experiment will generally show that the air does not pass along with uniform velocity throughout the whole area, and if the end of the water-gauge tube be moved about the drift all over any section of it, it will be found that the difference of pressure varies as the velocity of the air varies, whereas if the modified form of gauge is used in exactly the same way, the water-gauge—*i.e.*, the difference of pressure—remains constant, irrespective of the velocity of the air-current. The ordinary form of gauge cannot, therefore, be satisfactory unless the velocity is low.



When the ordinary gauge is used, the total depression recorded is the sum of the static head produced and the head due to the velocity of the air-current. Since the ventilating current is produced by a difference of pressure between the ends of the airway, it is evident that the true water-gauge can only be obtained by the elimination of the head due to velocity, which is considerable if the velocity is high, but may be neglected if the velocity is low. In the course of making experiments with the two forms of gauges while circulating air-currents of high velocity, the writer has found that the head due to velocity amounted in many cases to 70% or 80% of the total depression, and while passing currents of low velocity—*e.g.*, about 500 ft. per min.—the difference between the readings on the two gauges could not be distinguished, due to the oscillations of the water in the gauges."—*Colliery Guardian*, July 25, 1913, p. 170. (A. R.)

STEEL POINTS FOR CORE DRILLING.—"The recent application of granules of steel instead of diamonds has shown that in many instances they offer advantages over old methods of boring. The principal advantage of steel granules is in the matter of cost, facility in obtaining bores of large diameter, and boring very hard rocks—such as corundum. It is with hard broken rock that this method of drilling displays its superiority over the diamond-drill. The difficulty with steel-pointed drills appears when the bore-holes are far removed from verticality. Under favourable conditions the granules have penetrated 52 ft. in 24 hrs., but generally, in ordinary ground and moderate depth, the daily advance is from 16 ft. to 20 ft. This mode of boring seems destined to be extensively employed."—M. L. MARTEL, *Mining and Scientific Press*, from *Bulletin de l'Industrie Minerale*, August 2, 1913, p. 194. (A. McA. J.)

EFFICIENCY IN MACHINE DRILLING.—"This article deals with 'what the drill runner is doing.' The figures have been compiled after many tests and careful timing, eliminating all delays from accidents or power failure. Many mine engineers are 'hacking at the branches' of the expense account in mining, while the author is digging at the root, to locate every little rootlet that is sucking up costs. Having found the rootlets, his plan is to lop them off when possible, or limit their supply of nourishment, *e.g.*, perhaps cheaper men than 'drill runners' could do the setting up and tearing down, etc. 'The efficiency of the man is not to be measured by his muscular or brute ability—but by the intelligent direction of the tools and equipment by which he is supplied.'

The Drill Runner.—Mining laws are almost universally limiting the hours of labour to eight as a working day. Eight hours of 60 minutes each gives a total of 480 working minutes for a shift's pay. For convenience, we will assume that the average miner draws 1 cent a minute, which would be $4.80 a shift. As a matter of fact, in Nevada a machine drill runner draws $4.50 a day in dry mines, and 50 c. added for shaft work, and 50 c. added if mine is wet. Shovellers and common labour draw $4 a day, while carpenters, timber framers, blacksmiths, machinists, hoist runners, etc., draw from $5 to $7 a day; and figuring that the men are hoisted and lowered 'on company time,' it is not out of the way to compute labour as costing 1 cent a minute. In other localities with different wage scale, the figures would vary the cost accordingly. The efficiency of 'the man' is not to be measured by his muscular or brute ability, but by the intelligent direction of the tools and equipment with which he is supplied. As an example, we will hold a 'stop watch' on a drill runner for a shift, and for many shifts on many runners, and we get the following figures, on an 8 hr. shift at 1 cent a minute:—

Work.	Minutes.	Cost.	Per cent of time.
Setting up	82	$0.82	17
Shifting machines	66	.66	14
Changing steels	75	.75	15
Morning and noon starting	24	.24	5
Oiling, cleaning, etc. ...	15	.15	3
Getting steel and water ...	15	.15	3
Tearing down and blasting	45	.45	9
Actual drilling time ...	158	1.58	34
Total	480	$4.80	100

Lost lights, stuck drills, etc., are variable, and can only be estimated as from 1% to 3% of the time. Repairing or changing a leaking hose, or tightening fittings on pipe line, etc., are variable, and occur at such irregular intervals as to be only computable at the end of the year or month, as many shifts run through without a change of that kind. If mine is operating on a continuous 3-shift basis, there is an additional deduction of 5% for meals on shift, and 6% for blowing smoke after blasting—bringing the actual drilling time down to 23%. No fault can be found with the workman; he is working, and doing all he can, with his muscles, but there is something wrong with the 'engineering headpiece,' when a machine can deliver only one-third of its efficiency. Figure this back to the motor that drives your compressor, or to the coal bill, and you'll find that the compressor is making enough air to run three drills at the same rate of efficiency, and except for the automatic unloader and what it saves in the compression, the power bill is 'three times as big as the actual work performed calls for.' Some mines have put on a blasting crew, to save 'shift-time' in the drilling, and thereby reduce the 'dead-time' of blasting, but they lose the tearing down time just the same. This increases the actual drilling time a little. Compare the old time, single-jack work with the machine driller, and note that he draws 15% to 40% less wages than the above figures, and works harder, with more brute muscular labour, and can give about 80% of his efficiency upon the drill. It is evident that the easier the labour can be made for the workman the more he can accomplish by intelligent direction of the improved tools. Moreover, the intelligent direction of mechanical tools draws bigger wages—rightly so, because it accomplishes more results for the employer. 'How much would a man be worth who could get 300 min. actual drilling time, out of the machine drill,' even if the muscular exertions were less fatiguing? Think of the 'overhead costs,' the 'power costs,' 'top labour,' etc., that would not increase though the progress be doubled.

One of the greatest impediments to the more rapid displacement of brutalising physical labours of man, by machinery, seems to be the fear of the workman that his work will be done by machinery and he will 'lose his job.' He seldom stops to think that 'it will require a man to direct the machine,' and that it will be less fatiguing and higher-paid labour than what he is now doing. Did the automobiles put the teamsters out of work? The fact is there are 20 auto drivers at better pay to-day for every teamster whose horses were displaced, and a million mechanics secured work at good pay in making and repairing automobiles. Efficient machinery is not 'labour-saving machinery,' but 'labour-making machinery,' and the more machinery running the more employes on the payroll, and the better the wages. Hand labourers and machine operators are not to be compared in wage, nor in fatiguing exertions. It is an absolute fallacy to attempt to 'cut expenses' by cutting wages. It increases the cost per ton, or per article, and decreases profit, every time. But if you can 'increase the efficiencies,' by 'higher pay,' for more intelligent application and direction of mechanical equipment the costs can be lowered. The bonus system is feared by the labourer, thinking that if he does an extra amount of work that the minimum will be raised, that the employer is trying to squeeze every ounce of energy out of him within the 8 hr. limit. In some cases, perhaps, a brutal, inefficient boss may try to make progress at the expense of the muscular labour of his employees, but sooner or later a competent, efficient boss will replace that fellow. Any sensible employer of labour would rather pay a drill runner $6 a day than $4.80 if the drill runner could make the drill run 42% of the time instead of 34%, even if the drill runner sat on a box and rolled cigarettes half of the time, 'so long as the drill kept cutting.' Think of the saving on the power bill and the other overhead expenses if the drill could be kept cutting just 40 min. longer. The Moral: Not harder labour, but easier and more intelligently-directed mechanical energy."—Letson Balliet, *Mining and Engineering World*, August 16, 1913, p. 294. (A. R.)

The New Standard Base at Johannesburg.—"In March, 1911, the Council of the Institute drew the attention of the Surveyor-General to the proposal of the Johannesburg Town Council to build an Art Gallery on the site of the standard laid down by Colonel Sir Wm. Morris in Joubert Park. The Surveyor-General communicated with the Town Council, with the result that it was decided to establish a new standard, and Mr. Gilfillan instructed Messrs. W. K. Tucker and W. C. van der Sterr to select a suitable site and to transfer the standard.

After inspecting several sites it was decided that the north corridor of the new Law Courts in Von Brandis Square was the most suitable, and permission was obtained from the Public Works Department to lay down the line there. The corridor itself is 280 ft. in length, terminating at each end in a tiled vestibule 19 ft. in length. This gives quite a convenient length for laying down a line of 300 Cape feet.

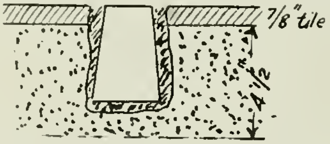
The following specification for the laying of the blocks (which has been kindly lent us by Mr. Tucker) will be of interest:—

Specification for Laying Down Permanent Marks for a Standard Base at the New Law Courts, Johannesburg.

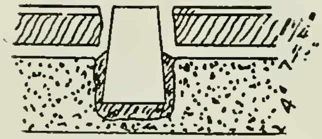
The base is to be laid down in the back corridor of the New Law Courts.

Five marks are to be fixed, at 0, 100 English feet, and at 100, 150, and 300 Cape feet. The two terminal points will be in the tiled vestibules at the ends of the corridor, and the remaining marks will be in the corridor. The marks are to be brass blocks 3 in. × 1½ in. at top, 3½ in. × 2 in. at bottom, and 2½ in. deep for the terminals, and 3 in. × 1½ in. at top, 4 in. × 2½ in. at bottom, and 4 in. deep for the intermediate marks.

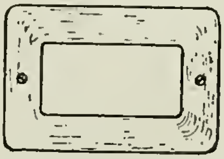
The terminal marks are to be let into the concrete of the floor, so that the top of the block is level with the tiled surface, and to be bedded and grouted in with cement.

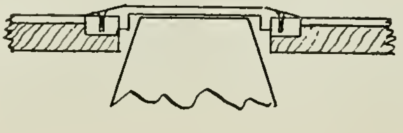


In the corridor the flooring is 1¼ in. thick, and is laid on a concrete floor 4 in. thick, with a ½ in. space between. The wood floor is to be cut through and the brass block let into the concrete to such depth as to bring the top of the block level with the surface of the linoleum, and the block is to be bedded and grouted in with cement.

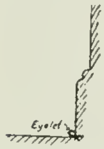


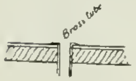
All the blocks are to be covered by brass plates ¼ in. thick, 6 in. × 4 in., recessed where they cover

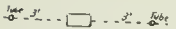




the blocks, to a depth of 3/32 in., so that they shall not come in contact with the blocks. The edges of these covers to be bevelled back 1 in. from each edge, so as to offer no obstruction to foot traffic, and the corners to be rounded. The covers are to be secured to the floor by 3/16 in. brass screws, with counter sunk heads, and screwed into bushes let into the flooring or tiling, as the case may be.







For the purpose of straining the measures under test, a small brass eyelet made of ¼ in. metal is to be let 3 in. into the wall at each end at floor level, and at each of the other two blocks a brass tube of ⅜ in. bore, with a flange, is to be let into the flooring 3 in. from the blocks on each side. All these tubes and eyelets are to be set truly in the line of the blocks. The tubes are to be of ⅛ in. metal, and the holes bored in the floor for their reception are to be no larger than absolutely necessary, so that the tubes fit tightly.

It was decided to transfer the length from the 150 ft. base in the Government Buildings at Pretoria, which was measured by Col. Sir Wm. Morris, with the following results :—

1st part	...	49·99740	Cape feet
2nd part	...	100·00298	,,
making a total of		150·00038	,,

The method of transference of the standard length, as defined by the Pretoria base referred to above, to the Law Courts base was as follows :—

Two steel bands kept specially for testing purposes were used.

One has a length of 150 Cape feet exclusive of terminals, and a breadth of ¾ in., made by Chesterman, marked at zero, 100 ft. and 150 ft. by finely cut marks.

The other has a length of 300 Cape feet exclusive of terminals, and a breadth of ⅛ in., made by Keuffel & Esser of New York.

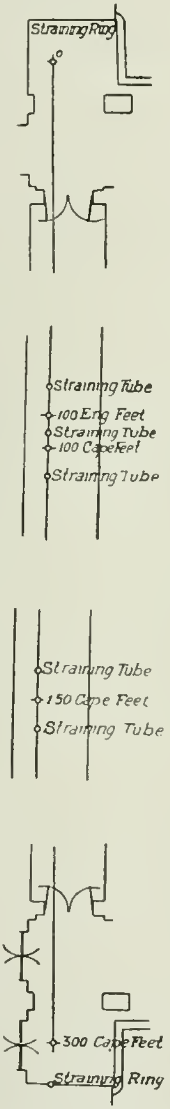
Operations at Pretoria Government Buildings Base.—The Chesterman band was first laid over the base under a tension of 15 lb. temperature 57° F., and the marks read off on the tape by means of a finely divided scale and a microscope, the zero mark of the band being at the south terminal of the base. The readings obtained were :—

0·0—0·0050 ; 100·0—0·0015 ; 150·00—0·0015.

The Keuffel & Esser band was next laid over the base and strained under a tension of 10 lb., the temperature remaining as before at 57° F. Marks were cut on the band exactly corresponding with the marks of the base. The 150 feet mark on the band was then made coincident with the zero mark of the base, the same tension applied as before, and the temperature noted to be constant at 57° F. The second half of this band was then marked in the same manner as the first. The band was thus marked with a total length at the tension and temperature mentioned, exactly twice that of the base.

Operations at New Law Courts Base.—The upper faces of the brass blocks for carrying the marks were covered a few days previously with thin white gummed paper, to enable temporary marks to be made with a hard pencil during the process of the measurement.

The Chesterman band was first applied with the same tension that was used at Pretoria, viz :—15 lb., and marks were made at 0·0, 100 ft., 150 ft., the temperature being 59° F. The band was then lifted, and the zero mark on it applied to the 150 ft. temporary mark, while the 150 ft. mark on the band under the same tension and tension and temperature as before, was reproduced on the paper on the block at the 300 ft. end of the base. The temporary marks thus obtained were then adjusted for the difference in temperature —57° F., under which the band was read at Pretoria, and 59° F., under which it was used at the New Law Courts.



The Keuffel & Esser band was next applied, the marks thereon having been made as described, to coincide with the marks of the Pretoria base. The zero mark having been carefully and accurately adjusted to the zero mark made with the Chesterman band on the zero block of the base, and the 10 lb. tension having been applied to the band laid from end to end of the base, the position of the various marks on the blocks at 100 ft., 150 ft., and 300 ft., with reference to the corresponding marks on the band as made at the Pretoria base, were re-noted ; and, finally, the adjustments for difference in length as found by Col. Morris, and of temperature, viz :—57° F. and 59° F—, were applied. The result was a definition of the marks so close to those arrived at by the Chesterman band that the difference was not capable of appreciation.

By means of a straight edge and a sharp knife, the lines on the paper marking the various base lengths was transferred to the metal blocks by cutting through the paper well into the metal. The mark for the standard length of 100 English feet was established by measuring 39·6 in. back from the 100 Cape feet — mark with a 5 ft. steel straight edge graduated to inches and hundredths of an inch.

This standard is as nearly as possible an ideal one. The two terminal points are set in concrete floors, surrounded by stone walls founded on solid rock. The intermediate marks are also in a concrete floor, surrounded by walls, and broken by two arches, so it is practically impossible for any movement or cracking of the floor in any part to be transmitted. The concrete and walls have had about three years to settle before the marks were laid down. Being inside a building surrounded by flagged pavements there cannot well be any settlement or movement caused by soakage of water or great changes of temperature.

The temperature in the corridor is very even. The windows are all in the south wall, so that there are no patches of sunshine and shade at any time. Outside business hours the corridor is quite free from visitors.

A specification has been drawn up for a special apparatus for accurately laying down such a line, and it is proposed to finally check this standard by its means. As this apparatus has been sent to the National Physical Laboratory for report, no details are yet available for publication."—*Land Surveyors' Journal* (Transvaal), August, 1913, p. 191. (A. R.)

MISCELLANEOUS.

Metering and Recording the Flow of Fluids—New Meters and Recorders of the Flow of Liquids through Venturi Tubes, Orifices, or Conduits, Based on the Integration of the Velocity Head.—"The flow of water and other liquids as well as gases follows the law of gravity. For steady flow this may be represented by the expression: 'Velocity is proportional to the square root of the head-producing velocity,' or, algebraically,

$$V = C\sqrt{H}$$

in which

V = the velocity, in any convenient unit,

H = the head producing such velocity,

and

C = a constant depending on the units of measurement and the character of space through which the fluid flows.

V may be expressed in feet per second, miles per hour, etc., and H, the effective head, may be expressed in feet or in pounds per square inch, etc.

The space may be a pipe of any length, a submerged orifice, an open inclined channel, a Venturi tube, or a nozzle. It is seen that this is the form of the simplest equation of a parabola $y^2 = cx$.

For practical purposes, the ideal form of meter would be a device, connected by small tubes with adjacent parts of a pipe through which fluid is flowing, which would measure automatically the square root of the head or pressure producing the velocity. Such a device would offer no resistance or obstruction to the flow through the pipe. It has been impossible heretofore to achieve this ideal, because the forces due to velocity head are so minute as to be difficult, if not impossible, to measure or integrate when the velocity is low.

Theoretically, the Venturi tube is a contraction in the pipe, formed so as to cause small frictional resistance to flow; and it is remarkable how slight this contraction may be in order to produce the same difference in velocity head as that caused by Pitot tubes without contraction in the pipe. In fact, the diameter of the contracted portion need not differ from that of the full section by more than 17·7% in order to produce the full effect of two Pitot tubes, one pointing up and one pointing down stream, with their openings in the centre of the pipe, or at the points of maximum velocity.

In practice, however, the Venturi tube is composed of a short pipe continuously connected with the main pipe by a conical section pointing down stream, the convergence of these two sections being dependent on experimental determinations, the object being to obtain the highest flow with the least frictional or unrecoverable loss of head through the entire Venturi tube. These angles of convergence and divergence have been determined by Venturi and, in more modern times, by Weisbach, Eytelwein, Francis, Smith, Herschel, and possibly others, but at any rate it is pretty well settled that they should be somewhere between fairly wide limits, beyond which the results are certainly not as good.

However, they may be relatively the same for all Venturi tubes, whether the contraction is nearly as large as the pipe or very much smaller, the difference being that in the former case the entire Venturi tube would be short and the latter long.

Of course, the purpose of a Venturi tube, the contracted section of which is small as compared to the full bore of the pipe, is to exaggerate the velocity head indications so that they can be observed and measured, especially for low flow.

To illustrate, consider the simple formula of a standard Venturi tube, having a coefficient of velocity = 0·977:

$$V = 7{\cdot}84\left(\frac{H}{(D/d)^4 - 1}\right)^{\frac{1}{2}}$$

in which the only two variables are V and H

V = the velocity through the full section, in feet per second;

H = the difference in piezometric heads between the up-stream section and the contracted section, in feet;

D = the diameter (or radius) of the full section;

d = the diameter (or radius) of the contracted section.

This formula may be written:

$$H = 0{\cdot}0163\,[(D/d)^4 - 1]$$

Now, assume a velocity in the main as low as 0·25 ft. per second. Then the formula may be written

$$(D/d)^4 = 1 + 982H$$

and assume 0·01 ft. to be as low as it is possible to indicate H; then

$$D/d = \sqrt[4]{10{\cdot}82} = 1{\cdot}86,$$

which means that when the velocity through the main is as low as 0·25 ft. per second, if it is desired to get an indication of the velocity head as great as 0·01 ft., the contracted section of the Venturi tube must be less than 54% as large in diameter as the main.

However, with a tube having ratios of diameters equal to 3 to 1, $H = 0{\cdot}0815$ ft. for $V = 0{\cdot}25$ ft. per second. Even this result cannot be obtained with the best apparatus formerly in use, unless one of the essential features of the Venturi meter is seriously sacrificed, namely, the measurement of high velocities.

The best types of Venturi meter registers have formerly depended for their registration on some form of cam having a parabolic curve, fashioned around a drum or revolving about a centre. Near zero, or the origin, the curve changes in direction rapidly, and at zero crosses one of the axis of reference at right angles and becomes parallel to the other.

It is probably this feature, more than any other, which has limited the minimum velocity measurable by a Venturi meter; and though the measurement of low velocities is attainable by a high ratio of tube diameters, there is thus encountered the other difficulty of excessive frictional loss through the Venturi tube for comparatively high velocities.

If it is even intended to measure velocities as high as 10 ft. per second, which is occasionally desirable, the frictional head through a Venturi tube having a ratio of 3 to 1 in diameters, or 9 to 1 in areas, would be 17 ft., obtainable from the formula $Hf = 0{\cdot}0021\,V^2_2$, in which Hf = the frictional head in feet and V_2 = the velocity through throat, in feet per second.

Besides that, in any form of register depending on mercury as the medium of indicating the velocity head, in order to cover ranges of velocity of from 10 ft. per second down to zero, the apparatus would have to be more than 11 ft. high. In a tube having a ratio of 2 to 1 in diameters, or 4 to 1 in areas, with a velocity of 10 ft. per second through the main, the frictional head would be only 3·36 ft. This is still a material loss, but a velocity of 10 ft. per second is usually high.

It must be remembered that, as far as metering a fluid is concerned, any form of orifice or contraction in a pipe would be just as good as a Venturi tube. It is also highly probable that the coefficient of flow would remain as nearly constant through just as wide a range of velocities; but the great advantage of a Venturi tube is in its small frictional resistance to flowing fluid and in this respect it is the best shape yet devised.

The writer will now describe a form of meter register particularly adaptable to the registration of flows through any form of orifice, such as by the use of Pitot tubes and Venturi tubes of small ratios and consequently minimum frictional heads. The fundamental principles were discovered by the writer in 1904 and were finally developed into the best practical working form in 1909. The basic idea is a float, shaped so that, when acted on by the head due to the velocity of flowing liquid, its movement will be proportional to that velocity.

The device is novel in form, and covers a principle in hydraulics, which, although rational and perfectly obvious on analysis, the writer believes has not heretofore been recognised. This may be stated as follows:

If a hollow body of any shape, with open bottom (Fig. 1.), floats freely in a liquid having the same specific gravity and touches another liquid of greater specific gravity, and if the relative pressures without and within this hollow body are changed so as to make the internal pressure less than the external pressure, then the body will sink into the heavy liquid, and the amount of displacement will be exactly equal to the quantity of the heavier liquid which will pass up into the interior of the body and above the level of the heavier liquid outside of the body.

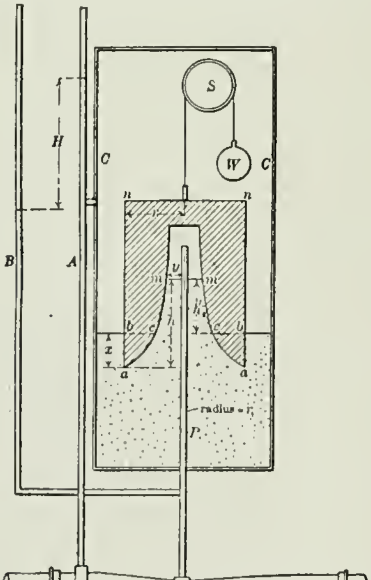


Fig. 1.—Meter Registering the Flow of Liquids through Venturi Tubes.

Let the hollow body, or float, be represented by *n n a a*, Fig. I.; assume it to be a figure of revolution, and call the total volume of the heavy liquid, *a a m m*, within the float *M*.

The principle may be understood by a consideration of the laws of equilibrium—the algebraic sum of the vertical forces, must be equal to zero; the sum of the horizontal forces, and the sum of the movements must all be equal to zero.

We need consider only the first condition as affecting the problem. At the plane *b c c b* the heavier liquid is subject to the same unit pressure within and without the hollow body, but the resultant of external forces acting downward on the hollow body is measured by the weight of the heavier liquid, *m m c c*, and the total upward force acting on the body is the displacement of the annular portion of the floating body, *b c a b c a*.

For equilibrium, the algebraic sum of these two forces must be equal to zero, and, as one is positive and the other negative, they must be equal in amount; but, if that is the case, the entire quantity of liquid equivalent to the displacement of the hollow body, and more, must be found in the interior of the hollow body, and hence the outer level of the surface of the heavy liquid does not change with variations of pressure outside and inside the float, as will be very easily understood.*

If there is a difference of pressure, *H*, then the float will sink into the heavier liquid an amount represented by the annular volume, *a c b*, which will be called *V*; and the net volume of liquid, *m m c c*, will also be equal to *V*, and h_2 will be proportional to *H*. As before stated, the level *b b* will always be the same, no matter what the values of *H*, h_2, and *V* may be. Let

x = the vertical movement of the float from a given datum;
H = the Venturi head;
h_2 = the corresponding mercury head;
v = the velocity through the Venturi tube;
y = the ordinate or variable radius of the inner portion of the float;
h = the corresponding abscissae, measured from the bottom of the float;
r = the constant radius of the outer cylindrical portion of the float;
M = the total volume, *a a m m*, of the heavy liquid within the float;
V_0 = the volume of heavy liquid above *cc*; and
V_1 = the volume of the float below *bc cb*.

Assume the radius of the pipe, *P*, to be small and negligible. Take, by hypothesis, $v \propto x$, also for unity the coefficient, $x^2 = h_2$, for $v \propto \sqrt{Ha\sqrt{h_2}}$.

Therefore, the cross-section of the float must conform to the following equation:†

$$y^2 = \frac{r^2}{2\sqrt{h + \frac{1}{4}}}. \qquad (1)$$

This is an equation of the fifth degree, as between the two variables, *h* and *y*. No attempt has been made to discuss this equation exhaustively or plot all the loci, for, though this would be interesting from a purely mathematical standpoint, it is evident that only one locus will satisfy the physical conditions wherein both *h* and *y* have positive, real values.

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*If the float were not free to move vertically, that is, if it were restricted in any way, as with a variable lever arm, counterweight or spring—then the movements of the liquids would not follow the law stated, but, as the difference of pressure became greater the surface level of the heavy liquid would fall below a fixed level on the outside and rise above a fixed level on the inside.

†The proof of this equation which is given in the paper is here omitted.

For $h=0$, $y=r$, and for $h=\infty$, $y=0$, thus it is seen that there are no values of h between 0 and ∞ which give infinite or imaginary values of y. As a consequence, the integrating apparatus operated by the sheave, S, is theoretically accurate from zero up to any practical limit that may be desired.

The shape of the float is such that its vertical movement is proportional to the square root of h_2 or H (the mercury and water Venturi heads, respectively); and as the velocity through the Venturi tube is also proportional to the square root of H, the vertical movement is directly proportional to the velocity or quantity of water passing through the Venturi tube.

The same principle will hold true if the Venturi tube is replaced by any form of orifice, or if the pipes A and B be connected with Pitot tubes in a main; for, in any of these cases, the H values represent velocity heads, the same as they do for Venturi tubes, and the velocity heads are always proportional to the square of the velocity producing the head.

It is perfectly practical to have x less or greater than h_2, for, if we assume $cx^2=h_2$, equation (1) becomes:

$$y^2=\frac{r^2}{2\sqrt{c}\sqrt{h+c/4}} \qquad (2)$$

which is the more general form, in which we can give to c any value, consistent with practical limitations. For general use, however, equation (1) is satisfactory, particularly as there is no value of c which increases or diminishes the accuracy of the apparatus.

In these calculations, the radius of the pipe, P, within the float, is assumed to be so small that it does not affect the results. Even if no correction is made for this volume, the error at low velocities would be negligible, and at maximum velocities, it would be less than half of 1%, provided the pipe is made as small as possible. Its effect is to raise slightly the level of the heavy liquid, both inside and outside the float, by the same amount and aggregating the volume of displacement of the pipe P; and, where it is desired to make this inside pipe P of an appreciably large outside diameter, allowance must be made for its disturbing influence.

The final rationally derived formulas for that purpose are as follows:

$$x^2=h_2$$

$$h=x+x^2+\frac{r_1^2x^2}{y^2+R^2-r_1^2-r^2}$$

$$x=\frac{(r^2-y^2)(y^2+R^2-r_1^2-r^2)}{2y^2(y^2+R^2-r_1^2-r_2)-2r_1^2(r^2-y^2)}$$

in which the only variables are x and y.

h_2 = the mercury rise as before;

x = the vertical movement of the float;

and

y = the variable radius of the interior.

The constants are:

R = radius of cylinder containing float;

r = radius of outer surface of float;

and

r_1 = radius of pipe P.

To use these formulas practically, it is necessary to tabulate x and y, h_2 and finally h; h representing the abscissae, as before, and y the ordinates. The necessity of this complication can be avoided by enlarging the outer cylinder at the location of the surface of the mercury.

Although the writer has devised many other forms and modifications, the form of float illustrated in Fig. I., and covered by equations (1) and (2), is believed to be by far the most practical, because it provides for maximum increments of volume and consequent accuracy at minimum velocities; and, with a float of a given radius, the minimum clearances are possible with the smallest cylindrical casing; also there are several other practical advantages of importance.

One of the first difficulties to overcome was to prevent air from accumulating at the upper interior of the float in Fig. I. Such an accumulation, of course, would lighten the float and therefore vitiate the results. Two check-valves opening upward are placed in the cap of the float. These are opened automatically when the velocity head reaches the maximum, and they can be opened at will, by a hand device, from the outside of the bottom of the pressure chamber, without otherwise dismantling the meter. This same device also automatically prevents the loss of mercury when the flow reaches an amount beyond the maximum capacity of the meter register, as it equalises the pressure within and without the float.

An additional device for this purpose has been designed to prevent loss of mercury due to a sudden break of either of the pipes leading from the meter register to the Venturi tube, but these are safety devices of not unusual novelty, and, therefore, a detailed description of them is unnecessary.

Fig. II. shows an application of this meter register

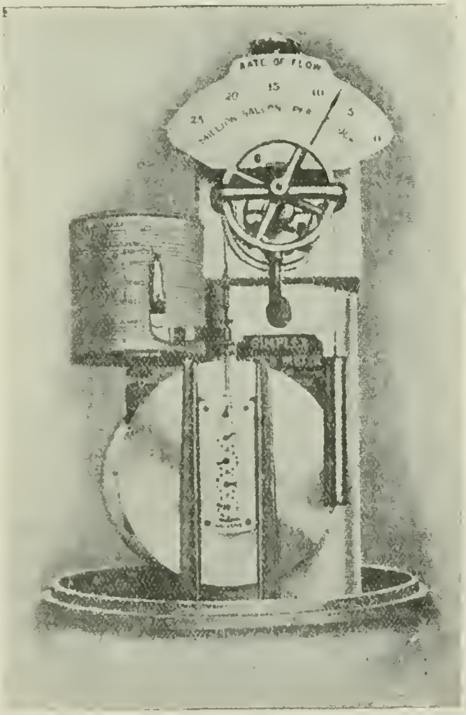

Fig. II.—Tap of Meter Register with Glass Dome Removed.

The float is made of hard rubber to conform to a metal template; but, to make sure that it is correct and will perform according to the theory on which it is designed, each float is tested in a regular working machine by applying, under pressure, piezometric heads corresponding to the velocity heads of the Venturi tube or Pitot tube with which it is to be used, and noting from a standard rate-of-flow dial the corresponding velocities, which are compared with the theoretical velocities.

For heads, *H*, greater than $\frac{1}{3}$ ft. the coefficient of velocity through a standard Venturi tube is so nearly constant as to secure results within 2% of the actual. However, with the apparatus above described, Venturi heads, *H*, of less than 0·01 ft. are readily integrated. The velocity coefficient through the Venturi tube, however, is materially different, amounting to as much as 7% at low velocities, and the instrument is modified accordingly so as to produce accurate results down to less than 0·01 ft.

The data for this modification, as well as the method of making the correction, are intended to be the subject of a later paper.

Leading up to the final development of the apparatus illustrated in Fig. I., it will be interesting to consider two of the earlier forms. Fig. III. shows an extremely simple apparatus, which, at first sight, would appear to be superior to the device shown in Fig. I. *P* is a piston, *F* is a shaped float, *S* is a sheave, and *W* is a displacement device, all within a casing. The upper portion of this casing is connected with a pipe from the full section of the Venturi tube, and the portion below the piston *P* is connected with a pipe, leading to the throat of the Venturi tube. The differential pressure or Venturi head, *H*, acts on the piston and forces the float, *F*, down into the mercury.

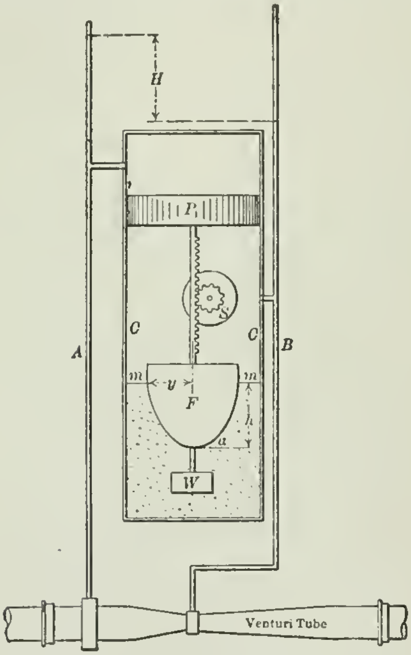

Fig. III.—Original Simple Apparatus.

As before, the problem is to shape the float so that its vertical movement from a fixed line of reference will be proportional to the square root of *H*. The displacement device, *W*, is such that when there is no flow in the Venturi tube and the apparatus is in equilibrium, the bottom of the float, *a*, will be tangent to the upper surface of the mercury. The level of the mercury, *m m*, will change, of course, as the float descends. The equation of the float, though rather more difficult to deduce, is as follows :—

$$y^2 = e - \frac{f}{\sqrt{h+g}}$$

e, *f*, and *g* being constants depending on the size of the piston, *P*, the assumed movement of the float, *F*, with the given Venturi head, *H*, and the relative weights of mercury and water.

The only radical difference in form between this formula and the one corresponding to Fig. I. is in the constant *e* before the fraction. The curve, however, is still one of the fifth degree.

The objection to this apparatus is the friction of the piston *P*. This was recognised immediately, and, to overcome it, a diaphragm of an accordian or some very flexible type was substituted for the piston ; this, however, introduced difficulties caused by the elasticity of the diaphragm and the necessity of modifying the shape of the float to provide for these variations, as the diaphragm could never be depended on to remain in a permanent position.

Another form is shown in Fig. IV. This is an entirely different principle, but the treatment of the problem is analogous to that of Fig. I.

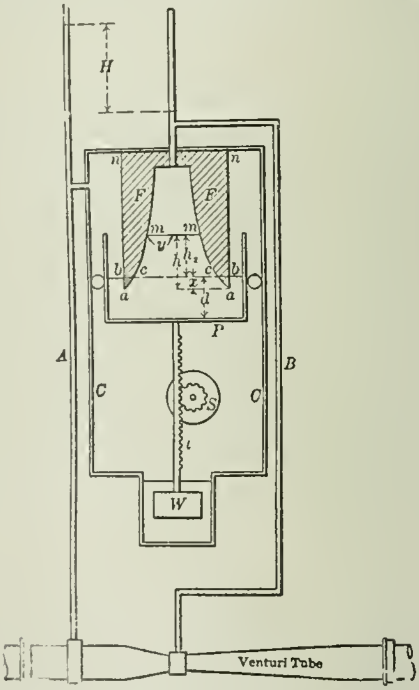

Fig. IV.—Another Form of Simple Apparatus.

C represents a casing within which is contained a fixed displacing device *F* and a movable cylindrical vessel *P* containing, say, mercury buoyed up by the displacing volume *W*. The pipe leading from the full section of the Venturi tube connects as shown above the vessel, and the pipe leading from the throat of the Venturi tube connects within the displacing device *F*.

The differential pressure, or Venturi head, *H*, forces mercury from the outside of the displacing device *F* to the inside ; but for equilibrium, the depth of the mercury in the vessel *P* must always be the same ; that is, *d* must always be constant.

With these data it is found that the equation representing the inner curve of the displacing device is exactly the same as that of the float in Fig. I., namely, equation (1) given before.

This device, in some respects, is superior to any of the others considered, and, in fact, it was the one of which a complete working model was first made and tested, and the results were good. The only objection is the large quantity of mercury required for the displacing device, *W*.

It is obvious that the vertical movement of the cylindrical vessel, *P*, may be guided by rollers, as shown, or in some other practical manner. The cross-section of the rod, *l*, need not be sufficiently large to make its variable displacement affect the results appreciably.

However, the form in Fig. I. is that which has been adopted as the most practical; and of all the twelve other devices, each of which has been developed by the writer to a working design, that shown in Fig. IV., next to that in Fig. I., is the most meritorious.

In all cases the registering mechanism outside of the casing may be exactly the same as shown. It comprises a rate-of-flow dial, a chart device to record the rate of flow, and a total-flow dial. The first requires only a graduated dial and a hand actuated by the shaft of the sheave, *S*. The second requires a pen reciprocating along the surface of a revolving figure having straight-line elements, such as a cylinder. The third requires a dial having hands which always revolve at a speed proportional to the angular position of the sheave, *S*.

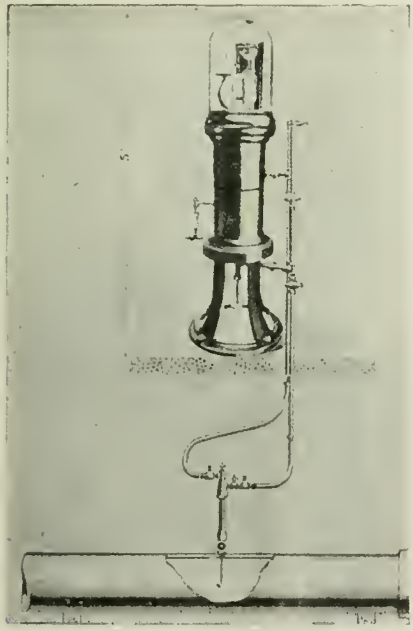

Fig. V.—Stationary Meter with Pitot Tube.

This is readily accomplished by hanging from the sheave, *S*, the dial which is connected with and operated by a traction wheel, the latter being rotated by a constantly revolving disc. When the sheave, *S*, sets at zero, the traction wheel stands at the centre of the disc, and, therefore, has no rotating motion. As the flow increases from zero and the sheave, *S*, revolves, the traction wheel moves away from the centre of the disc and therefore revolves at a speed proportional to its resistance from the centre of the disc and to the rate of flow through the Venturi tube.

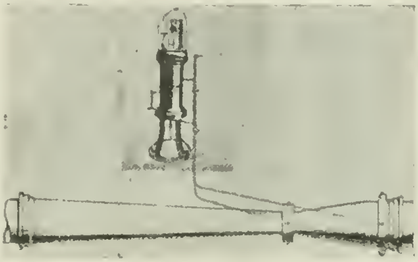

Fig. VI.—Stationary Meter with Venturi Tube.

There are other equally simple means of actuating the total-flow dial, but they are all on the same principle, and have been in use for several decades. The revolving flat disc is probably the oldest, but the revolving cone rivals it in antiquity. The Nicholson ship's log is a rather recent application of the latter device. It will be seen from Fig. II. that the chart-recording cylinder and the disc are operated by the same clockwork.

Where it is intended to meter a liquid of a specific gravity different from that for which this meter is adjusted, as, for instance, if the meter has been adjusted for measuring water at 50° F., and it is desired to measure water having a temperature of 200° F., the results can be corrected according to the following principle:

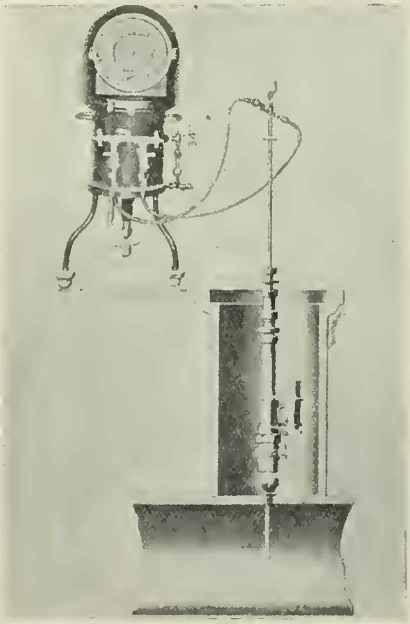

Fig. VII.—Portable Pitot Recorder for Measuring and Recording Flow or Leakage of Water Mains, Flumes, Penstocks, etc.

In a fluid passing through a Venturi tube or orifice, the weight is directly, and the volume is inversely, proportional to the square root of the unit weight of that fluid, for:

Letting

W = the weight of fluid,

Q = the volume of the fluid,

H = the Venturi head,

and

w = the unit weight of fluid;

then

$$Q \propto \sqrt{H}$$

$$H \propto \frac{1}{w}$$

$$W \propto wQ$$

From these equations we obtain

$$Q \propto \frac{1}{\sqrt{w}}$$

and

$$W \propto \sqrt{w}$$

The latter two equations are the stated relations.

TABLE I.—Test of a Meter Register Based on Venturi Heads. Float No. 46.

C	V	V_1	D	P
0·010	0·195	0·194	+0·001	+0·52
0·013	0·226	0·225	+0·001	+0·45
0·023	0·300	0·298	+0·002	+0·67
0·036	0·381	0·379	+0·002	+0·53
0·054	0·470	0·467	+0·003	+0·64
0·078	0·571	0·562	+0·009	+1·60
0·112	0·677	0·675	+0·002	+0·30
0·154	0·800	0·797	+0·003	+0·38
0·208	0·931	0·930	+0·001	+0·11
0·279	1·079	1·080	−0·001	−0·09
0·368	1·240	1·210	0·000	0·00
0·487	1·429	1·428	0·001	0·00
0·638	1·635	1·635	0·000	0·00
0·833	1·872	1·874	−0·002	−0·11
1·092	2·150	2·133	+0·007	+0·33
1·426	2·460	2·460	0·000	0·00
1·873	2·820	2·825	−0·005	−0·18
2·455	3·250	3·235	+0·015	+0·47
3·250	3·750	3·720	+0·030	+0·81
4·320	4·321	4·280	+0·041	+0·96
4·830	4·570	4·540	+0·030	+0·66
5·800	5·010	4·970	+0·040	+0·81
7·270	5·638	5·600	+0·038	+0·68

Table I. contains data relating to a typical test* of meter registers constructed on the principle described herein, in which

C = the Venturi head as measured, in feet;

V = the theoretical velocity through the full section of the Venturi tube, in feet per second;

V_1 = the velocity as indicated by the meter register;

D = the differences:

and

P = the percentages of error.

This meter register is in use in several important cases for recording and integrating the flow measured by Pitot tubes, and has also recently been adapted in portable form utilising Pitot tubes for city main leakage survey purposes. This form is shown in Fig. VII. Quite a number of the instruments are already in service. (A typical test of one of these recorders is given in the paper, but not reproduced here.)"—J. W. LEDOUX.—*Metallurgical and Chemical Engineering*, July, 1913, p. 403. (J. A. W.)

DYNAMITE IN AGRICULTURE.—"In order to bring under cultivation virgin soils which are not easily attacked by the plough, American farmers sometimes use dynamite. They make blast-holes 13 ft. to 23 ft. apart and 30 in. to 40 in. deep, at the bottom of which they place dynamite cartridges containing 15% to 20% of nitro-glycerine and weighing 5 oz. to 8¾ oz.; the holes are then filled with sand or clay, and well rammed. Each cartridge is provided with a fulminate cap and match, an extremity of which protrudes a few inches above the soil. The cartridges are fired either by lighting the matches direct or by electricity. The explosion raises the ground, and breaks it up sufficiently to allow it to be easily ploughed. The cost of the operation is reckoned at £2 8s. to £4 per acre. In order to form an opinion on the utility of this process, new to Switzerland, the Federal Institution of Agricultural Chemistry at Lausanne made, in November, 1912, some experiments on the property of Colonel Ribordy at Planisse sur Saint Léonard (Valais.) Some land intended for a plantation of fruit-trees was prepared by exploding at the spot to be occupied by each tree a cartridge of about 8¾ oz. of 'gamsite,' a safety explosive manufactured by the Gamsen (Valais) works, and containing about 24% of nitro-glycerine. This operation prepares the ground very well for planting young trees. The explosion of the charge, placed at a depth of 40 in., raises and breaks up a volume of from 35 to 52 cu. ft. of earth in the form of an inverted cone, the base of which measures on the surface of the soil about 6 ft. 6 in. The grassy sods were projected to a short distance. In spring all that will be required before planting will be to shovel out a hole large enough to accommodate the roots so that they may develop freely without encountering resistance. The whole operation costs about 7d. per tree, while the holes for planting made by hand labour often cost more and their volume does not exceed 17 cu. ft. Experience has proved that trees planted in soil prepared by dynamite develop more rapidly and come into bearing earlier than those planted in the usual manner. It appears also that dynamite might be advantageously used for preparing land for plants possessing deep roots, such as the vine, especially when the subsoil is hard and compact. With holes at 16½ ft. apart and cartridges containing 5 oz. to 8¾ oz. of explosive (which seems sufficient), the cost of breaking up the land does not exceed £8 per acre, while the same work done by hand to a depth of 2 ft. costs about four times as much. Among other experiments, one was made in a moist soil on which it was intended, after drainage, to grow lucerne. The resistance of the water did not allow of useful work being done; the energy of the explosive was spent in projecting the soil to a great height, making holes measuring about 17 cu. ft. This method does not, therefore, seem advisable for soils possessing a water level not far from the surface. By using the so-called safety explosives (cheddite, gamsite, telsite, etc.), prepared by various factories, the soil can be worked without incurring any serious danger, provided, of course, that elementary precautions be taken to guard against accidents."—C. DUSSERRE, abs. in *Journal of the Royal Society of Arts*, Vol. lxi., No. 3166, July 25, 1913, p. 846-847. (J.G.)

BLOOD-PRESSURE VARIATIONS.—"In view of the fact that doctors practising at moderately high altitudes are being told repeatedly by their patients that they must leave the country on account of their health and the influence the altitude has on their condition, a series of readings was taken to see if a change of from 1,400 ft. to 1,700 ft. would be of any importance on the blood-pressure. The Homestake

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* The paper contains in a table the results of a second test which are not reproduced here.

Mining Company, through whose courtesy this work was carried on, is a gold mining concern whose underground workings are naturally ventilated. This is accomplished by having shafts in different gulches, widely separated from each other, the current of air in some of them being downward, in others upward. The mine is quite free from gases because the character of the rock does not generate any *per se.* A small amount of carbon dioxide is present, due to the decomposition of dynamite, the burning of candles and acetylene lamps, and the exhalations of the miners. At the surface the humidity is very slight, while in the mine the air is practically saturated. The elevation of Lead, South Dakota, is 5,250 ft. above sea level. The shaft is 1,700 ft. deep, having an altitude at its bottom of 3,550 ft. The following table shows the barometric pressure at the different locations cited in this article :—

Place.	Altitude Above the Sea. Feet.	Approximate Mean Barometric Pressure. Inches.	Millimeters.
Sea Level	0	30·	760
Lead, South Dakota	5,250	24·55	622
1,700-ft. level ...	3,550	26·20	663

There is thus a difference of 1·65 in. barometric pressure between the top and the bottom of this shaft. The readings were taken on the miners of this company as they were preparing to go down the shaft to work and again as soon as the bottom was reached. Again, the readings were taken at the bottom after a day's work and then at the top as soon as the ascent had been made. To make the decent the men stood on a cage or elevator and went down 1,700 ft. in approximately two minutes. Coming up, the journey was more rapid, only one minute being necessary. No attempt was made to pick out special men, for they were taken just as they came. They were of all ages and conditions and represent eight nationalities—Americans, English, French, Austrians, Finns, Germans, Italians and Scandinavians. No attempt was made to find out the history of previous diseases. These men were so healthy that they were able to carry on manual labour for eight hours each day, and none seemed to be worn out after the day's work. All readings were taken in the standing position, the only practical way under the circumstances. The instrument used was a standard mercury sphygmomanometer with a 5 in. cuff placed above the elbow. As stated above, the altitude at the top is 5,250 ft., at the bottom 3,550 ft. The 100 men were divided as to ages as follows :—

20 to 30	51
31 to 40	31
41 to 50	13
51 to 60	5

The average age is 32·42 years. The following readings were taken :—

Descending.		Ascending.	
Average at surface	146·45 mm.	Average at bottom	145·27 mm.
Average at bottom	141·07 mm.	Average at surface	139·94 mm.

The average pressure, according to decades, was as follows :—

——20 to 30 years. 51 men.——

Descending.		Ascending.	
Average at surface	145·33 mm.	Average at bottom	144·19 mm.
Average at bottom	139·31 mm.	Average at surface	138·43 mm.

——31 to 40 years. 31 men.——

Descending.		Ascending.	
Average at surface	144·28 mm.	Average at bottom	141·25 mm.
Average at bottom	138·96 mm.	Average at surface	137·35 mm.

——41 to 50 years. 13 men.——

Descending.		Ascending.	
Average at surface	150·30 mm.	Average at bottom	156·61 mm.
Average at bottom	150·77 mm.	Average at surface	159·23 mm.

——51 to 60 years. 5 men.——

Descending.		Ascending.	
Average at surface	157·60 mm.	Average at bottom	152· mm.
Average at bottom	146·40 mm.	Average at surface	147·60 mm.

Maximum Pressure by Decades.

	Descending.		Ascending.	
	Top.	Bottom.	Bottom.	Top.
20 to 30 years ...	198	204	186	194
31 to 40 years ...	210	202	190	172
41 to 50 years* ..	222	226	208	208
51 to 60 years ..	210	184	170	166

Minimum Pressure by Decades.

	Descending.		Ascending.	
	Top.	Bottom.	Bottom.	Top.
20 to 30 years ...	112	105	116	116
31 to 40 years ...	126	116	110	102
41 to 50 years* ...	222	226	208	208
51 to 60 years ...	136	120	116	110

*Immediately after this reading this man developed marked signs of pulmonary tuberculosis, and is now unable to work on that account.

The Greatest Variation.

Descending.			Ascending.		
Age.	Top.	Bottom.	Age.	Bottom.	Top.
35 ...	121	to 142	40 ...	120	to 150
37 ...	142	to 110	26 ...	208	to 178

Variations for 100 Men.

Descending.		Ascending.	
Blood-pressure increased ...	27	Blood-pressure increased ...	24
Blood-pressure decreased ...	61	Blood-pressure decreased ...	65
Blood-pressure, no change ...	13	Blood-pressure, no change ..	11

Number of men whose blood-pressure increased both up and down	10
Number of men whose blood-pressure decreased both up and down	43
Number of men whose blood-pressure remained the same either way	3
Number of men whose blood-pressure was variable	44

It is of interest to compare our readings on the surface before descending with those of Thayer's for normal cases.

	Thayer's Cases.	Our Cases.	Difference.
20 to 30 years	136·9	145·33	8·43
30 to 40 years	140·8	144·28	3·48
40 to 50 years	142·2	150·30	8·10
50 to 60 years	154·8	157·60	2·80

Conclusions.—The rapid change in altitude, either up or down, was responsible for a fall in blood-pressure of approximately 5 mm. Lower readings were obtained from the decade from 30 to 40 than from 20 to 30. It was impossible to forecast by a man's build or any other factor what influence the ride would have on his blood-pressure. It was demonstrated that after a day's work 1,700 ft. below the surface the average blood-pressure was about the same as on the surface before starting to work. It was demonstrated that abnormally high blood-pressure was not a contra-indication to hard work underground. Admitting that the standing position is responsible for an increase of from 4 to 8 mm., these readings show that there is little difference between sea-level readings (Thayer's) and those taken a mile above the sea. As factors influencing some of these higher readings, there must be taken into account the facts that the readings were taken in the standing position, that the men were, for the most part, heavy users of tobacco, tea and coffee, and moderate users of alcoholic beverages. Only one or two men seemed to be nervous at the time of the examinations."—F. E. CLOUGH, *Archives of Internal Medicine*, Vol. ii., June, 1913, p. 590. (H. A. W.)

Reviews and New Books.

THE MINERALOGY OF THE RARER METALS. By E. Cohen and W. O. Wootton. Charles Griffin & Co., Ltd., London, 1912. Pages 211 + xiv., including Index. Price 6s. net.

"The list of minerals dealt with in this work may be considered complete up to March, 1912. The authors follow Dana's well-known classification. In order that the data may be correct in detail many chapters have been revised by specialists in their particular line. This work cannot fail to be of value to prospectors, who, as the authors say, have often walked over ores in the past, and failed to recognise those of the rarer metals. An approximate account of the uses to which such metals may be placed, and also their market value is given.

The work gives evidence of great care having been taken in its compilation. The final section dealing with geographical distribution is particularly useful; also the particulars given for the assay of the rarer metals."—*The Chemical World*, Sept. 1913, p. 297. (J. W.)

DISINFECTION AND DISINFECTANTS.—By M. Christian. Translated from the German by Chas. Salter. 112 pp., 18 illustrations. London: Scott, Greenwood & Son, 1913. Price 5s. net.

"Dr. Christian is Physician-in-Chief of the Royal Institute for Infectious Diseases at Berlin, and he treats of disinfection from the very rational standpoint of the prevention of infection. Scientific disinfection is indeed, as he shows, only a broader application of the principles of asepticism. As such it is capable of being dealt with by definite principles, far removed from the empiricism which used to render a strong smell and a fancy name the chief requisites for a successful (!) disinfectant. The author has arranged his matter very systematically, first studying physical disinfection by heat, light, electricity, etc., and then reviewing the various chemical disinfectants and combined methods of disinfection. An advantage of the German origin of the book is that it does not take sides with any of the much pushed products of rival British manufacturers."—*The Oil and Colour Trades Journal*, Aug. 16, 1913, p. 571. (J. W.)

Arup, Paul S. Industrial Organic Analysis. For use of Technical and Analytical Chemists and Students. Cr. 8vo. pp. 362. *Churchill.* Net 7s. 6d.

Blake, E. H. Drainage and Sanitation. A Practical Exposition of the Conditions Vital to Healthy Buildings. Illustrated. 8vo., pp. 534. *Batsford.* Net 15s.

Brown, S. E. Experimental Science. 1, Physics. 2, Chemistry. In one Volume. Cr. 8vo., pp. 428. *Cambridge University Press.* 5s

Coal Resources of the World (The). An enquiry made upon the initative of the Executive Committee of the 12th International Geological Congress, Canada, 1913. 3 Vol. and Atlas. 4to. *Dulau.* Net 105s.

Collis, A. G. Switchgear and the Control of Electric Light and Power Circuits. 12mo., pp. 86 *Constable.* Net 1s.

Desch, Cecil H. Metallography. 2nd Ed. Cr. 8vo., pp. 444. *Longmans.* 9s.

Fermentation of Cacao (The). Edited by H. H. Smith. Cr. 8vo. *Bale.* Net 10s.

Freeman, H. and Jobbing, E. Examples in Physics. Cr. 8vo., bds., pp. 104. *W. Heffer.* Net 1s.

Gibson, Charles R. Wireless Telegraphy and Telephony without Wires. Cr. 8vo., pp. 156. *Seeley* Net 2s.

Groth, L. A. Welding and Cutting Metals by aid of Gases or Electricity. New Edition. 8vo., pp. 298. *Constable.* Net 6s.

Lee, J. The Economics of Telegraphs and Telephones. 8vo. *I. Pitman.* Net 2s. 6d.

Maycock, W. Perrin. Electric Circuit Theory and Calculations. Cr. 8vo., pp. 370. *Whittaker.* Net 3s. 6d.

M'Lachlan, Norman W. Practical Mathematics. For Students attending Evening and Technical Classes. Cr. 8vo., pp. 192. *Longmans.* Net 2s. 6d.

Ripper, W. M. Heat Engines. Being a new edition of "Steam." Cr. 8vo., pp. 362. *Longmans.* 3s. 6d.

Rossetti, G. A. Metric-English and English-Metric Lengths. 16mo. limp. *Spon.* Net 1s.

Waddell, John. Quantitative Analysis in Practice. 8vo., pp. 170. *Churchill.* Net 4s. 6d.

Wilda, Hermann. Cranes and Hoists, their Construction and Calculation. Illustrated. Cr. 8vo., pp. 168. *Scott Greenwood.* Net 3s. 6d.

Zeeman, P. Researches in Magneto-Optics. With special reference to the Magnetic Resolution of Spectrum Lines. 8vo., pp. 236. *Macmillan.* Net 6s.

Abstracts of Patent Applications.

(C.) 375/12. Albert Henry Guy. An improvement in bearings for shaft rollers, or other mechanical haulages and the like with the necessary fixings thereof. 17.7.12.

This application relates to rollers used in incline shafts of mines and on the tracks of mechanical haulages for carrying the hoisting or haulage rope. The particular feature of the invention consists in providing the roller with resilient bearings. This is accomplished by the use of springs which are placed beneath the bottom half of the bearing bush at each end of the roller spindle.

(C.) 572/12. Edward Henry Johnson (1), Arthur Edward Winterton (2) Ore separating means. 31.10.12.

This application refers to a revolving trommel, placed between mortar box and ore feeder and adapted to separate the finer portion already small enough to pass the battery screen used.

(C.) 596/12. Louis David Mills. Improvements in hydraulic concentration and the like. 9.11.12.

This application refers to improvements in conical classifiers in which the stream of pulp to be separated is introduced at the bottom of the inverted cone and water supplied to clean the concentrate produced. A plurality of annular partitions of successively increasing diameter are provided with upwardly directed slots through which the settling material falls, being at the same time protected against the main stream of ascending pulp.

(C.) 643/12. Sidney Sherrard Osborn. Improvements in pulverizers. 7.12.12.

This patent relates to application for pulverising ore and other material, in which the reduction is effected by means of moving masses of pebbles and the like.

(C.) 6/13. James Stuart. Improvements in or appertaining to shaking tables or like apparatus for recovering diamonds from matter associated therewith. 2.1.13.

This application relates to improvements in or appertaining to shaking tables or like apparatus for recovering diamonds from matter associated therewith.

(C.) 303/13. John Francis Staley (1), Cyril Frank Jackson (2). Improvements in safety mechanism for the winding gear of coal pits and the like. 26.6.13.

This invention relates to a safety winding mechanism and consists of a weighted brake and steam valve lever, which, in combination with a governor and traversing stop-arms, trips the mechanism in the event of excessive speed or when overwinding occurs.

(C.) 310/13. Wm. Fennell. Improvements in or in connection with tube mills. 30.6.13.

This application relates to improvements in connection with tube mills and refers particularly to the grinding bodies employed in the tube-mill to carry on the grinding operations. The particular claims refer to the elongated grinding bodies being hollow and having holes in the shell so that the weight of the bodies relative to the overall dimensions is small, and that the bodies act as carriers or distributors of the material to be ground. Other claims of the application refer to specific detailed construction of these grinding bodies.

(C.) 318/13. Ray C. Newhouse. Improvements in roll crushers. 4.7.13.

This application refers to an improved roll crusher in which the roll is in a fixed bearing and is provided with renewable projecting teeth.

A curved jaw plate is held in position by stiff springs and gives a tapering opening for receiving the ore to be crushed, the lower end of the jaw being in close proximity to the roll.

A feed hopper is provided and it is claimed that masses of ore too large to enter the opening between the roll and the jaw are broken by the roll teeth acting on the underside of the mass and eventually pass into the opening and are there crushed to the required size.

(C.) 350/13. Alexander Purser. Improvements in valves. 19.7.13.

This application has reference to improvements in valves and cocks for allowing a certain minimum of fluid to pass through the valve when it is full open, without back pressure. When the valve is connected to a machine such as a rock drill, the full amount of air is allowed to pass automatically.

(C.) 379/13. William Augustus Hall. Process of desulphurizing ore. 8.8.13.

This application refers to a process for desulphurizing ores consisting in subjecting ore in the presence of steam to the action of a non-oxidising flame at a temperature higher than the temperature at which the sulphides are distilled, and between the distillation and fusing points of the sulphides, and also agitating the ore, thus obtaining a direct recovery in the form of elemental sulphur of the fixed atom of sulphur in sulphuric ores.

(C.) 380/13. William Augustus Hall. Improvements in the art of desulphurizing ores. 8.8.13.

This patent refers to an improvement in the art of desulphurizing ores consisting in distilling off all or substantially all the sulphur contents in elemental form by the direct application to the ore within a furnace of the flame of a reducing gas containing a high percentage of hydrogen in the presence of a volume of oxygen admixed with the gas sufficient to permit combustion of liberated nascent hydrogen or sulphuretted hydrogen formed by reaction consistent with the requirement that the flame shall not acquire an oxidising character, and withdrawing the sulphur from the furnace at a level above that at which the oxygen is admitted.

(C.) 381/13. William Augustus Hall. Improvements in furnaces for decomposing pyrites and other metallic sulphides. 8.8.13.

This application refers to a shelf furnace which has been modified so as to be able to carry out therein the decomposition of pyrites or metallic sulphides according to the requirements of Patents Nos. 379 and 380/13 of William Augustus Hall. The peculiarity of this furnace consists, in my opinion, of the means for introducing into the furnace at points just above the shelves steam and reducing gas and air, as well as in the apparatus for controlling the supply of the said steam and of gas and air to the inlets adjacent to the various shelves.

(C.) 390/13. Dr. Graf Botho Schwerin. Improvements in the electro-osmotic extraction of water from animal, vegetable or mineral substances, and in apparatus therefore. 16.8.13.

This specification describes a process for the electro-osmotic extraction of water from finely-divided mineral and other substances. It consists in subjecting the material to be freed from water to the action of an electric current in a space of confined volume (*e.g.*, filter-press) containing electrodes under pressure or suction exerted or transmitted by the material itself and without any motion of the drying mass. It is claimed that by this process very low pressures suffice for filtration owing to the migration of fine particles from the filter-cloth into the cakes of compressed material under the influence of the electric current so that the filter surface is not choked.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(P.) 449/13. Chas. Kirk (1), James Hamilton (2). An improvement for connecting the flanges of rollers carrying wire ropes in mines which will prevent them coming loose. 23.9.13.

(P.) 450/13. Herbert Alfred Read. A new or improved means for detecting unexploded blasting charges in mines and the like. 22.9 13.

(P.) 452/13. Charles Henry Perrins A process for recovering tin from scrap metal. 23.9.13.

(P.) 453/13. Walter Augustus Clive. Improvements in safety catch devices for mine skips, cages, lifts and the like. 24.9.13.

(P.) 454/13. George Tom Cannon. Improvements in rope grips or jockeys for mechanical haulages and the like. 24.9.13.

(P.) 455/13. James Dunlop Wilson (1), Maurice Solvie Aaron (2). Improvements in drills and detachable bits therefore. 25.9.13.

(P.) 456/13. James Hamilton Austruther Macadam (1), John James Robert Smythe (2), Charles Wilson McKelvie King (3), Ferdinand George Krause (4). Improvements in rock drilling machine valves. 25.9.13.

(C.) 457/13 Carl Hartman. Improvements in or relating to the manufacture of hexanitrodi phenylsulphide. 25.9.13.

(C.) 458/13. Carl Hartman. Improvements in or relating to the use of explosives for filling projectiles, mines, torpedoes, primers, detonators and the like. 25.9.13.

(C.) 459/13. Harry Alexander Stenning. Improvements in and relating to controlling devices suitable for the cylinders of locomotives. 25.9.13.

(C.) 460/13. Cyril Douglas McCourt (1), Bonecourt Surface Combustion, Ltd. (2). Improvements in or in connection with heating by combustion of combustible mixtures and in furnaces therefore. 25.9.13.

(P.) 461/13. Richard Monra Dixon. Compressed air filter. 25.9.13.

(P.) 462/13. Alexander Granger. Improvements in pulverising apparatus. 27.9.13.

(P.) 463/13. James Wilshir (1), William Martin (2). Improvements in hose connectors. 29.9.13.

(P.) 464/13. William Calder. Improvements in grinding apparatus, of the tube mill type. 1.10.13.

(P.) 465/13. Robert Anderson. Improved tube mill and liner. 4.10.13.

(P.) 466/13. William Simpson. A new and improved central coupling buffer. 4.10.13.

(P.) 467/13. David Laird. Package and mode of packing liquids contained in bottles. 7.10.13.

(P.) 468/13. John Hawes. Double fly catcher. 7.10.13.

(C.) 469/13. Walter William Wright. Improvements in or relating to apparatus for crushing, and if required amalgamating gold or silver ores. 7.10.13.

(P.) 470/13. Thomas Reay Brown. A new type of detachable bit for rock drilling machines. 7.10.13.

(P.) 471/13. David Gillespie. A new type of renewable truck bearing. 7.10.13.

(C.) 473/13. Donald Donald. Improvements in wire strainers. 8.10.13.

(C.) 475/13. Paul de Brunn. Improvements relating to zeolites or base exchanging bodies. 9 10.13.

(C.) 476/13. Charles Kratt. Improvements in and relating to apparatus for heating liquids electrically. 9.10.13.

(C.) 478/13. Frank Knight Woodroffe (1), Harold Cecil Hodgson (2). Improvements in centrifugal liquid pumps. 9.10.13.

(C.) 479/13. Carl Travers. Free dust breathing for mining and other plants. 9.10.13.

(P.) 480/10. George Watson Tait. Improvements in and relating to railway level crossing gates, consisting of booms and making them also suitable as an apparatus for directing road traffic. 10.10.13

(C.) 481/13. Jean Repetto. Improvements in refrigerators. 10.10.13.

(P.) 482/13. Denis Francis Breslin. Improvements in rock drills. 10.10.13.

(P.) 484/13. George Green. Improvements in spray ejectors. 11.10.13.

(P.) 485/13. Thomas Sidney Jory. Improvements in rock drilling machine chucks. 11.10.13.

(P.) 487/13. Rudolph Goch Else. Spraying nozzle. 14.10.13.

(C.) 488/13. William Russell Degenhardt. Improvements in apparatus for generating or producing gas. 16.10.13.

(P.) 489/13. James Mason. An improved element for electrical cooking and heating. 16.10.13.

(C.) 490/13. Maurice Leblanc. Improvements in and relating to high speed rotating bodies. 16.10.13.

(C.) 491/13. John George Deyner. Improvements in oil burning forges. 16.10.13.

(C.) 492/13. John George Deyner. Improvements in drill sharpening machines. 16.10.13.

(C.) 494/13. Benjamin Holt (1), Pliny Eastman Holt (2). Traction engine. 17.10.13.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

Aspland, C. H., *l/o* Germiston; Sub-Nigel, Ltd., P. O. Box 50, Nigel.
Brown, G. D., *l/o* Barberton: P. O. Heidelberg, Transvaal.
Fyfe, A., *l/o* Mexico; 74, Cedar Street, Roxburgh, Boston, Mass., U.S.A.
Graham, W. H., *l/o* England; Kimberley Reef Mine, Bindura, *via* Salisbury, Rhodesia.
Haugh, H., *l/o* "Rhodesia" Bondi: Calani, Forbes, N.S.W. Australia.
Howe, B., *l/o* Australia; Mining Club, 3, London Wall, London, E.C.
Lindsay, R., *l/o* Cleveland; Transvaal G. M. Estates, Ltd., Pilgrims Rest.
McConnell, T., *l/o* Rosetta Mine; Ivy Creek Cyanide Works, Barberton,
Merrill, C. W., *to* 121, Second Street, San Francisco, Cal., U.S.A.
Nellmapius, E. H., *l/o* Roodepoort; City Deep Ltd., P. O. Box 1411, Johannesburg.
Nicol, G., *to* P. O. Box 108, Germiston.
Paterson, G. S., *l/o* England; c/o The Niger Co., Mining Department, Jas, N. Nigeria.
Phillips, J. C., *l/o* Rietfontein; Van Ryn Deep, Ltd., P.O. Box 225, Benoni.
Price, W. S. V., *l/o* Florida; 49, Hanover Street, Mayfair, Johannesburg.
Ross, H., *l/o* Roodepoort; P. O. Box 181, Barberton.
Smith, J. H., *l/o* Scotland; Rose Deep Ltd., P. O. Box 6, Germiston.
Stockdale, C., *l/o* Langlaagte; Van Ryn Deep, Ltd., P. O. Box 225, Benoni.
Tillard, M. O., *to* 62, Muller Street, Yeoville, Johannesburg.
Tom, I., *to* P. O. Box 2951, Johannesburg.
Ward, J. P., *l/o* Maraisburg; Govt. G. M. Areas, Ltd., P. O. Box 28, Brakpan.

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Vol. XIV. NOVEMBER, 1913. No. 5.

Proceedings
AT
Ordinary General Meeting, November 16th, 1913.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, November 16th, 1913, Mr. Alex. Richardson (President) in the chair. There were also present:

32 Members: Messrs. J. E. Thomas, H. A. White, T. Donaldson, C. Toombs, F. W. Watson, John Watson, A. Whitby, Prof. J. A. Wilkinson, Dr. W. A. Caldecott, W. R. Dowling, Dr. J. Moir, J. R. Williams (Members of Council), A. S. Allam, H. C. Boydell, J. Q Braidwood, C. Y. Bruce, J. Chilton, W. M. Coulter, T. D. Delprat, R. Dures, B. J. Hastings, W. E. John, S. Morison, S. Newton. E. A. Osterloh, F. D. Phillips, W. S. V. Price, E. Roberts, J. A. Taylor, A. Thomas, W. E. Thorpe and E. M. Weston.

16 Associates and Students: Messrs. E. G. Baskett, G. F. Blenkinsop, C. L. Dewar, O. A. Gerber, J. Gibson, M. Green, W. Human, A. C. Lurie, E. J. Moynihan, S. E. Willows-Munro, H. Rusden, P. Scatterty, W. J. Sutherland, I. R. Trollip, W. Waters and J. A. Woodburn.

12 Visitors, and Fred. Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

NEW MEMBER.

Mr. Jas. E. Thomas and Mr. T. D. Delprat were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidate for membership had been elected, as follows:—

ROGERS, PERCY ARDEN, Hamburg Post Office, near Roodepoort. Shift Boss.

The Secretary announced that since the last meeting of the Society the following Associates and Student had been admitted by the Council:

As Associates—

ANDERSON, WILLIAM, 92a Regent Street, Bellevue, Johannesburg. Analytical Chemist.

MUNRO, STANLEY EVETT WILLOWS, City Deep, Ltd., P.O. Box 1411, Johannesburg. Cyanide Learner.

THOMAS, JOHN HERBERT, Campfield Villa, Truro, Cornwall, England. Analytical Chemist.

As Student—

ESTILL, FREDERIC (Crown Mines, Ltd.), 123 Sivewright Avenue, Doornfontein, Johannesburg. Cyanide Learner.

GENERAL BUSINESS.

OBITUARY.

The President: I should like to place on record our sincere regret at the death of Sir William Preece, the distinguished electrical engineer. Though not a member of our Society he kindly attended one of our meetings in and gave us an illuminating address on the progress of technical education in Great Britain.

NOTES ON WASTE ROCK VALUES.

Mr. Robert Dures (*Member*): For many years past I have been increasingly interested in the fascinating question of what is the lowest value of ore that will pay on these fields, and its ever shifting answer. This interest was greatly accentuated by a residence of some years in Rhodesia, where opportunities presented themselves of seeing very low grade ores worked at a profit. Looking back on the past history of mining on the Rand, and remembering the methods of sorting then in use it occurred to me that there must be some good values in the fine dirt brought up during development operations. I therefore set to work on my return to Johannesburg last year to sample some of the old waste rock dumps, and in a few months had taken a large number of samples from over 30 dumps—many of which had been lying out for 15 or 20 years. Especial care was taken over those dumps that consisted mostly of development rock, and particularly from the mines that were developed *on the reef*, as it was among these, naturally, that the highest values were obtained. In each case a classification was made as follows:

1. *Fine*—material passing 1 in. mesh. This is very friable, and could be sent direct to the tube mill.
2. *Medium*—pieces of rock from 1 in. to 2½ in., to be sent to the battery.
3. *Coarse*—the big lumps of waste, some of which had pieces of reef attached, and what was of more importance, pieces of "contact" carrying gold bearing mineral.

I found that the best way was to go over the dump three times, sampling each of the three classes separately. With some experience it is easy to detect the valuable *contacts*, and to know which reefs they have been lying in touch with. From the results of my tests I am convinced that many mines have at grass, or had a year ago, from two to twelve weeks crushing of payable ore in their development dumps—in a condition that lends itself to easy treatment. The amount of *fine* varied from 30% to 50%, and its value was mostly above the average of the dump, and in several cases reached twice the value of the rest of the rock, in others three and four times as much.

Good sampling faces were found in the waste rock dumps which were penetrated when material had been taken from them to make concrete, or, when quantities had been carted away to make "metal" for road making and mending. From the values obtained it was clear that these particular dumps had not been sampled, else the ore would not have been sold to road contractors, and I have not been able to find any records of a thorough and systematic sampling of these dumps.

Here are the results of samples taken from an ore dump with which I was familiar.

1	North side	5·8 dwt.
2	Centre	3·5 ,,
3	South side	1·3 ,,
4	North side	6·2 ,,
5	Centre	3·0 ,,
6	South side	4·0 ,,
7	Fine	3·7 ,,
8	Medium—from 1 in. to 2½ in.	4·3 ,,
9	Coarse—big pieces of rock ...	1·8 ,,
10	Reef sorted out	2·5 ,,

This dump would be worth about 16s. a ton, and it is safe to say that with up-to-date handling and treatment a profit of 10s. a ton could be made from it. The profit to be made would to some extent depend on the capacity of reduction plant.

Here are the results of general samples taken from a very old dump.

1913		Dwt.
February	18	3·6
	19	6·6
	21	3·1
	22	3·9
	27	4·8
March	5	2·8
	6	2·3
	7	3·1
	8	3·3
	12	2·6
	13	3·5
		39·6
	Average	3·6

As far as I have been able to ascertain, 3·5 dwt. is about the average value to be expected from dumps of this class. This is borne out by what I am told of the values of two large dumps now under course of treatment. As a further confirmation of the above, some of the details of the sampling and assaying of another dump, 15 miles away from the previous one, are given here. The two halves of the dump were sampled separately, and are marked

W end of dump
E end of dump.

The former consists for the most part of low grade reef, the latter is a mixture of country rock and the fine dirt from development work. The assays were done in duplicate and parted, so as

No.	Description.	Mesh Holes to 1 in. Linear	% Weight Sample.	Mean Assay Dwt.	
1	W end of dump Fine Original			5·5	Taking
2	Fine	+8	48	4·6	Fine = 50% of tonnage
3	Fine	−8+20	22	4·8	Medium = 20% ,,
4	Fine	−20+40	11	3·7	Coarse = 30% ,,
5	Fine	−40	19	9·2	The assay of the W
6	Medium			2·0	end = 3·63 dwt.
7	Coarse			0·5	
8	E end of dump Fine Original			5·6	In the same way as
9	Fine	+8	48	6·1	above
10	Fine	−8+20	21	3·2	E end = 3·63 dwt.
11	Fine	−20+40	9	3·5	
12	Fine	−40	22	5·4	
13	Medium			4·2	
14	Course			0 8	

to make sure of the fine gold contents. As already stated, the samples were classified as:—

1. *Fine*—passing 1 in. mesh.
2. *Medium*—from 1 in. to 2½ in. rock.
3. *Coarse*—chips from big lumps.

The *fine* from each half of the dump was quartered down and a portion called "original" taken for assay. The rest was graded—

a. +8 holes to 1 in. linear.
b. −8+20 ,, ,,
c. −20+40 ,, ,,
d. −40 ,, ,,

and each product was assayed.

The mean of the two halves of the dump = 3·46 dwt. 3·46 dwt. at 4s. 3d. per dwt. = 14s. 8d. per ton of ore. By rejecting the large pieces of rock the value could be brought up to 18s. or 20s. a ton.

As you dig into a dump you will frequently find the big pieces covered with fine dirt, and it was interesting to discover that on washing the big pieces that the sediment from the water used for washing assayed 12·5 dwt. Occasionally high samples are encountered even in waste rock which has been sorted recently, and the following are assays of samples taken during this year of current waste rock:—

22 dwt.
24 ,,
267 ,,
4·5 ,,
5 ,,

In searching for waste dumps it was curious to find that one or two of them had been sent underground for "packing stopes," or, merely to fill up an abandoned shaft, and at least one had been flattened out to make room for construction work. Several valuable dumps too have been covered up or partly covered with ashes. Others lie under masses of more recent waste rock, and I have in mind now one dump that has been uncovered, and the more recent rock carefully turned over and hand picked. The old development dirt still stands, and on sampling the fine stuff on the surface, and reducing the higher assays the result was 4 dwt. per ton.

Extraction tests were made:

1. By shaking with mercury, when extractions were got of from 65% to 85%—one sample of 4·7 dwt. giving an extraction of 87%.
2. Laboratory tests with cyanide showed that there is apparently no difficulty in the treatment of waste rock dumps.

Owing to the friable nature of the ore in the dumps, the crushing would be very easy, also in the tube mill, so that the reduction costs would be lower than on current rock. Probably the difference would be more than enough to pay for tramming the waste to the reduction works. In most cases it would be unnecessary to do any sorting at all with the old development dumps, and thus save the cost of washing and sorting the big pieces. In some it would be of advantage to sort out 25% to 30% of big rock.

NOTE ON A PROPOSED METHOD OF ASSAY OF TIN ORES WITH SOME REMARKS ON THE ELECTRICAL CONDUCTIVITY OF MINERALS.

Mr. A. Whitby *(Member of Council)*: With regard to the question Mr. White raised last meeting as to the depth of the deposit of tin on a crystal of cassiterite, we found that using zinc as the anode, a saturated solution of zinc chloride as the electrolyte and a crystal of cassiterite weighing 0·5 gm. as cathode and leaving it in contact for 24 hours, only 1% of the tin was dissolved; but it is curious to remark that taking another crystal of cassiterite of a similar nature and size and placing it in contact with a zinc plate covering with zinc fume and adding hydrochloric acid, in the space of about 10 minutes we managed to get into solution nearly 5% of the tin. The penetration must have been considerably greater in that case than in the case where the current was used, showing very probably in the latter case that the deposit is much denser.

THE NATURAL SODA DEPOSITS OF AFRICA; WITH SOME NOTES ON THE ALKALI TRADE.

By John Watson, F.I.C. (Member of Council).

I will commence this paper by dealing with the African soda deposits outside the Transvaal, and then pass on to a more detailed account of one near at hand and its industrial possibilities.

The Soda Lakes of Egypt.—In the Libyan Desert, about 40 miles N.W. of Cairo the chief deposits of Egypt occur. In a valley running in a north-westerly direction for 21 miles, there is a string of eleven lakes. The total area of these lakes is about 12,000,000 square meters, containing about 100,000 tons of alkali in solution. The lakes all contain, also, a high percentage of common salt.[*] The bed of the lakes consists of natural soda covered with a layer of salt. There are several distinct varieties of soda, the richest being termed *gem natron*; it is a hard crystalline deposit. Overlying it, as a rule, is a variety named *sultani* which occurs in large quantities and is easily extracted in the summer when the lakes are dry. This variety has been chiefly used for the manufacture of soda ash.

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[*] V. Seymour Bryant, B.A., *Jr. Soc. Chem. Ind.*, Vol. xxi pp. 785-7.

There are also extensive fields, around the shores of the lakes, of a variety named *Korcheff*, which forms an incrustation on the surface of the sand. Analyses of these varieties, by Mr. Seymour Bryant show :—

	Gem Natron.	Sultani.	Korcheff.
Sodium carbonate ...	42·71	36·85	35·56
Sodium hydrogen carbonate	33·79	28·59	17·22
Water (chemical and mechanical) ...	16·56	15·68	14·80
Sodium chloride ...	1·83	8·43	7·49
,, sulphate ...	1·91	5·00	6·63
Insoluble (SiO_2, $CaSO_4$, $CaCO_3$, Fe_2O_3. ...)	3·19	5·38	18·30
	99·99	99·93	100·00

The first two analyses indicate that the soda occurs as :—Na_2CO_3, $NaHCO_3$, $2H_2O$ or $Na_3H(CO_3)_2 \cdot 2H_2O$ known to mineralogists as *trona*.

The lakes are red in colour, which is said to be due to the presence of *artemia* in large numbers.

The new *Webster* defines *artemia* as :—"A genus of phyllopod crustacea, found in salt-lakes and the brines of salt-works ; a brine shrimp." The length of one of these little organisms is about ¼ in. from the jaw to the tip of the tail.

Exploitation in Egypt.—A limited company was formed about 1890 to exploit these deposits ; entitled "Compagnie des Soudes Naturelles d' Egypte." Mr. M. Naumann reported to this company in 1891 that some 2½ million tons of soda salts were stored in the Wadi, of which 1½ million tons were rich enough to work. Dr. Lunge, the well-known Emeritus-Professor of technical chemistry, of Zurich, visited the Wady Natrun in 1897 and took a large number of samples. The Egyptian Salt and Soda Co., Ltd., was registered in London, in 1899 with a capital ol £451,000 and took over the Natural Soda Co. of Egypt. It has paid dividends varying from 4% up to 6½% per annum. In 1900, as a result of Dr. Lunge's experiments, a factory was established at Bir Hooker in the Wadi Natrun.[*] The process adopted was to dissolve the crude natron at 80° C. in the water from one of the lakes, allowing it to cool to 40° C., then carbonating by CO_2 gas. Bicarbonate of soda was precipitated and this was calcined in Thelen pans to form commercial soda ash. The CO_2 evolved, on heating the bicarbonate, was used in the carbonating process just mentioned. In the 1909 edition, Dr. Lunge remarks that the manufacture of pure soda ash has been discontinued there, during recent years, for reasons unknown to him.

German East Africa.—Between 2° and 3° South of the Equator and about 36° East of Greenwich, there is in German East Africa, a lake named Natron, some 60 miles long and from 2 to 9 miles wide. This is said to have been discovered by Fischer in 1883. The soda deposits of that district were explored and described by Dr. Carl Peters,[*] who found that natural carbonate of soda was used as a trading article in Masailand.

The commercial salt is dug out from Lake Natron, which lies about 100 miles W.N.W. from Kilimanjaro. The water of the lake is said to be soft, of a reddish-yellow colour and has a temperature of about 50° C. The shores are covered with a salt crust and abound in hot alkaline springs. Dr. Peters took samples of the natural soda crystal, or natron, to Germany where they were analysed by Prof. Kämmerer. From an article which appeared in the *Chemiker Zeitung*,[†] we gather that considerable interest is taken, in Germany and German East Africa, in the working of Lake Magadi : as Lake Natron in German territory is four times as large as Lake Magadi and only about 31 miles from that lake.

Magadi.—In British East Africa, about 93 miles from Kiu station (on the Uganda railway) is a lake of considerable size. The natural soda deposits there are said to cover an area of some 30 square miles. The Magadi Soda Co., Ltd., has been formed with a capital of £1,312,500 to acquire a 99 years lease of, and to develop, this large deposit of natural soda. Kiu station is 267 miles from Mombasa, and a railway from Kiu to Magadi Lake is nearing completion. Works are now being erected by the Magadi Soda Co., on the Manchester Ship Canal, 7 miles west of Manchester ; they propose to manufacture soda ash, soda crystals, bicarbonate and caustic soda. The Central Mining and Investment Corporation hold about £150,000 in Magadi shares. Dr. J. Bland-Sutton, F.R.C.S., in his book,[‡] published in 1911, refers to "Lake Magadi at the southern end of the eastern arm of the Rift Valley. The deposit is estimated to contain 200,000,000 tons of soda ; whenever a portion of the surface soda is removed, it is quickly replaced, as though it were pushed up from below."

Fezzan.—Lake deposits of natron also occur in the oasis of Fezzan, in Northern Africa.

Zoutpan, North of Pretoria.—Some 25 miles north of Pretoria and 14 miles west from Hamans

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[*] "Sulphuric Acid and Alkali," Lunge, Vol. II, Part I, p. 62.

[*] *Jl. Soc. Chem. Ind.*, 1895, p 616.

[†] See *Jl. Soc. Chem. Ind.*, 1910, p. 210.

[‡] "Man and Beast in Eastern Ethiopia," p. 251.

Kraal station in the Bush-veld, there is a crater-like depression. I was first told of this deposit of natural soda by Mr. David Laird about 1903. He described it as a red lake in the rainy season. Mr. Laird took samples from the salt-pan and sent them to Messrs. Brunner, Mond & Co., but that firm, which earns handsome profits by their works in Cheshire, apparently did not worry about the South African imports at that time. This deposit on the farm Zoutpan has been described by Drs. Hatch and Corstorphine,* also by Cohen. A bore-hole was put down in the middle of the pan, under the supervision of one of the first-named geologists. A section of the beds intersected is as follows :—

Soda-pan (Zoutpan) 25 miles N. of Pretoria. Photo takes in the shore and pan.
[Photo by P. A. Ogilvie, Esq.]

	Thickness. ft.	in.
" Brine (sp. gr. 1·25, and containing 21·28% of chloride and 7·22% carbonate of sodium.	0	11
Mud and brine	0	8
Salt deposit, consisting of mixed chloride and carbonate (16·09% NaCl ; 68% $Na_2CO_3 \cdot 10H_2O$; and 15·20% insoluble matter).	1	6
Greyish clay, impregnated with saline matter.		
Salt deposit, consisting of mixed chloride and carbonate (8·08% NaCl ; 78·88% $Na_2CO_3 \cdot 10H_2O$: 4% insoluble matter.	2	4
Dark-coloured clay, impregnated with saline matter.		
Salt deposit, consisting of mixed chloride and carbonate (6·94% NaCl ; 88·50% $Na_2CO_3 \cdot 10H_2O$: 4·56% insoluble matter.	8	6
Pure carbonate of soda, containing only 0·6% of NaCl, and 0·3% of insoluble matter, sp. gr. 1·39.	2	0
Salt deposits (6·20% NaCl ; 83·80% $Na_2CO_3 \cdot 10H_2O$; 10% insoluble matter	3	0

followed by clays down to a depth of 188 ft., at which depth the borehole ceased.

The pan is surrounded by granite kopjes, so that the sodium carbonate has probably come from decomposition of the felspar contained in the granite. This "salt-pan" has been exploited by two well-known firms with a view to manufacturing common salt (sodium chloride). Apparently their chief trouble has been the quantity of the more valuable soda carbonate present.

The Red Colour of the Soda Lakes.—It will be noticed in the descriptions of all these natural soda deposits, that, at certain times of the year, or under certain conditions, the water is described as of a reddish colour. Payen has ascribed this to the presence of small crustaceans, *artemia salina*, which appear in large masses when the water has attained a density of 1·16. They are then said to be of a grey or greenish colour ; on further concentration to sp. gr. 1·21 they die and form a red froth at the surface.* Chatard also found large masses of these in the soda lakes of North America. A. H. Hooker found the springs feeding the soda lakes of Egypt to be full of algæ ; the springs themselves contained sulphate and chloride of sodium. Nevertheless Dr. Lunge, who has been a great controversialist in chemistry as applied to alkali manufacture, declines to accept the assumption that the red colour is regularly caused by the presence of *artemia* or other animal organism. He took samples of red water from the lakes of Wadi Natrun and they have preserved that colour for many years. It remains after frequent filtrations. The red colour shows *nothing* under the microscope, the colour is at once discharged by adding nitric acid or hypochlorite and hydrochloric acid. Dr. Lunge thinks the red colour is due to organic substances present in *solution.* There is no iron present.

Alternate Layers at the Transvaal Soda-pan.—The writer has twice, recently, visited the Pan. In the dry season, a party can walk across the crystallised surface. Indeed, a double-way truck line has now been carried to near the centre of the pan. On taking a 6 ft. length of $\frac{3}{8}$ in. iron rod and pushing it down through the crystallised surface, one comes upon a series of layers of crystal. A few inches thickness of crystal, then a few inches of soda solution and black mud, again another layer of hard crystal and so on. When the rod is withdrawn, bubbles of gas are evolved which catch fire on applying a match. This is probably marsh gas from decomposed vegetation.

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* "Geology of South Africa," 1905. pp. 184-185.

* Lunge's "Sulphuric Acid and Alkali," Vol. ii, Part i, p. 58.

South African Alkali, Ltd.—This company was registered in 1912 as a private company, but has now been converted into a public company. Its objects are to acquire the lease over the farm Zoutpan, to work the natural deposit and purify the same, with a view to manufacturing soda crystals, soda ash, granulated soda carbonate, and, later on, caustic soda. An experimental plant has been working in Main Street, Johannesburg, near "The Stadium", where soda crystals of excellent quality were manufactured from the crude soda sent down from the pan.

The writer took a series of samples at these works in Main Street, last February, which on testing showed :—

(*a*) *Crude Soda from Pan, sun-dried only, average sample from several bags*:

Sodium Carbonate	61·60%
,, Chloride	6·37%
Real Soda (Na_2O)	36·03%

(*b*) *Sample of Calcined Crude Soda, ex Furnace*:

Sodium Carbonate	67·50%
,, Chloride	17·32%
Real Soda (Na_2O)	39·48%

A larger furnace has now been erected on the high ground above the soda-pan. A tram-line, hauling-engine and boiler have also been installed and have begun work.

Analysis of Crude Calcined Soda.—The writer took an average sample on 6th September of the first 5 or 6 tons from the new furnace. This gave the following analysis :—

Sodium Carbonate ...	82·58
,, Hydrate... ..	·85
,, Chloride ...	8·46
,, Silicate... ...	1·89
,, Aluminate ...	·14
,, Sulphate ...	·13
Insoluble in water ...	4·39
Moisture...	2·02
	100·46
Available Alkali, real equivalent ...	= 49·29%
or, in dry state ...	50·07 ,,

Calcined Crude Soda as a Substitute for Lime.—The gold mines of the Transvaal in 1911 consumed 48,639 tons of lime, value £120,511. Several of the Rand gold mining companies have recently tried this crude roasted soda, chiefly for neutralising acid water underground before pumping. The cost per ton, £2 10/- f.o.r. Haman's Kraal, is about the same as lime. The soda contains a lower percentage of available alkali, but it is much more easily soluble in water. According to Lunge and Hurter's "Alkali Maker's Handbook", 2nd edition, pages 16-19 :—

100 parts cold water dissolve 0·128 parts $Ca(OH)_2$ whilst
100 ,, ,, at 15 C. dissolve 16·20 ,, Na_2CO_3

In other words sodium carbonate is 127 times more soluble than lime hydrate. Hence, there should be less waste in dissolving and using soda as compared with lime. Mr. Crosse has pointed out* that calcium sulphate is a cyanicide; experiments carried out by myself supported his contention.† Sodium sulphate has not this objectionable character.

The late Mr. G. O. Smart has shown very fully‡ in his admirable chapter on stamp-milling, that the mill service water pipes become choked with calcium sulphate, which is very slightly soluble in water. He says :—"No remedy is yet known for economically preventing this deposit from mill service water, and it is therefore necessary for all mains delivering such water to the mill to be of ample size, so as to allow of considerable incrustation before they are incapable of conveying the water required. They should likewise be duplicated throughout their whole length so that each pipe-line can be cleaned out in turn, an operation which requires considerable time and necessitates taking apart the pipes." Sodium sulphate being much more soluble in water than calcium sulphate, this choking of the pipes should not occur. According to Lunge and Hurter :—

100 parts cold water dissolve only 0·23 parts $CaSO_4$, whereas
100 parts ,, at 13·3°C. dissolve 11·74 ,, Na_2SO_4

In other words, sodium sulphate is 51 times more soluble in a given bulk of cold water than is calcium sulphate.

Mill Plates and Slime Tanks.—It remains to be seen whether the continuous use of such crude soda, instead of lime, will injuriously affect amalgamation on the mill-plates. Where possible, the soda should be added so that the precipitated mud (chiefly $CaCO_3$) has a chance to settle out. Mill-men like clean water over their plates, not muddy water, full of flocculent matter.

When the physiologist Harvey first propounded his discovery of the circulation of blood, he was denounced as a heretic. When, about 1898, our past president Mr. John R. Williams and others recommended the use of lime in the ore bins or

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*See this *Journal*, Vol. XII (1912), pp. 405-407.

†See this *Journal*, Vol. XIII. (1912), pp. 55-57.

‡"Rand Metallurgical Practice" (1912), Vol. I., pp. 71-72.

mill water, they drew upon their heads the wrath of many amalgamators and battery managers, who held very different opinions about the advantage of using lime, sufficient to make the mill-pulp distinctly alkaline. As Mr. Williams said, it was like a red rag to a bull[*]. Some metallurgists seem to doubt if soda will coagulate and settle slime as well as lime does. Our *Journal* for June contains a paper on "Slime Settlement," taken over from the *Australasian Institute of Mining Engineers*: on page 638 it says:—"Sodium carbonate, if the water contains mineral salts, settles slime by precipitating salts in solution as carbonates, which carry down suspended matter as they settle."

Acidity of Water from Sand-filled Workings.—Dr. Caldecott and Mr. O. P. Powell, in their interesting paper read at the September meeting of this Society, predict that the acidity of mine water will steadily increase as sand-filling becomes more general. Nature, which has already endowed the Transvaal so lavishly with gold, has also provided as an antidote to the above acidity a large deposit of carbonate of soda, within about 60 miles of the Main Reef Series. It is to be hoped that the sale of crude calcined soda, for this and other purposes, will prove beneficial to both buyer and seller.

Green's Economiser Tubes.—It is well known that the vertical cast iron tubes of Green's Economisers become gradually coated with scale, where hard water is used for boiler-feed. The scale formed in these tubes is very hard and difficult to remove. A set of such tubes had to be cleaned recently on the City & Suburban mine. Mr. Pattrick, the Government Boiler Inspector, suggested the use of soda, and some South African crude calcined alkali was used. This, after digesting, worked to the back of the scale, causing it to loosen, when it was easily removed by a spring scraper. (A sample of the scale which was brought off was exhibited.)

Soda Crystals.—Over 30 tons of crystals were manufactured at the experimental works in Main Street. A sample taken by myself was tested, and I give the analysis of others, for comparison:—

	S.A. Alkali Ld.	Tyneside Samples.		Theoretical Pure.
		Allhusen	Tennant	
*Sodium Carbonate	38·40%	34·19	34·19	37·07%
,, Chloride	1·19	0·10	·05	—
,, Sulphate	·12	3·40	2·21	—
* = Real Soda (Na_2O) ...	24·46%	23·00	20·00	21·68%

Soda Ash.—The writer prepared a sample of soda ash by desiccating crystals (as above) and obtained an ash testing:—Sodium Carbonate 96·40% = available Soda (Na_2O) 56·38%. Such a product should prove useful in South Africa for many industries.

Refined Alkali.—It is hoped to manufacture a purer product, granulated and with a mere trace of chloride, to be sold at a higher price.

Caustic Soda.—From Soda Crystals (made from the calcined product) I prepared a small batch of Caustic Soda, working under several disadvantages; however, the caustic so made tested:—

*Sodium Hydrate = 88·13%
* = Available Soda(Na_2O) 68·3 %

The Alkali Trade.—Dr. Moir, in his valuable presidential address,[*] thought it would be a good thing if an alkali works were started in South Africa. He suggested that some of the saltpans in this country might be treated for the production of caustic soda and hydrochloric acid. Presumably, he had in mind an electrolytic process.

The Leblanc Process.—In this process, coarse-grained "chemical salt" is treated with sulphuric acid, sulphate of soda and hydrochloric acid being obtained. The sulphate of soda, mixed with limestone and small coal is converted in revolving ball furnaces into "black ash," which on lixiviating gives solution of sodium carbonate with insoluble "tank-waste" containing chiefly calcium sulphide. From the crude sodium carbonate solution, soda ash or crystal may be manufactured; or, by causticising with lime and evaporating, caustic soda may be produced.

The Ammonia-Soda Process.—This is founded on the conversion of ammonium bicarbonate and common salt (or brine) into sodium bicarbonate and ammonium chloride, according to the equation:—

$$NaCl + NH_3 + H_2O + CO_2 = NaHCO_3 + NH_4Cl$$

The first patent, based on this reaction, was that of Messrs. Dyar and Hemming, in 1838, the process being carried out for a while in Cheshire. Then from 1855 to 1858, Schloesing and Rolland experimented near Paris, producing about 300 tons of soda in two years. It remained for M. Ernest Solvay, in Belgium, to make a great commercial success of this process. In England, Messrs. Brunner, Mond & Co. working Solvay's patents, near Northwich in Cheshire, have also been highly successful and have paid, for many years, handsome dividends to their shareholders. As showing how the ammonia process has gained upon, and overtaken, the older Leblanc process,

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[*] "The Indirect Advantages of a Slimes Plant," this *Journal*, Volume II., p. 657.

[*] This *Journal*, July 1910, p. 9.

I quote the following figures showing the world's production of soda ash[*] :—

Years.	Total Production.	Leblanc Ash.	Ammonia Soda Ash.
From	Tons	Tons.	Tons.
1864-1868	375,000	374,700	300
1884-1888	800,000	515,000	285,000
1889-1893	1,023,000	538,000	485,000
1894-1898	1,250,000	570,000	680,000
1899-1900	1,500,000	600,000	900,000
1900-1903	1,750,000	150,000	1,600,000

It is somewhat remarkable that up to the present the manufacture of soda ash has been practically confined to the Northern Hemisphere. Since about 1885, it may be said that the Leblanc process has managed to survive because of the world's demand for chlorine, in the form of bleaching powder or chlorates. The recovery of good marketable sulphur, both "block" and "flowers," by the Chance-Claus process, from the tank-waste of the lixiviating vats, has also helped the Leblanc process. The writer can remember when there were some thirty Leblanc alkali works in the N.E. of England, from the Tyne to the Tees; competition with the Solvay process has been so keen that, now, only two Leblanc and one ammonia-soda works are operating in that district.

The United Alkali Co., Ltd, represents a great "combine" of British Leblanc manufacturers. Formed in 1890 with a capital of £8,218,670, it included some 48 works in Great Britain and Ireland. Within about a year, something like half of those 48 works were shut down, by a system of centralising and giving orders to the best-situated works. Now, some, 20 U. A. Co. Leblanc works are still operating and the same firm has ammonia-soda works at Fleetwood.

Electrolysis.—Another competitor to both the processes I have mentioned, is the electrolytic method. By this, brine is decomposed into sodium and chlorine. The metallic sodium forms an amalgam with mercury, which is used as the negative pole. This amalgam flows into a separate compartment of the electrolytic cell, where it decomposes water, forming a solution of almost pure caustic soda. Carbon is used as the positive pole and the chlorine evolved there is led off into chambers for the manufacture of bleaching powder.

The Castner-Kellner Alkali Co. has been the most successful firm with this process. Their works at Weston, near Runcorn, Cheshire, started 17 years ago, now employ 600 men, in the manufacture of bleaching powder, caustic soda, sodium peroxide and liquid chlorine, and are being extended. This firm has large works in Germany, also a sodium plant on the Tyne, and is interested in the manufacture of sodium cyanide, by the Cassel Cyanide Co. of Glasgow, according to Mr. H. Y. Castner's patents. Another firm, the Electrolytic Alkali Co., which worked the process of Hargreaves and Bird at Middlewich, Cheshire, has recently collapsed.

In delivering the Ferdinand Hurter Memorial lecture to the Liverpool Section of Chemical Industry[*] in 1911, my friend Mr. Max Hasenclever said :—We often read that the old Leblanc process has almost entirely been replaced by electrolysis. It is however not sufficiently known that manufacturers of Leblanc soda have also been progressive and improved their proces up to quite recently. Instead of the old, we may call it, today, the new Leblanc process. . . . . In my opinion it would have been more economical to produce by the Leblanc process all the electrolytic products derived from the chlorides of sodium and potassium, as far as they are based on fuel."

Imports of Alkali into the Union of South Africa[†] :

	1910.	1911.	1912.
Caustic soda, value	£23,834	£20,637	£29,898
Washing Soda, *i.e.* "crystals" ...	634	450	578
Soda, not otherwise described ...	14,521	15,182	20,671
Total ..	£38,989	£36,269	£51,147

Soap.—It is interesting to note here that the imports into the Union, which amounted to a value of £187,285 for 1911, fell to a value of £112,367 in 1912; the result of South African manufacture.

Soda Ash from Natural Soda.—Dr. Lunge writing in 1909, quotes the fact that 22,000 tons of soda ash from natural soda, are produced annually in the States of Nevada and California.

Recent Exports from United Kingdom.—The following figures showing exports of heavy chemicals from the United Kingdom are from *The Times*[‡] :—

	Soda Ash,	Bleaching Powder.	Caustic Soda.
	Tons.	Tons.	Tons.
Yearly average, 1880-85	164,385	77,509	97,013
,, ,, 1900-06	58,211	34,732	77,509
,, ,, 1907-12	97,952	39,206	77,098

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[*] Dr. G. Lunge's "Sulphuric Acid and Alkali," Vol. II., Pt. II., p. 500.

[*] See *Journal Society Chemical Industry*, Vol. XXX., p. 1293.

[†] See Blue-books, "Trade and Shipping" for 1911 and 1912, Union Government of South Africa

[‡] See *The Chemical World*, Aug. 1913, p. 241.

Hostile tariffs abroad and a great rise in the price of small coal, at Home, have injuriously affected British alkali-makers during the past 20 years. It is often cast as a slur on this newest Dominion of the British Empire, that we import more per head of white population than any other British colony. It may be that the large native population somewhat helps to swell those imports, for we know how the Kafir delights to ape the European. In California and Australia, the population (attracted there, firstly, by the magnetism of gold), has developed agricultural and other industries. We see such developments beginning here, already. As good South Africans, we hope in the near future to increase our exports; meanwhile let us remember the advice of Carlyle: "Produce! Produce! Were it but the pitifullest infinitesimal fraction of a product, produce it. . . . 'Tis the utmost thou hast in thee: out with it, then."

Mr. A. Whitby (*Member of Council*): I should like to propose a very hearty vote of thanks to Mr. Watson for this paper This is a very interesting paper, and shows earnest advocacy of a new South African industry. It is, however, long, and deals with so many points, that it is impossible to discuss it in detail to-night. One point I note is with regard to the question of using crude soda as a neutralising agent for mill solutions. This is extremely controversial, and, although it may neutralise acidity, carbonate of soda is not the coagulant that lime water is, and the latter is necessary in slime settlement.

Mr. J. R. Williams (*Past-President*): I beg to second the vote of thanks. I do not think any of us are prepared to discuss this paper to-night, but I feel sure it will lead to interesting discussions at future meetings.

EXPLOSIVES IN HAND AND MACHINE LABOUR STOPES.

By T. D. Delprat (Member).

Looking through any number of the Rand Mining statistics one will note that a greater quantity of explosives is required in machine stopes than in hand labour stopes to break a ton of reef, for instance:—

Modder B.—Cost of explosives per ton of reef broken in hand labour stopes 6·6d., *ditto* machine stopes 10d., increase 54%.

City Deep.—Cost of explosives per ton of reef broken in hand labour stopes 11·1d., *ditto* machine stopes 16·7d., increase 50%.

Rose Deep.—Cost of explosives per ton of reef broken in hand labour stopes 10·6d., *ditto* machine stopes 14·0d., increase 34%.

Geldenhuis Deep.—Cost of explosives per ton of reef broken in hand labour stopes 11·4d., *ditto* machine stopes 16·8d., increase 47%.

Durban-Roodepoort Deep.—Cost of explosives per ton of reef broken in hand labour stopes 8d., *ditto* machine stopes 12·6d., increase 57%.

If one takes into consideration the fact that usually machine stopes are wider than hand labour stopes this discrepancy becomes even more surprising, for comparing wide and narrow hand stopes or wide and narrow machine stopes the conclusion one invariably arrives at is that less explosives are required to break a ton of reef in a wide stope than in a narrow one. Take for instance the following mines working under approximately the same conditions as regards nature of reef:—

Robinson Gold Mining Co.—Average width of hand labour stopes 65 in., cost of explosives per ton 7·3d.

Village Main Reef.—Average width of hand labour stopes 43 in., cost of explosives per ton 12·4d.

Robinson Gold Mining Co.—Average width of machine stopes 92 in., cost of explosives per ton 7·4d.

Village Main Reef.—Average width of machine stopes 58 in., cost of explosives per ton 16·2d.

These figures include the cost of fuse and detonators. A machine stope producing the same tonnage as a hand labour stope uses less fuse and fewer detonators, but as these are details of secondary importance they will be left out of this consideration.

That the difference of cost in explosives per ton is a matter worth looking into will be shown by the following example:—

City Deep.

Cost of explosives per ton of reef in machine stopes	16·7d.
Cost of explosives per ton of reef in hand labour stopes	11·1d.
Difference	5·6d.

Tonnage from machine stope say 23,000 monthly, 23,000 × 5·6d. = £480.

The same money otherwise expended would serve to break 1,400 tons of reef.

The average number of inches drilled per shift per boy is also of influence on the cost of explosives per ton of reef broken for instance.

Crown Mines.—Drilling average 42 in., explosives per ton 12·5d.

Bantjes Consolidated Mines.—Drilling average 35 in., explosives per ton 14·0d.

The drilling average depends on the efficiency of the boys employed. When a number of raw boys come in, fresh from their kraals, this average declines. A boy reaches his full efficiency about 6 weeks after his first shift below. I cannot find any data which would enable me to compare drilling averages and cost of explosives in machine stopes; more likely one would find that there also an increase in the drilling average would be accompanied by a decreased cost of explosives per ton. Machine holes on an average being deeper than hand labour holes, one might be led to expect a saving of explosives in using machines, while, as already pointed out, the reverse is the case.

After the detonator has exploded the top cartridge of a hole the ensuing chemical action is completed very quickly, but it does not take place simultaneously in all parts of the charge. This action travels from top to bottom by a wave of pressure, known as the "Explosive Wave" It has been found that space is necessary for the propagation of this wave and that both its power of penetration through the unexploded part of the charge and its speed are functions of the section of the cord of explosive in the hole.

I now enumerate some of the chief advantages which machine holes to be blasted have over hand labour holes :—

1. The hole is deeper.
2. Its section is large.
3 Stope faces into which machine holes are drilled are wider.

Yet nothwithstanding these three important advantages more explosives are used to break a ton of reef in a machine stope than in a hand labour stope. If the explosives were to be as efficiently applied in the former case as in the latter the reverse would be the case, and this allows one to believe that there is room for improvement.

The following causes I believe are the most important ones which counteract the advantages gained by use of machines. Let me here explain that I speak only of large machines; small machines in all statistics occupy an intermediate position between hand labour and machine labour. The causes referred to are :—

1. Machine benches practically always carry more than two holes; the outside two are charged heavier than is necessary to ensure the formation of a large free face, which again facilitates the breaking of the remaining holes. In actual practice however it is found that the sockets invariably left by the holes blasted first handicap the remaining holes considerably. In hand labour stopes this disadvantage is not so pronounced because benches carry one hole only.

2. At the end of a shift a machine miner generally has 12 holes to charge, while the hand labour miner may have 30. It follows that the machine man attaches at least two and half times as much importance to each of his holes as the hand labour miner does to each of his. If a machine hole breaks badly there may be one less bench available for the next shift, and this is naturally a more serious matter than the loss of a hand labour bench. The temptation therefore exists for the machine man to overcharge his holes. If he be a careful man and have twice as many benches as machines he should blast no more than two holes per bench per shift, thus saving the powder of the inner holes should the outer holes break badly or missfire. This procedure would also enable him always to see the actual burden of the hole he is about to blast, and this of course is a great advantage when he estimates the quantity of powder required.

3. The third cause is to be found in the manner of tamping generally adopted. Let me illustrate my point by an example. Assume that 6 in. of tamping is just sufficient for a hole $\frac{7}{8}$ in. in diameter. It will not require much thought to understand that this depth of tamping would be quite inadequate for a hole 18 in. in diameter, should it ever be necessary to charge a hole of such absurd dimension. As the section of the hole increases the depth of tamping must also be increased. If the diameter of one hole be 1 in., and of another 2 in., then the depth of tamping in the latter hole should theoretically be twice as much as in the former. This of course only applies to holes of cylindrical shape, but one should remember that a hole made by machine labour is of more pronounced conical shape than that made by a hammer boy, and that the tamping in it is therefore at a decided disadvantage. It is possible that the quantity and quality of tamping generally used for machine holes does not offer sufficient resistance to the pressure put thereon, and that considerable leakage and loss of pressure result. That insufficient tamping leads to bad results is demonstrated when breaking rock by plastering or blistering, the resistance then offered by the tamping is merely its weight or inertia, and the efficiency of the explosives used is consequently very low.

4. It stands to reason that when four holes are drilled by a machine from one position of the bar, these holes can not all be placed in such a manner that each of them takes full advantage of slips and cleavage planes in the reef. It must also be true that when a bench only carries one hole as in hand labour stoping the position of this hole can be made nearly perfect, but I doubt very much whether the average miner be suffi-

ciently observant to secure the greatest possible benefit from such natural advantages, and assuming even that he have the necessary skill, the opportunity for exercising it would not occur more than once per stope per shift.

5. The velocity with which the explosive wave travels through the charge is not uniform, it gradually diminishes, and thus the explosion becomes less and less violent as it approaches the bottom of the hole. The first few cartridges decompose at a much quicker rate than those deeper in the hole, the gases evolved last are therefore subjected to cooling and leakage for a longer space of time than those liberated at an earlier stage of the chemical change. The pressures in the lower part of a hole are never as high as those in the top part, and a machine hole being deeper than a hand labour hole is, in this respect, at a disadvantage.

To lower the cost of explosives per ton of reef broken the following suggestions are offered :—

1. Always use a large detonator, certainly not smaller than No. 8. By these means the powers of penetration and velocity of the explosive wave are increased and though this wave will still "tail off" its pressure will never fall so low as when small detonators are used.

I note that the sale of large detonators is increasing in the Transvaal, and hope that the small detonator will soon be a thing of the past.

2. Use electric detonators and use this detonator in the bottom cartridge, reversing the usual method of charging. By these means one ensures that the most violent part of the explosion shall coincide with the region of largest burden.

I understand this method was used for sinking a shaft 2,000 feet deep in a mine east of Johannesburg. Ordinary fuse was used, but this would appear risky as the fuse might set fire to the explosives before reaching the detonator.

3. Make careful tests in many mines to ascertain if it be absolutely necessary to use blasting gelatine to such an extent as it is at present. I am under the impression that a gelignite containing a high percentage of nitroglycerine would in many cases give excellent results. The cost of such an explosive would of course be lower than that of blasting gelatine.

4. In machine stopes the tamping should always reach to the collar of the holes.

5. Offer some monthly reward to the machine and hand labour miner whose explosives bill is lowest per ton. In order to make these costs comparative a formula must be framed which takes into consideration stope width and drilling average.

Not long ago much was expected from an Italian process. The idea was to mix finely powdered aluminium with the explosives. As is known, nitroglycerine ($C_3H_5O_3[NO_2]_3$) is slightly over oxidised, and the idea was to allow this O to combine with the Al into Al_2O_3.* Al is used for some such purpose in the Goldschmidt process. The surplus of O contained in nitroglycerine is, however, needed for the oxidation of the collodion cotton and parchment paper present, and such a mixture would give rise to an excessive amount of CO. This, of course, would be most undesirable. It may be that the difficulty can be overcome by a further addition of nitre. It is not difficult to find the correct proportion in weight between nitre and aluminium, but this mixture produces but little gas. Adding this nitre-aluminium mixture to blasting gelatine therefore reduces the weight of the gases evolved per unit weight of explosive, though raising the temperature at which they are produced. The calculation of the proportion between blasting gelatine and nitre-aluminium mixture which would give the highest possible pressures then becomes a rather involved problem of maxima and minima, which I found too laborious to work out.

It is, of course, clear that the cost of explosives per ton does not determine the cost per ton broken although it is an appreciable part of it. Briefly my point is that if a ton of reef can be broken in a narrow hand stope for, say, sixpence worth of explosives, one ought to be able to do as well in a wide machine stope, other considerations remaining the same.

I hope discussion will bring out some interesting, points and that where I have made mistakes I shall be let down gently.

Mr. J. Chilton (*Member*): I have great pleasure in moving a vote of thanks to Mr. Delprat for his most interesting paper. As he criticises my work, I intend, at a later date, to submit a defence.

Mr. W. Waters (*Associate*): I have very great pleasure in seconding the vote of thanks. Great attention and study are given on the Rand to the question of gold recovery on the surface, but it has often occurred to me that too little attention is paid to the science of blasting out the rock underground. The author has afforded some useful information on the latter, and the only fault I have to find with him is that his paper on the subject is, if anything, too short.

Mr. E. M. Weston (*Member*): I should like to add my thanks to the author for his most interesting paper. I am afraid, however, he has been doing what I have done on several occasions, that is, he has started to discuss a subject without reading up authorities first, or he would have found out that there is a definite law governing the relation of charge to burden. The amount of explosive used increases with the square of the burden on the hole. That, of course, is the main reason that when deeper holes are put in with a larger burden on them, one is naturally bound to use more explosive than where shorter holes are put in with less burden. I should advise the author to look up former discussions on this subject in the *Journal*.

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*This oxidation is accompanied by a large liberation of heat which should serve to produce larger pressures in the resulting gases.

The President: I must congratulate the author on this, his first paper to our Society. I think he will find that the main difference which causes the higher consumption of explosives with machine work lies in the interdependence of machine holes as compared with the independence of hand holes; and that the extra amount used in machine stoping may be justly regarded as an insurance against the interruption of an important chain of events.

MINING COPPER ORES AT MESSINA.

(Read at August Meeting, 1913.)

By J. Allan Woodburn, M.Inst.M.M. (Associate).

DISCUSSION.

The President: I am sure you will all join with me in welcoming Mr. Allan Woodburn, the manager of the Messina mine, who is present with us to-night. His paper on the mine was most interesting, and I am sorry that distance prevents him from visiting us oftener.

Mr. Allan Woodburn (*Associate*): It was very encouraging to read in the *Journal* the complimentary remarks regarding the paper which I submitted in August, and I wish to express my thanks for the expression of welcome which has been given me to-night. If the paper was appreciated by some, and if it has been of any value to the *Journal* of the Society, I am more than satisfied. As is well known, the Messina mine is the only producing copper mine in the Transvaal, and, as such, I thought a description of it might be of interest to members.

Since writing the paper, the new plant has had more than a trial run, and although we are still waiting for the heavy rains, which have fallen here, but which have not yet reached us in the north, and although we are rather handicapped by a scarcity of water, yet in October we were able to produce 600 tons of concentrate, assaying close on 50% copper, which output was of the value of fully £15,000; several hundred tons having been transported by ox wagon and donkey wagon to the rail-head, which is still about 20 miles away from the mine. We are looking forward to having the railway actually at the mine within the next month—certainly, we hope, before Christmas. Within a few months we are hoping to reach a total monthly production of close upon a thousand tons of 50% concentrate.

The paper just read on the Soda Deposits of Africa causes me to bring forward rather an interesting point, which we found out during the last month in connection with the furnaces which have been running since the beginning of July.

We required a certain amount of lime for fluxing purposes, and locally there is a certain amount of lime lying around on the surface which we have used for the past five years for building purposes. Unfortunately there is a large percentage of silica in this lime, and it was not so suitable as a chemical flux. We tried a truck load of lime from Pretoria, and found it suitable, but in the meantime we found that the ash from the boilers contained a very high percentage of lime, and this boiler ash is now being used as a flux in the furnaces for copper smelting, and we find it very satisfactory. This has enabled us to cease obtaining lime from a considerable distance, as we have a large quantity of this wood ash from the boilers available.

I have again to thank you very much for the manner in which you received the paper, and for the pleasant welcome to-night

VENTILATION OF THE MINES OF THE RAND: THE PROBLEM OF OBTAINING HEALTHIER CONDITIONS.

(Read at September Meeting, 1913.)

G. H. Blenkinsop, M.Inst.M.M. (Associate).

DISCUSSION.

Mr. J. Gibson (*Associate*): The author quotes largely from reports, and *inter alia* makes certain assumptions and draws conclusions which are not so far found to be correct in actual practice. With regard to underground temperatures it is well known that these do not vary in the deeper mines to anything like the extent experienced on the surface. The rock temperature at say 2,500 ft. vertical is about 75° F. and the air temperature in the workings adjacent is much the same. The author assumes (for the Rand) 1° in 200 ft. as the rate of increment, which is much above observed rates and applies this rate to both vertical and longitudinal distances, thereby confusing a natural rate of increase for depth with the increment in a current traversing workings longitudinally. This latter increment for any particular level is hardly apparent.

It is generally admitted and understood that the evil effects of humidity in the air are largely nullified when means are provided to remove the bodily heat as fast as it is produced. This can be done by having a good current of air to carry off the heat produced by the exertion. It is also well known that it is not the actual temperature of the air, but the relative humidity which affects the persons working. This of course applies only up to certain limits. To strive to create satisfactory conditions of ventilation (more particularly in places where the temperature rises near to or above that at which detrimental results ensue) should be our aim, and I believe that the history of the methods introduced into our mines within recent years bears strong evidence that this is being attained. It is hardly fair to condemn fan ventilation, because it does not happen to be a direct method of supplying fresh air to working places, and to offer as, the author does, a system of direct delivery by use of blowers and ducts. Fan ventilation as practised on the Rand can be fairly described as a method of exhausting the foul air and indirectly supplying fresh air by induced currents. To my mind it is preferable to the author's proposals whereby all travelling ways, etc. would be the routes for exhaust air and gases and all other workers except those in the immediate vicinity of the delivery pipes would have to work in vitiated air. It would seem therefore that where ideal methods cannot be attained without prohibitive expense or for other causes, we must have recourse to the most reasonable compromise. In the case of the mines of the Rand, this is to have all shafts used for hauling and travelling converted to downcasts, using one or more as the upcasts. In some cases the outcrop mines have been used for this purpose, in others, one of the main shafts has been used or some abandoned shaft, whilst in the case of the City Deep a large circular shaft purely for this purpose is now being sunk, and for new mines it would be difficult to devise a better system, as this ensures good fresh air to all the working shafts, stations and levels, and no waiting and but little work done in the return airways.

I might here record that over 30 years ago my late father, in his capacity as consulting engineer to many of the mining companies working the deep alluvial leads of Victoria, was obliged to install several systems similar to that now proposed by the author for the Rand. But the conditions are so dissimilar that comparison is out of the question. In the deep alluvial mines mentioned the leads were very wet, in some cases taking years of pumping to drain them and they were also charged with large quantities of gases, chiefly carbonic acid gas, which in days of low barometric pressure issued into the mines and for days no work could be done. Obviously no system of exhausting could possibly be of any use, and this led my father to introduce a pressure system by installing one or more Roots' blowers with galvanised piping reticulating through the workings. On very bad days, however, it was not uncommon to find this was not altogether satisfactory, and whilst the shift could go down not much work would be done. It did, however, give satisfaction generally, and almost entirely did away with the loss of time previously occurring at frequent intervals.

To advocate a pressure system on the Rand, however, where totally different conditions prevail, and indeed where the workings are so extensive, does not appear to offer the same chances of success as the system of exhausting by fans.

Since writing the above I have read the contributions of some other members, and notice a reference to the effect of falling water on the temperature. This effect is very evident at the No. 4 shaft (downcast) of the Geldenhuis Deep, Ltd. where water is almost continually falling in the vertical portion. This is about 1,200 ft. deep, to the 6th level below which the bend occurs. At this level the mine temperature varies from 55° F. to 62° F., and no higher readings have been observed even at midday during very hot weather when the surface temperature may be 74° F. or higher. This shews a cooling effect of about 12° for a hot day, whilst on a very cold day the air going down would, if at a lower temperature than 55°, be warmed up to this degree the lowest reading recorded during cold weather.

Mr. H. C. Boydell (*Member*): The system advocated by Mr. Blenkinsop, ventilation by Roots' blowers, has been tried in Australia, chiefly in the deep level alluvial mines of Victoria, and amongst others, was investigated by the Mine Ventilation Board in the years 1899-1900. The whole matter is dealt with in the reports issued by that Board, and the following are a few of the results obtained in tests made with blowers.

At the Berry Consols Mines at Allendale (near Ballarat) two No. 4 Roots' blowers were in use. These, working under a water gauge of 17 in., had a combined capacity of 5,310 cu. ft. per min. at the blowers. The amount of air actually delivered at the faces amounted to only 895 cu. ft. per min., there thus being a loss of 4,415 cu. ft. per min., equivalent to approximately 83%.

In another test made in the No. 4 Roots' blower at the No. 1 shaft of the same mine, the quantity of air given out by the machine was 2,003 cu. ft.

per min. under a water gauge of 21·25 in., whilst that received at the different faces was only 1,015 cu. ft. per min. An "air lock" was used in this mine to maintain a constant pressure of the air in the workings. The barometer at the surface read 25·34 in., whilst that in the air lock was 29·85 in. The total length of main air pipes, not including the smaller ones to the different faces, was 2,700 ft. This consisted of one 11 in. pipe and branched off into two 7 in. ones.

At the No. 2 shaft of the same mine a No. 4 Roots' blower supplied, under a water gauge of 25·3 in., 2,118 cu. ft. per min., of which only 1,227 cu. ft. per min. found its way to the faces. The barometer was 29·42 in. at the surface and and 30·35 in. in the air lock. The length of main pipe, not including the smaller ones to the working places, was 3,135 ft., the dia. of which is not given.

At the Charlotte Plains Consolidated Deep Levels, Moolert, Victoria, a No. 9 Roots' blower was used. This was connected to 2,175 ft. of 14½ in. pipe, part of which led down the shaft 337 ft. deep. Then followed a reticulation of 1,550 ft. 11 in. pipe, 1,000 ft. 10 in., 2,270 ft. of 7 in. and 90 ft. of 5 in, making altogether 7,085 ft. of pipe. The blower working under a 28½ in. water gauge delivered 4,630 cu. ft. per min., of which only 1,136 cu. ft. per min. reached the working faces of the mine.

At the South Star Mine, Ballarat, a No. 3 Roots' blower was connected to 11 in. pipe led down the vertical shaft 3,100 ft. deep and then 130 ft. along a cross-cut and level. The blower ran under a water guage of 18·7 in. and had a capacity of 3,830 cu. ft. per min., of which only 510 cu. ft. per min., approximately 13% was delivered at the face. The temperature at the surface intake was 52° F., whilst that of the air reaching the face was 89° F.

A noticeable feature of all the tests was that no matter what the temperature of the air at the blower intake was, it reached the face at the temperature of the latter place, thus showing that ventilation by blowers does not necessarily mean a supply of cool air in the workings. In all the above cases reasonable care had been taken to keep the air pipes and their joints in as good condition as can be maintained underground. A number of results, such as the above, caused the Board to report *inter alia* (Final report p. 6): "The mine managers appear to have assumed that nearly all the air pouring into a Roots' blower at the surface reaches the men for whom it is intended. It is clear however from the evidence, that, owing to the way in which the air pipes have to be put in and owing also to their length and often unavoidable leakage, the proportion of air that reaches the men is often but a small fraction of the air sent from the engine room. *In all cases the amount of air supplied was quite insufficient to keep the roadways and faces in a fresh condition.* This emphasizes the opinion, expressed by the Board before and reiterated here, that the Roots' blower will be found suitable *only for mines where a few men are at work, or as an auxiliary appliance to other methods of ventilation.*"

The effect of the oxidation of pyrite on the temperature of mine air, referred to by the author, is surely negligible. The rate of oxidation under ordinary conditions underground on the Rand is so slow that it would form a good example to illustrate the difference between quantity of heat and temperature. The only possible effect that it could have on the air would be a withdrawal of oxygen and consequent increase in the proportion of nitrogen. The decay of timbers likewise can have no measurable effect on the temperature of air underground generally. As in the case of the oxidation of pyrite, it would lead to abstraction of oxygen and thus raise the quantity of nitrogen present, but not necessarily produce CO_2. In any case, influences of this kind on the temperature of the air are more than counterbalanced by the large amount of low temperature evaporation that goes on in underground workings.

Dr. J. Moir (*Past-President*): In connexion with this paper I wish to call your attention to an important article on "Temperatures in Mines in Great Britain," by Prof. J. Cadman*, and to its bearings on labour efficiency. The Table on pp. 515-517 should be carefully studied by everyone who has the future of the Rand at heart and who realises how much that future may be curtailed unless improved efficiency is obtained. Judging by these English results, a temperature of only 75° F. (wet-bulb) produces a marked 'slackness,' and if we consider that our deeper mines have already higher wet bulb readings than this, we see that this question of efficiency will become urgent in the next year or so: this is why I think that there is something in the basal idea of Mr. Blenkinsop's paper, faulty as it is, namely that *local* ventilation of hot places is at present too neglected, and could be greatly improved by a scientific *suction* method, which would only require the movement of a very small volume of air, say only 1% of the output of some of our big fans, though fan ventilation would still be required to keep the main airways clear. The author unfortunately gives so few details that no one can say if his proposals are scientifically sound or not: it is to be hoped, anyway, that they are sounder than his remarks on mine-

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**Trans.*, I.M.E., 1913, part 3, pp. 512 to end.

temperatures and on underground physics generally.

I also insert the following important *data* on the four factors which govern underground temperatures (Cadman's paper, p. 512): (1) increase of rock temperature with depth, here say 4½° F. for each 1,000 ft. vertical: (2) increase due to oxidation (of pyrite, etc.) equal to say 3½ calories per cc. of oxygen absorbed: (3) increase due to compression of air in descending the downcast, say 5½° F. per 1,000 ft. vertical: and (4) decrease due to taking up moisture (chiefly in downcast shaft), equal to about 3° F. for every grain of water evaporated into a cubic yard of air. The last is the main favourable factor here, but it disappears entirely during summer rains, which pleasant as they are to the town-dwellers, bring only bad ventilation and higher temperatures to the mine workers.

Mr. Allan Woodburn (*Associate*): I had written out a criticism on this paper a week or so ago, but after reading the transactions of the last meeting, and hearing the discussion just submitted, I find a goodly number of my remarks have been anticipated, so it is unnecessary to go over the same ground.

I am firmly of the opinion that ventilation by blowers is impracticable and excessive in cost, but there is one point which, I think, has not been touched on with the amount of importance that should be attached to it. This is not the question of blowers, and not the question whether a certain amount of air should go down the down-cast and up the up-cast, but how it is to be distributed throughout the mine workings.

The subject of ventilation in mines is a most important one, and any paper which promotes a discussion on this most interesting subject is very welcome.

The author appears to be rather sweeping in his remarks regarding fan ventilation, which has been proved to be so efficient in other mines as extensive as those on the Rand. He appears to ignore the very important item of splitting the air current into as many sections as possible, with the object both of reducing the velocity of the air in the airways, and the total horsepower necessary to pass the total quantity throughout the mine. If the total quantity is split up sufficiently a low enough velocity can be obtained to obviate the evils of dust raising, which he appears to think a serious defect in fan ventilation. His remedy is apparently a very satisfactory one of carrying fresh air in pipes right to the working face, but I agree with most of his critics, that it is an impracticable remedy both as regards the quantity of air which would be passed through the pipes with ¾ lb. to 1 lb. per square inch, or, say, 28 in. water gauge pressure, while the power required would be enormous.

The question of efficiently ventilating a large mine would be rendered much easier if the mine were originally laid out to suit; and when laying out the development of any large area, the question of ventilation should then be considered. It is immaterial whether the fan be placed on the surface or underground; the main point is to get a difference of pressure between the intake and the return, but very little benefit will be obtained even if 200,000 cub. ft. of air be passing down the down-cast and up the up-cast, if it is not split up and properly distributed throughout the working places. Of course one cannot expect to get ideal conditions, and compromises must be made, and other questions of rich and poor areas, faults, dikes, etc., have also to be considered, but methods may be found suitable which will greatly assist the ventilation and yet not retard the actual development and ultimate mining.

Some time ago I suggested main haulage roadways in the footwall, 20 ft. or so below the reef, and boxholes put up to the stoping level above, which would constitute a true level. If this were done the air could be split at each of these haulage levels and pass along to near the face, then up through the boxholes to the stopes: and as the stopes in the upper levels would be further advanced than the stopes in the lower levels, the air from the lower stopes would pass up through old stopes, and not join with the air from the higher levels until the top level was reached, which would be near the upcast shaft. Air crossings at places might have to be cut to allow the return from one section to pass over or under the intake of another section; and if the reef could be worked in anything like a regular manner, much could be done by diagonal tram roads from the stoping level up to the face, and using self-acting haulage if double roadways would stand. If a single roadway could only be used, then a balance weight at the side would enable the full truck to be lowered and the empty one filled up without putting such a severe strain on the brake wheel. On a self-acting roadway with, say, 20 cub. ft. trucks the full truck would probably weigh 25 cwt. and the empty one 5 cwt., making the weight to be held on the brake 1 ton. By using a balance weight on a single roadway of 15 cwt. there would be always a difference of 10 cwt. to be held back on the brake, which on steep roadways is an item of consideration. The single roadway would not have the same capacity as the double roadway.

I mentioned that it was immaterial whether the fan was placed underground or on the surface, because it has been stated that a fan should be used for exhausting and not as a blower, but some years ago a fan working on a colliery in Scotland, passing about 100,000 cub. ft. of air was for a time made to act as a blower by simply opening the ears of the fan to the atmosphere and carrying a casing from the fan chimney to the shaft, and although the air currents were by this process reversed, the total quantity passing was practically the same.

In dealing with large coal areas, down shafts are usually sunk as close together as the law permits, and the air is split into main sections as near the down-cast shaft as possible, and not allowed to join until as near the upcast shaft as possible. By this means very large areas are efficiently ventilated by a water gauge pressure of from 2 in. to 4 in., and the horse power required is not very great. It is not good practice to ventilate one large mine through another mine, but it may be better than relying on natural ventilation only. Of course, if the upper mine has old disused roadways or stopes through which the return air can pass, the evils may be reduced, but the great point to be arrived at is to give all the working places as much fresh air as possible and to help the return air from all the sections to reach the upcast shafts. Having been originally trained in coal mining practice I may be somewhat prejudiced in its favour, but when one considers the changes which have taken place by adopting methods of sand filling, haulage and ventilation, one must admit that coal mining methods are being adopted more than ever. It has always seemed to me that the Rand mines are peculiarly suited as gold mines for the adoption of coal mining methods; and I feel sure that in laying out fresh development areas if coal mining practice be aimed at, if not actually adopted, better conditions will prevail, both as regards health and safety, and also as regards efficiency and lower working costs.

Mr. E. Pam *(Member of Council)*: Several members have pointed out the usefulness of various methods of local ventilation. Probably the majority of mines use, for very bad places, some form of stove pipe into which they place a small jet of compressed air, or water under pressure. This has the effect, of course, of drawing in a certain quantity of free air from the intake; and it might be of interest to the members if I gave the result of some experiments I made on this subject.

Air Tests made at the Geldenhuis Deep, Limited, with Different Sized Nozzles Discharging into a 9 in. Pipe.

Length of 9 in. pipe, 538 ft. with four right-angled bends. Air pressure 72 lb. per sq. in.

	Size of Nozzle.	Discharge in Cubic Feet per Minute.	(Theoretical from Tables) Free Air per Minute Produced by Jet.
Flange off	$\frac{1}{16}$	210	5
	$\frac{1}{8}$	530	19
	$\frac{3}{16}$	770	43
	$\frac{1}{4}$	1,080	76
Blind flange on intake of pipe	$\frac{3}{16}$	30	These figures give the quantity of air as actually measured coming out of the nozzle.
	$\frac{1}{4}$	80	

It must be remembered that the conditions in the experiment were much more favourable than they would be in practice, the pipe being larger and straighter and the joints tighter; but this method is certainly more economical than ventilation by the escape of free air at the face of a working place.

ELECTRIC BLASTING.

(Read at October Meeting, 1913.)

By Wm. Cullen, (Past-President), Thos. Donaldson (Member of Council), and W. Waters (Associate).

DISCUSSION.

Mr. J. H. Veasey *(Member)*: The thanks of the Society, and of the mining industry, are due to the authors for this most comprehensive paper, and to those in charge of the Meyer and Charlton mine for allowing the result of their labours to be made public.

On the subject of electric blasting I am able to speak with seventeen years intermittent experience, consequently some of my views may be of interest.

The first thing apparent about the Meyer and Charlton installation is the amount of care and thought expended on the score of safety and efficiency, and the thoroughness with which every contingency has been provided for, there being no fewer than four safeguards againt premature blasting. This thoroughness was essential to success, as with the conservatism and prejudice commonly attributed to miners, an accident in the experimental stage would have gone far to condemn electric blasting with them.

I differ with the authors on the subject of testing fuses. I do not agree that it should be necessary to test fuses before use, nor that fuses should be supplied with flaws or breaks in the platinum bridge. When using fuses from stock of unknown age, it has always been my practice to test them in manner described. But if electric blasting is to become general, manufacturers will have to supply *tested*, or even *guaranteed* fuses, and in these days of advancement there should be no difficulty in this.

I would endorse the remarks of the authors on the blasting of the cut. I am of opinion that many cuts which to-day have to be blasted twice would come at the first time if blasted with instantaneous electric fuses. A small amount of experimenting should enable the charges to be so adjusted and the holes so placed, as to enable the round and the cut to be blasted together.

At the last meeting some criticism was raised on the costs given, in that interest and depreciation on cost of installation and the cost of shot firers were not taken into account when comparing the system with ordinary safety fuse blasting. Considering that electric blasting enables the whole mine to be blasted with every employe safe and sound on the surface, the cost of installation is not likely to be grudged; and in the matter of the shot firers it must be remembered that the system has but recently been installed. These "trained men" may be to-day acting the part of instructors: and there would seem no reason why, later on, their duties should not be divided between the miners and the shift bosses.

Now for a word for and against the miner. First against him. We heard a point raised the other night, "what about when the wrong hole goes off first?" If the wrong hole goes off first it can only be the miner's own fault. Safety fuses have to be cut for length. What more has to be done to delay action fuses than to cut for length? There is but one small difference, the delay fuse must be cut to length before the primer is inserted, so that the miner must also decide the rotation of his holes before inserting the primer, with safety fuse he can cut to length after charging up. Another grievance is the trouble (?) of twisting the wire leads together, and in a wet shaft of insulating the joint with prepared tape. Can the ordinary safety fuse, when cut for lighting, be allowed to get wet any more than the wires? In a wet shaft, sinking before water has been caught up, where is the miner who has not experienced a little trouble in keeping his fuses dry? What about the greasing and double fusing of wet holes? And how would the cost of this effect the comparison of costs given in the paper? Electric fuses are manufactured watertight, and require no grease.

In favour of the miner let me say, after fourteen years close contact with him, that I have not yet met one, who, with time and patience, could not be taught to successfully carry out electric blasting. A highly skilled electrician is *not* the man to teach electric blasting to a miner. The resistance of the fuses, voltage of the firing current and amperes available, are matters for the management only. All the miner need be told is the maximum number of fuses that may be connected in series, or, for simplicity, in one 'group.' He is then shown how to clean the ends of the wires, twist them together, and cover all bare wire or metal with prepared tape. Later, he can be shown how to test the circuit for continuity with the pocket galvanometer.

Our present blasting certificates are specially endorsed for miners competent to blast in fiery mines. Would it be out of place here to suggest that it would be an incentive for miners to become proficient in electric blasting, if a small examination were instituted covering such points as:—

1. Cutting and assembling delay action fuses.
2. To connect up and insulate one or more groups of fuses and connect to mains.
3. To give an intelligent explanation why:—
   (*a*) Ends of wires are cleaned before twisting together.
   (*b*) All exposed wire is covered with prepared tape.
4. To test for a complete or broken circuit with pocket galvanometer.
5. To explain what would happen when blasting if a group of wires had the covering (insulation) knocked off in two places and these two places were in water (slightly acid).

And on satisfactorily passing this examination his blasting certificate were endorsed:—

"Passed Examination in Electric Blasting."

Mr. E. J. Moynihan (*Associate*): I must congratulate the authors of this careful and able paper. The discussion of electric blasting for metalliferous mines, combined with single shift and surface firing, as an economic measure, and a preventive of disease, can only do good, whatever may be the final verdict on the system. Great credit is due to all concerned for the thorough prompt and energetic way in which the experiments and the permanent installation have been carried out at the Meyer and Charlton. And I should like to say that a great deal of that credit is due to the *Rand Daily Mail* for putting the suggestion forward, and sticking to its guns so energetically. But for that the experiment would not have been so promptly tried.

I think the authors have been a little too conservative in their remarks as to the feasibility of blasting the round at one operation. In view of experience gained during the last month I am sure they will themselves admit it. There are a number of possible ways of overcoming the difficulties supposed to exist, without sacrificing the advantage of simultaneous explosion of the cut holes. This gives a greater advance by some 6 in. per round than is usual with time fuse. Some of these methods are now being tried, and I have very little doubt of the result.

The criticisms advanced on the paper have not been particularly enlightening. Most of the critics have talked about what "might" or "would" happen, just as if the installation on the Charlton were a proposal instead of an accomplished fact. And our old friend the "practical" man is trotted out as is usual when any departure from present practice is to be discussed—and opposed: not by the "practical" man, but by those who boast of being "practical." Most of these criticisms come from Mr. Weston.

The self-styled "practical" man, when dealing with innovations has in general a bad record. He is usually a man who has never invented anything but difficulties himself. He is proud of the fact. He finds all sorts of objections on authority which has no existence. He is positively proud of the very worst points in present methods. He is an adept at foreseeing disasters which do not occur. And when he has to admit that there is some improvement in the new method, he tells us that any fool could have thought of that anyway.

The "practical" astronomers of his day put Galileo in gaol. The "practical" engineers of the 17th century dubbed the Marquis of Worcester mad for building a steam engine. The "practical" carriers shut the motor car off the roads by law in 1830. The "practical" sailors and admirals scoffed at the idea of steam ships and iron vessels. They were helped even by James Watt—once he had joined the ranks of the "practical" men. "Practicality" is a disease deadly even to genius like Watt's. It made him oppose and hold back for 50 years all the modern improvements of steam engine and locomotion. The "practical" steelmakers expunged Bessemer's paper from their records so as not to make themselves ridiculous. The "practical" electricians of the fifties—with Lord Kelvin now enrolled as a practical man—scoffed at electric motors. The "practical" cable men scoffed at wireless, and the practical aeronauts and naval engineers scoffed at breech-loading guns, submarines and aeroplanes. If the "practical" man had always had his way we should all still be wearing fig leaves and throwing stones for our meat dinners.

According to Mr. Weston, the authors and the Meyer and Charlton people should have taken the advice of the "practical" Rand underground man. Then they would have never done this wicked thing. Well, I should think that is quite possible, if the "practical" man is really of the character ascribed to him by Mr. Weston. He tells us himself to "remember that there are more fools and irresponsible persons per square yard underground here on these fields than anywhere else in the world," and then chides the authors for not going to them for advice. This seems a trifle illogical.

This is not the only illogical point in Mr. Weston's criticism. For instance, he says in one place that electric blasting will *not* "remove the necessity of men having to travel through dust and fume-laden drives and stopes;" and later on he says, "there will doubtless always be found isolated areas in mines where electric blasting might be adopted with advantage to render any long travel through smoke areas unnecessary." Mr. Weston contradicts himself, and is apparently unable to see that his reasoning is self-contradictory. It is therefore not a matter for surprise that he should tell us first that he is "unable to follow the reasoning of the authors that the adoption of this system gives any longer working time in the faces," and then goes on to tell us his reason. His reason is that an alternative system would give the same advantage. This reminds one of the Irishman who said that the day was not as fine as he expected: and added, that he never thought it would be.

Real thought-out criticism is always welcome and valuable. But Mr. Weston simply tries to score lawyer's points. These have no place in scientific and technical discussions. Some of his opening remarks are quite out of place and unjustified. "The middle portion" of the paper, he says, "while useful as a summary of the methods employed, can be found in modern text-books, and calls for little criticism." This is hardly more than a mere lofty sneer. And it is not an accurate statement. The middle portion *cannot* be found in *any* modern text-books. It would have been more convincing and to the point—but still quite unimportant—if Mr. Weston had given us a full list of the books he referred to.

Mr. Weston ventures to say that the CO and CO_2 from ordinary fuse is negligible in comparison with other sources. He asks for experimental results as to smoke holding dust in suspension. If he will read up some of the latest literature on condensation he will find that vapours only condense and become visible in space that contains dust. He will find matter for study in connection with the electron theory. What experiment has Mr. Weston to produce in favour of his CO and CO_2 theory? He will find ample experimental evidence against his theory in a paper read before this Society by Mr. Cullen in November, 1908. He will also find it implied, a little further on, in his own criticism, because he says he rather likes the smoke from fuse "as giving a very good indication of the presence of noxious gases."

Mr. Weston talks of the dangers of electric leakage, "where there are powerful currents being led about underground for lighting, ventilation and haulage." What has that got to do with an electric blasting system in which no currents at all are being led about when there is anyone in the mine, and in which the voltage is less than half that used for domestic lighting in Johannesburg? He also tells us that the blast "would" cause an immense wastage of cable. Well, the answer to that is that it does not, as he can easily find out for himself. Then he says that the method undoubtedly *means* the employment of much skilled labour in maintenance. The answer again is that it does not. There is no more skilled labour on the Meyer and Charlton than there was before. There are no more miners—including the blasting gang—than before. The capital cost to be spread over the life of the mine with its amortisation and interest is another bogey. I am authorised to say that the whole capital cost of the Meyer and Charlton installation was charged to current monthly costs—and did not materially affect them.

The most illogical part of this criticism is the claim that all the advantage of the electric system can be obtained by a reorganisation on the present time-fuse basis. It cannot. The chief advantage of the electric method is that you can make and easily observe a cast-iron rule that general blasting is only to take place after *every* one has come to the surface: and that is not possible with any modification whatever of the system of lighting fuses by hand at the working face. The next advantage is that you can break more rock per hole electrically than by the other method, and I am authorised to say that the Charlton management is satisfied of the fact.

Criticisms such as those I have dealt with are like those made by the orthodox astronomers of the time of Galileo. When Galileo made his telescope and discovered the satellites of Jupiter he invited the "practical" astronomers to come and have a look at his new little moons. The astronomers said: "No—we are not going to look at them, or make ourselves ridiculous by looking through your silly telescope. There are no satellites of Jupiter. We have said so: and we have known it for years!"

Mr. W. S. V. Price (*Member*): If I was in doubt before about the advantages of electric blasting, I have at least one argument in favour of it. I have here the label taken from a packet of so-called safety fuse, which states that the rate of burning is from 90 seconds to 110 seconds per yard. According to that, a miner using 6 ft. fuses would have to allow for a possible difference of 40 seconds between his shots. With electric blasting, even if this fuse was used in the delay action, the difference over a few inches could be overcome owing to the shortness of the fuse used. A few days ago I had practically a round hung up owing to a shot going off out of its turn, and I know the fuses were cut, lit correctly, and in their right order, so it must have been the faulty fuse. I think it has been well proved that blasting a cut by electricity, where simultaneous detonation is assured, must be greatly superior under any conditions; and when electric blasting is used instead of an inferior fuse, it should make an enormous difference.

There is no doubt that electric blasting must have a great future before it; and although it may not suit every case, that is no reason why it should not be used where it is suitable. Of course, one must use common sense in blasting. For instance, if one were to blast simultaneously two sides of a stope which had bad hanging, there would probably be lots of trouble in the next shift, in fact, I know of one or two mines that would be practically finished if such a thing happened.

Mr. Pam has stated that only about 10% of the men in the mine travel through the smoke. I think he must include the boys in this estimate, for on most of the mines I have worked on 95% would be nearer the average of white men who have to stay in the smoke, while the other 5% are mostly shift bosses, surveyors and underground managers, who usually take good care to be out of the way in time.

With regard to one point raised by Mr. Roberts about the difficulty of priming holes with the delay action fuses in a wet shaft, I think that could be overcome by the miner who prepares the primers at the box. When tying the detonator he can leave a length of string, and beginning with the shortest fuse put one knot in it, two knots for the second, and so on. The same system could apply to a developer in a drive.

Mr. W. Waters (*Associate*): May I be allowed to offer an explanation on the point in question. The last speaker produced a label which states the rate of burning speed of the fuse to be 90 to 110 seconds per yard. The label is in accordance with Government regulations, and every brand of fuse imported into South Africa must be so marked. The Government regulation reads as follows:—

> "All safety fuse shall burn between the limits of 90 seconds and 110 seconds per yard, and must burn evenly throughout its length."

Mr. W. S. V. Price (*Member*): This fuse was manufactured, I believe, in Johannesburg—at all events there is only a local name and address given on the label. With other fuses there is usually stated exactly the number of seconds it takes to burn a yard, 90, 92 or 96, which makes a miner feel more certain of his fuse.

Mr. W. Waters *(Associate)*: I would like to state that no safety fuse is manufactured in South Africa.

GOLD PRODUCTION IN RELATION TO HUMANITY; A STUDY OF THE QUESTION: HAS GOLD DEPRECIATED IN VALUE?

(Read at April Meeting, 1913.)

By Mr. Eustace M. Weston (Member).

REPLY TO DISCUSSION.

Mr. E. M. Weston (*Member*): I am pleased that my paper has falsified Mr. J. R. Williams' dismal predictions and has brought forward so many critics, and I must thank them for showing that the members of this Society are interested in the vital questions of to-day, and are not wholly given up to feats of molecular gymnastics or the practice of metallurgical crumb hunting. I cannot deal with every point raised by each critic; but there are first a few hard knocks to be returned with interest.

Mr. B. I. Collings chooses to be severe in discussing my dream of gold miners getting £6 per oz. at some future time. The thought I had was of the intrinsic value of gold as apart from its legal value in currency. It is quite possible that this might so increase as to raise its price, as platinum has to-day risen owing to scarcity. Platinum has been used for coinage. Consider the case had the Russian government fixed its value by law. The amusing side of the matter regarding Mr. Collings' law is that apparently the bankers of to-day know nothing about it, and know so little about their business that they to-day attempt to perform the absurdity of buying gold "valued in terms of itself" at a premium. Quite recently gold has been bought by countries in urgent need of it (of which more anon) at a premium of ¼d. to 1d. over the legal £3 17s. 9d. per oz. Let me quote "But the Bank of England gets very little gold as other countries will give a higher price."* Mr. Collings' evident affection for theory and his neglect of historical and other facts prevent him knowing that this impossible feat has been performed before. Again let me quote "Whatever reason induced Cæsar to enhance the value of gold there can be no doubt of the fact." "Charles the V. also increased the value of gold." I am, however, inclined to admit that the statement I made that gold costs more per oz. to produce to-day may not be correct. It cannot be proved in any case owing to the difficulty in comparing cost and value at remote periods.

Mr. J. N. Normand's thoughtful contribution can be dealt with mainly in my general reply. Two points however require attention. Mr. Normand does not understand the position of affairs in India, nor is he acquainted with recent literature on the subject, or he would not attempt to argue that bankers are wrong when they attribute the present scarcity of gold partly to the drain to India. The *Statist* has recently published a report by an Indian banker on this subject. He shows that the hoarding of gold is mostly in the form of jewels or gold coins strung in necklaces, and is due to the religious and other customs of the country. The only way a husband can legally make provision for his wife or daughter is by presenting her with ornaments. These cannot be sold to pay the husband's debts, and the son cannot sell the widow's jewels. Personal credit is unknown, banking facilities are few, the mortgaging of property is cumbersome and expensive, so that jewels are mostly used as pledges for loans. This gold does not prove a basis for international credit. Mr. Normand is almost the only one of my critics who attemps to bring any figures forward to refute arguments founded on facts. Among my critics generally, argument and theories I find in plenty, but figures and facts are lamentably wanting. Many are apparently sincere worshippers of the god of things as they ought to be, and disdain to consider the god of things as they are. Unfortunately Mr. Normand's facts do not support his argument. He tries to prove that as the gold output has increased 400% and trade only 200%, prices must have risen owing to a proved surplus of gold. He has here fallen into a simple error regarding the rule of three. In 1894 the world's stock of gold was about 850 millions, used in financing the trade of the world. Since then only a certain proportion of the production has been devoted to coinage. If this amount has increased gold stocks to 2,000 millions the amount of increase is only 230%, a very different figure from my critic's 400% "rate of increase." Mr. Weston, says our critic, confuses cause and effect when stating that rise in wages causes rise in prices. This rise in labour cost is a real factor however, and one cause is the development of new countries, where owing to abundance of natural resources it pays capital to raise wages to attract labour. He writes again, "there is no evidence to show the gold absorbed by hoarding, the arts, losses, etc., has increased in a greater proportion to the total supply than in the past." I have already justified my assertions regarding India; but Mr. Normand will be startled to find what this "less rate" has been in the past. Again I quote from Del Mar, a table showing total production of gold and silver in Western nations, including 103 millions taken by the Portuguese from Japan :—

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*From Del Mar's "History of Monetary Systems,"

Year.	Cumulative Supplies to date.	Gold and Silver Coins Stocks.	Cumulative Consumption and Loss.	Per cent. of Consumption and Loss.
	Millions.	Millions	Millions.	Per cent.
1675	509	250	259	50
1828	1,441	313	1,128	78
1860	2,040	560	1,480	72
1883	2,853	700	2,153	75
1894	3,472	750	2,722	78

These figures show that nearly three-quarters of the production in the past has been lost, used, hoarded, or has drifted to Asia.

I have to thank Mr. John Watson for his friendly criticism, which generally supports my position. Mr. Cullen also is a friendly critic. In my final correction of proof I had modified the sentence regarding the downfall of the Russian Empire by striking out "tin." This however escaped the printer's notice. Of course shortage of gold could be only one cause among many causing the "decline and fall." Mr. Cullen's arguments as to shortage of money and waste of capital remain unanswered by any of my critics. Mr. Richardson and Dr. Lehfeldt maintain the so-called orthodox theory of the "goldites," which will be discussed later. Mr. Richardson gives a diagram of prices and gold production, and apparently set out to prove some relation between them; when however he examines them, we find him admitting that many other factors have influenced the course of prices; and even when Mr. Normand's remarks regarding the demonetization of silver are considered, the evidence of price records is not in favour of the "goldites" contention. When I predicted a shortage of the more valuable metals, I was but voicing the opinion of experienced mining engineers, like Mr. Hoover, who know the situation; and Mr. Richardson would find, should he set out to discover a payable mine, that the world has been well raked over, and that possibilities of new discoveries are small in comparison to those of the past.

Dr. Lehfeldt is evidently a typical "goldite," and I was quite prepared to be dealt severely with. We are all liable to fall into traps, and some of us dig them for our own feet; but, personally, I object to have traps dug for me, and then to be pushed in whether I like it or not. Let us consider Dr. Lehfeldt's first terrible example of my illogical arguments. It will be noted that I made two statements: firstly, that greater efforts were being made to produce gold, *i.e.* there were more people and more power being employed in this work, and, secondly (and incidentally) that gold cost more per oz. than it ever did. Now regarding the second statement, as I already have admitted, this cannot be proved. Dr. Lehfeldt disregarding my main argument, founded on the first statement, makes the general statement that if gold becomes expensive to produce, prices must fall. Mr. B. I. Collings, however, is of opinion that the cost of producing an oz. of gold cannot vary, and Mr. Normand states that the cost of producing gold can have but little effect on prices. The matter is not so simple as Dr. Lehfeldt would have us believe; and it is the habit of making the syllogism divided from accompanying factors to be considered, that has discredited political economy in the past. We have in practice to consider the speculative element of gold mining, and the fact that gold miners usually open new countries and are themselves withdrawn from the production of goods. This side of the question, which Dr. Lehfeldt did not attempt to deal with, shows that if, say, one million people are withdrawn from productive enterprise and are engaged in gold mining, the supply of goods is reduced and the demand for goods is at the same time increased, tending to raise prices, other things being equal, even if the expense of producing an ounce rises. There are two holes on the East Rand; in one a million has disappeared and in another about half that sum. Does Dr. Lehfeldt really believe that these *mines* have helped to lower prices? Let us turn to Dr. Lehfeldt's other example, "Increase of demand must tend to raise prices and shortage of gold to create financial panics," so I wrote. Thus writes Dr. Lehfeldt, "It is exactly the other way of course. Increase of demand (*for gold*) would reduce prices." Now I wish humbly to apologise to this Society for presenting a paper in which the style was so involved and the argument so obscure that anyone should fall into the error that Dr. Lehfeldt evidently has. That Dr. Lehfeldt has read my paper in a careless or slipshod manner is, of course, impossible. I did, however, try all through my paper to make clear that the supply of certain goods in relation to demand had fallen, and gave many instances to prove it. Surely it was clear to most readers that in the sentence quoted I was referring to increase of demand for *goods*—not *gold*—as Dr. Lehfeldt inferred. I deny that the construction of the sentence should lead anyone to infer I wrote thus of *gold*, and the whole argument of the paper is against this assumption. So much for the less important aspects of this question. It may appear impertinence on the part of a mere dabbler in political economy, as I am, to differ from such an authority as Dr. Lehfeldt: but if the members of this Society will try and remember that I am merely voicing the views of a growing school of economists, they will perhaps pardon my presumption.

Since writing my paper, Prof. J. A. Hobson's book "Gold, Prices and Wages," has come into my hands. One of the best known economists, Mr. Chiozza Money has dealt with the subject in the *Contemporary Review*, and the "goldite" position has been stated in the *Fortnightly Review* by Walter A. Ford. I would recommend Prof. Hobson's book to anyone interested. I have also collected information from other sources. The recent rise of prices is a fact admitted everywhere. Whether it amounts to an average of 25% or 30% over the world since 1906 might be questioned. The authorities before me show a rise of 7% to 15% between 1900 and 1911; the rise being larger in protected countries (wheat in Prussia 30s. a quarter, in England 23s.). Dr. Lehfeldt places his reliance on averages of prices of all commodities; but as I have already tried to point out, many commodities show falls in price instead of rises, which negatives the view that there is one main overmastering cause. Quite a number of the most alarming rates of increase are in raw materials and in metals, whose price can be shown to have nothing to do with gold. Taking the Board of Trade Index, for example, which is founded on 45 articles of commerce, and shows price variations as follows:—

PRICE LEVELS.

Year.	English, Sauerbeck's.	London *Economist.*	Board of Trade.	U.S.A.	Germany.	Canada.
1900	100	100	100	100	100	100
1908	97	108	103	111	107	112
1911	107	117		117		118

1871	Index No.	136
1880		130
1890		104
1896		88
1900		100
1905		98
1910		109
1912		115

One would expect to find that of the 45 a vast majority would show a fairly general rate of increase. As a matter of fact 30 show rises and 15 falls in price. Among those showing increase of price are:—Zinc 17·4%, cotton 38·3%, jute 35·3%, tin 44·7%, wheat 17·7%, bacon 42·3%, sugar 19·5%, palm oil 133·7%, rubber 48·6% and hides 35·2%. Others show smaller increases, such as:—British and foreign wool and flax, British foreign wheat, barley, oats, maize, hops, potatoes, beef, milk, eggs, tea, wine, tobacco, cotton seed, linseed and olive oil and hides. Decreases in price are shown in the case of paraffin 40·7%, silk 18·7%, lead 20·19%, coal 31·5, pig iron 23·8%, copper 22·8%; others are rice, sherries, coffee, rum, cocoa, bricks and hewn fir. Such an analysis raises a number of awkward questions. Wheat rises in price, yet rice falls; paraffin is down 40% (it has since risen): palm oil is up 133·79%; and all these amazing variations are due to one cause, excess of gold! I have already shown that only the most obstinate controversialist could succeed in tracing any connection between the prices of metals and the course of gold production. Metals figure largely in all indices and render them unreliable. Any trade knows just why palm oil has gone up and coffee has fallen in price, yet the armchair economist would bring forward the gold theory to account for it. The real rise has been in foodstuffs and raw materials. "Rifleman" maintains in his book "The Struggle for Bread," that this is due to relatively decreased production of these products as against increased production of manufactured goods.

It must be remembered that the rate of increase of price

$$=\frac{\text{amount of gold}+\text{amount of credit money}\times\text{its velocity}}{\text{amount of goods bought and sold}\times\text{their velocity}}$$

where velocity is used to mean the number of times exchanges take places. It is quite easy to convict Dr. Lehfeldt of using metaphors that might make us think that he sometimes forgets this, or when he speaks of gold being a shortening rule used to measure a room at different periods, where the whole metaphor becomes inapplicable by the assumption that the room representing the volume of trade does not meanwhile alter its size. I brought forward much evidence to show that everything pointed to the volume of trade having expanded at a greater ratio than the gold supply. Neither Dr. Lehfeldt, Mr. Richardson, nor Mr. Normand, attempt to deal with this evidence or to challenge it in any way. Dr. Lehfeldt takes refuge in an alleged "consensus of opinion" (which does not exist) that the rise is due to gold production. Mr. Normand, as shown, falls into a simple arithmetical error, and Mr. Richardson implicitly admits my contention.

I have recently come across some further evidence regarding the amazing growth of the United States. Last year the value of crops was

9,000 million dollars. The mines produced 2,000 million dollars in 1911 as against only 418 million in 1885, or a growth of about 450% ; the exports in 1913 were 2,427 millions as against 727 millions in 1885, or a growth of 350% ; and the imports 1,774 millions in 1913 as against 562 in 1885, or a 350% increase ; railway earnings also increased 400%. The greatness of India as a factor in gold absorption may be realised, when it is remembered that it has under crop 250 million acres, whereas Great Britain, Canada, Australia and New Zealand have between them only 50 million acres India produces 426 million bushels of wheat every year and her sea-borne traffic has increased 50% in 10 years. The German income is estimated at £2,000,000,000 in 1913 as against £1,250,000,000 in 1895. One recent writer points out that the rise in certain foodstuffs is due largely to emigration of millions from European and Eastern countries of a poor and low dietary to a liberal dietary like Canada and U.S.A. Even, however, disregarding the greater relative growth of trade to that of gold output, we may again ask what evidence there is of a surplus of gold.

The "goldite" argument runs somewhat on these lines: New production of gold finds its way principally to the banks Credit is based on gold. When bankers have gold they manufacture new credit in a certain proportion, and desire to lend this money ; to do so they lower (the price) the bank rate, which attracts customers who spend the money on new goods and manufactures, this stimulates trade and commerce and causes a rise in price. As Prof. Ashley writes in "Gold and Prices," "the most direct and immediate way in which an influx of gold affects trade is by causing the banks to make advances on easier terms, so stimulating enterprise and causing an increase in the demand for commodities and services, and consequently a rise in prices." Unfortunately the facts are dead against this naïve theory. The discount rate has been going up, as the following table shows :—

Period.			Rate Per Cent.
1890—1894	...	...	3·1
1894—1899	...	...	2·8
1900—1904	...	...	3·6
1905—1909	...	...	3·66
1910—1911	...	...	3·59

Present day: England 4½ to 5, France 4, Europe 5½ to 6. Over a long period, vault bullion reserves and gold output show a similar disagreement with this theory. There is thus evidently something wrong with the theory. The other school of thought, while not seeking to deny a relation between gold reserves and credit, would offer another explanation. As Hobson says, "large gold output rendered it possible to pass large sums to bank reserves where they were required, by the demand for credit which depended upon the profitable uses to which it might be put. These uses imply high prices. This hypothesis reverses the order of causation, but it explains how a large output of gold, large bank reserves, large credit, high rate of discount (owing to completion of loans and scarcity of capital in relation to demand) and high prices exist together." If it had not been for the large gold output it might have been difficult to feed the money markets of the world with the increased quantity of bank money required to conduct the volume of trade on its most profitable basis. "The evidence as to disposal of the new gold suggests that the demand for a large volume of credit draws gold into the banks of the countries which require it to sustain credit." Has the area of profitable investment expanded faster than the supply of new capital? Yes ; otherwise it is impossible to explain the rise in price (interest) of new capital. The demand for capital has been in excess of the supply. No other explanation accounts for the fact of the rise of rate of interest of securities, etc. This period has been marked by an immensely rapid expansion of the area of profitable investments, due to the opening of new countries. The large influx of gold does not then play the directly determinative part claimed for it by the "goldites." It has not been a chief efficient cause of increase of purchasing power and higher prices, but it has been a favouring condition to this process. Had there been a constriction of the gold supply and low reserves the price of money would have been higher, borrowing less, demand reduced, and prices lower. "On the theory that the supply of credit rises and falls with gold, the actual rise of the price of credit shows that the supply of gold has been insufficient and inadequate to keep down the price of money." On every hand we hear that the amount of money in the world is not sufficient for the demands made upon it. The chairman of the Standard Bank recently told us so, and there is no need to enlarge upon it. As Mr. Cullen pointed out, vast sums have been locked up in development schemes, and vast sums have been lost.

The other contention of my paper was that, apart from special cases examined and other causes mentioned, any further general rise of prices was due not to growth of gold reserves, but to a progressive growth of credit money, and I gave figures to show this. Dr. Lehfeldt's position with regard to this is somewhat obscure. He admits that this has occurred, and he also states, "prices would not have risen so much if it were only a matter of production of gold"; yet in his final sentence "he is convinced that the rise of prices is chiefly due to excessive gold production." The question whether credit money is based entirely on gold reserves is answered in the affirmative by most "goldites." Dr. Lehfeldt, however, is somewhat cautious on this point; but he does not appear to realise what his admissions imply: "Bankers have to keep a stock of gold proportional *more or less* to amount of credit"; "ratio of deposits to cash in U.S.A. rose to something over 25% relative to former periods"; "ratio of gold is fairly constant." In another passage, however, he admits, "taking the world at large a progressive economy in gold will be found." These admissions practically give the "goldites" case away, as they disprove the contention that the existence of the great stock of credit money in the world to-day is dependent entirely on the increased supply of gold. Let us examine Professor Fisher's position; it will be found equally wobbly. He writes: "It has even been argued that this interposition of circulating credit breaks whatever connection there may be between prices and the quantity of money (gold). This would be true if circulating credit were independent of money (gold). But the fact is that the quantity of 'credit money' tends to hold a definite relation to money (gold). That is, deposits are normally a more or less definite multiple of money." Just "definite," then, under normal circumstances, "more or less definite." Again he admits: "If we continue to follow up the ultimate basis of the banks' liabilities we shall find it in the visible tangible wealth of the country." In other words, as I showed in my paper, the great growth of credit money tends under certain modern conditions to be based more or less on gold, and if there has been a rise in price due to excess of money it proves that prices depend on credit, not on gold. "More and more in modern civilised countries financial crises are met by a suspension of the right to demand gold." This means that goods, not gold, are the basis of the modern money system. The vast addition of late years of shares and scrip of various sorts, to the world's stocks has formed a large basis for credit. As I pointed out in my paper, during the recent Balkan crisis the banks of Austria would have been depleted of their gold had not the government practically forbidden it.

We now enquire why this scramble for gold continues. The explanation Dr. Lehfeldt brings forward does not account for the facts. If gold were being produced in excess, it might, under his theory, be accepted as a kind of necessary evil by banks. But we have the strange phenomenon of a fierce competition for it, which even raises its market price. There are several reasons for this condition of things. This is a fact that I cannot understand economists like Dr. Lehfeldt ignoring. As Lord Milner recently stated, there is a "scramble for capital" everywhere, and new capital can be put to most profitable use. For instance, in Canada 10% can be got with safe mortgages. With one pound of gold the banks can manufacture anything from £4 to £10 credit money, so naturally gold is in demand when money is dear. Political considerations also influence this demand. When Europe is peaceful credit is good and gold goes a long way in making credit; when storm clouds arise it is another matter. Gold is used in international finance in much the same way that Dreadnoughts are used in international politics. Take an instance. Germany threatened to go to war with France over Morocco. Immediately, French banks called in all their loans and gold from Germany, and caused a bad crisis; and the Government of Germany was warned by the commercial and financial magnates of that country to let the matter drop, and France scored a diplomatic victory. The German Government controls the Reichsbank, which by law keeps a 25% gold reserve, and word went round that the gold reserve must be increased from 40 million pounds to 60 or 70 millions. The result was that during the last 38 weeks Germany bought at a premium 20 million pounds worth of gold—or absorbed the greater part of the Rand's output for a year. New gold goes, however, and has gone mainly to new countries, where the demand for capital is greatest. Credit being good in England, she uses 1,000 millions credit money with only about 100 million pounds gold reserve. Again, she has increased the velocity of her money as the following table shows:—

Year.	Gold output.	Net Gold imports U K.	London bank clearances.
1895	41 millions.	14 millions.	7593 millions.
1900	52 ,,	7·8 ,,	8960 ,,
1905	78 ,,	7·8 ,,	12228 ,,
1910	96 ,,	6·4 ,,	14659 ,,
1912	98 ,,	6·2 ,,	15962 ,,

In conclusion, the facts I have brought forward can be explained only in one way. The immense expansion of the world has widened immensely the area for profitable investment, and while producing a vast amount of new goods for trade has caused a shortage of some commodities, especially food stuffs. The demand for credit money has been so large that banks have used more and more of other forms of wealth, especially shares, debentures, etc., as a basis for credit money, and have at the same time eagerly sought for all the gold they could obtain. This immense amount of new capital may have had a certain minor effect in raising the general level of prices; but the fact of a high bank rate tends to show that there has, owing to wastage of capital in wars and preparation for wars (during the last ten years Europe has sunk £1,500,000,000 in navies alone), in luxuries and in development schemes that have not yet matured, been a scarcity of money rather than an excess with regard to the trade of the world. It is hence evident that we must look to the causes mentioned in my paper rather than to the mere fact of gold output to explain the recent rise of prices of so many articles. This rise has not been universal, hence no one cause is exclusively operative. The main cause cannot be the increased gold output, since it has barely kept pace with the output of metals, manufactured goods, and coal. It has relatively been smaller than the growth of trade, railways, increased luxury of living, and the increase in growth of territory opened to trade, and of the growth of population using gold in the arts and as coinage. Unrest in the world to day is largely due to the fact that while rising prices have greatly lowered the real wage of the worker, the scarcity of capital and the demand for it have allowed it to claim an increasing share of the profits of industry.

THE EFFECT OF CHARCOAL IN GOLD-BEARING CYANIDE SOLUTIONS WITH REFERENCE TO THE PRECIPITATION OF GOLD.

By Morris Green (Associate).

Freshly-ignited charcoal is a recognised precipitant of gold in cyanide solutions. Graphite, the author found, does not precipitate gold.[*] The action of graphite has been the subject of much discussion during the last few years. The advocates of the 'precipitation theory have mentioned the effect of charcoal as an argument in their attempt to prove that graphite precipitates gold, but why charcoal has that effect they do not say. On the whole, remarkably little experimental evidence has been adduced to either prove or disprove any contention. Briefly summarised, it is *thought* to-day that the action of charcoal may be referred to one or more of three causes,[†] namely: (1) a chemical reaction in which the carbon participates; (2) a reaction due to osmotic pressure; (3) a reaction with occluded gases.

So as to render comparable experiments in which charcoal was in contact with gold-bearing cyanide solutions for various periods of time, it was necessary, at the outset, to determine whether gold precipitated through the agency of charcoal was easily dissolved out of the latter, or otherwise, by potassium cyanide solution. (Whether dissolution and precipitation proceed simultaneously does not, in this connection, matter, the final result only being of importance.) Experiment showed that a true relative measure of the efficiency as precipitating agents of different forms of charcoal is obtained by determining the amounts of gold precipitated at the end of the period of contact, experiments being performed in which this varied up to about 30 hours.

Finely-crushed wood charcoal that had been stored (in lump form) in bags for about a year was added, in equal weights, to several gold-bearing cyanide solutions (cyanide strength = 0·2% KCN), the latter being equal in volume and withdrawn from one stock solution. The mixtures were simultaneously mechanically agitated under the same conditions. After 3, 6, 9, 12, 15 and 18 hr., successive agitations were removed, filtered and washed. The gold contents of the various solutions were practically identical, as also were the amounts of gold carried by the charcoal. It follows, therefore, either that the solution (free cyanide being present) is incapable of dissolving the precipitated gold during the time of the experiment, or that the rates of dissolution and precipitation have become equal. The charcoal will not precipitate indefinite quantities of gold.[*]

Next, a gold-bearing cyanide solution was agitated with charcoal as before. The charcoal (carrying about 5 mgm. precipitated gold) was filtered off, washed and added to 250 cc. 0·25% KCN solution. The mixture was then agitated for 20 hr., the charcoal filtered off, etc., as before, and agitated for an additional 20 hr. with 250 cc. 0·25% KCN Aq. taken, as in the previous case, from one stock solution. Four per cent. of the gold originally precipitated by the charcoal was re-dissolved by the first fresh solution and 1·6% by the second.

Examined under the microscope by reflected light, the wood charcoal was seen to be made up of cells which were long as compared with their breadth. The walls of the cells consisted of minute laths of carbon which were arranged along the grain of the wood and were parallel at most points. Occasionally they radiated from or towards a centre. A circular cell structure was also observed, similar to the common vegetable cell structure. The cells were never individual and distinct but communicated one with another. The carbon showed, as a rule, shallow punctures and pronounced porosity. The surface of the carbon was covered with a thin, bright layer, probably deposited during manufacture of the charcoal. The lump and powdered charcoal showed similar structures. Charcoal made from different forms of wood and at different temperatures exhibited the same general arrangement, varying chiefly in the length of cells and porosity.

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[*] *Jl. of Chem., Met. and Min. Soc. of S.A.*, Sept. 1912.

[†] R. Stuart Browne, Am. Met. Soc., *Eng. and Min. World*, Dec. 2, 1911, p. 1108.

[*] The numerical 'limit of enrichment' quoted by Julian and Smart in their 'Cyaniding Gold and Silver Ores,' p. 155 (Ed. 1904), was not verified with the charcoal employed in the present investigation (under the prevailing conditions), this charcoal being able to precipitate a greater proportion of gold than is stated in the reference.

The gold was precipitated through the agency of the charcoal in the metallic state, the colour being the typical yellow of metallic gold.

The effect of different forms of charcoal was determined. Freshly ignited wood charcoal was a more efficient precipitating agent than the same charcoal exposed to the atmosphere for several months. On re burning old charcoal its power of precipitating gold from cyanide solution was restored. Animal charcoal was a moderately efficient precipitant. Cane sugar was treated with sulphuric acid. The liberated carbon, which exhibited, under the microscope, a pronounced cell structure, had no precipitating effect. The product obtained by partially burning sugar in vessels open to the air precipitated poorly, while charcoal made by carbonizing sugar in closed retorts was a fairly efficient precipitating agent. Using equal volumes of one stock gold-bearing cyanide solution, the table shows the percentage of gold precipitated, under the same conditions, by equal weights of the different forms of charcoal.

TABLE I.

Precipitating Agent.	% Gold Precipitated.
Freshly-ignited Wood Charcoal ...	90·3
Wood Charcoal exposed in lumps for about 6 months (ii) ...	50·0
The same Charcoal (ii) re heated to incipient redness	84·2
Sugar Charcoal (by heating sugar in closed retorts)	62·1
Sugar Charcoal (by partially burning sugar in contact with the air	5·6
Sugar Charcoal (by attacking sugar with H_2SO_4 and washing out acid	0·0

It has been shown above that if charcoal be added to a gold-bearing cyanide solution a condition is set up after a time whereby the amounts of precipitated and dissolved gold remain constant (within certain limits of time). To discover the effect of the presence of charcoal in a cyanide solution used to dissolve gold, gold cornets, equal in weight and similar in all respects, were placed in different solutions which were of the same cyanide strength, volume, etc. The charcoal was then added to the solutions and the mixtures were mechanically agitated. A certain amount of gold was found to have entered into solution, and the remainder was associated with the charcoal filtered off. Here, too, a condition of equilibrium was found to exist, since continued agitation for a number of hours did not result in a perceptible alteration of the amounts of gold dissolved and precipitated. Further, when the volumes and KCN strength of the solutions, total amounts of gold present and weights of any particular form of charcoal were respectively equal in two agitations, then the amount precipitated from a gold-bearing cyanide solution, containing at the beginning no undissolved gold, was the same as the amount remaining undissolved in (or precipitated from) a cyanide solution, which at the beginning carried no dissolved gold but was in contact with a gold cornet; *e.g.*, to each of mixtures A (350 cc. 0·25% KCN Aq. carrying 5·7 mgm. dissolved gold) and B (350 cc. 0·25% KCN Aq. in contact with a gold cornet weighing 5·7 mgm.) was added 0·22 grm. old wood charcoal and the mixtures were simultaneously mechanically agitated under the same conditions. After 25 hours, 38·6% (A) and 39% (B) of the total gold originally present in each case was found in the undissolved state. Several hours' further agitation did not influence the result.

The preceding experiments were repeated with several forms of wood charcoal, for each of which a different constant was obtained; *e.g.*, with freshly-ignited wood charcoal and with the quantities, etc , used above, 90% was ultimately found undissolved.

The general effect of charcoal in cyaniding has now been dealt with; a discussion of the underlying principles follows.

The supposition that carbon *per se* participates in the reaction is so obviously incorrect that it requires no further comment. Nevertheless, in confirmation of this assertion, a quotation, *in extenso*, of a portion of the author's former work* on graphite will not be out of place here.

"Gold-bearing cyanide solutions were . . . agitated with the 'graphite.' Percolation of such solutions over graphite was also tried. In every case the whole of the gold was retained in solution. . . . A sample of mineral graphite was next used in performing a few similar experiments. This graphite, found in Natal and containing 37% ash, exhibits the characteristics usually associated with true graphite.

"Cyanide solutions, with varying concentrations and total amounts of gold in solution, were agitated with and made to percolate over Natal graphite. The whole of the gold remained in

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*A Research upon a Refractory Gold Ore. *Jl. of Chem., Met and Min. Soc. of S.A.*, Sept., 1912, pp. 84—101.

solution, whether contact between solution and graphite was brief or extended over several days. The state of division of the graphite, which was employed both in the lump form and in the finest powder, was without influence.

"To discover the effect of the presence of graphite in a cyanide solution being used to dissolve gold, two gold cornets, of the same thickness and form of surface, and each weighing 0·0150 grm. were placed into different beakers, each containing 175 cc. 0·23 % KCN solution. To beaker No. I. was added 0·1 grm. Mt. Morgan graphite, and the solutions mechanically agitated, under exactly similar conditions, for 40 hours. The solution containing the graphite was then found to have dissolved 96% of the gold added to beaker No. I., the remaining 4% being present as several small particles of undissolved cornet: the graphite itself was free from gold. Solution No. II., or that containing no graphite, had, in the same time, dissolved 36·7% of the gold at its disposal. These results appeared remarkable, for not only was the graphite innocent, in effect, of any attempt to precipitate gold from, or hinder the latter from entering into, solution, but it actually aided the cyanide to dissolve gold. Similar experiments, in which the variables, such as amount of gold and graphite, strength and volume of cyanide solution used, and time of treatment were altered, all yielded results which conclusively proved the graphite to accelerate solution of the gold. Natal graphite was now substituted for Mt. Morgan graphite and a similar series of experiments performed. . . . The solution to which graphite was added had . . . dissolved 100% of the gold present, the gold remaining in solution and not contaminating the graphite. The other cyanide solution, free from graphite, dissolved 55% of the gold in the same interval of time. . . . Further tests . . . confirmed the first impression; the graphite accelerates solution of the gold and does not cause what has been termed 'premature precipitation.'

. . . "The observed effects of 'graphite,' regarded as 'carbon,' are in perfect accordance with the laws of electro-chemical action. Carbon being strongly electro-negative to gold in cyanide solution does not precipitate gold; also, by the formation of an electric couple where gold and carbon meet, the former enters into solution more readily than in the absence of the electrical effect. . . . "

Guesses have been occasionally made as to the supposed existence of an osmotic pressure set up when a solution is in contact with charcoal. This existence, however, does not appear to be real.

A circular excavation was made in a prismatic block (about 6 in. × 1 in. × 1 in.) of dense charcoal, which showed the cell structure, by drilling to within half-an-inch of the bottom; the diameter of the hole was about half-an-inch, so that the least thickness of the walls was approximately a quarter of an inch. A gold-bearing cyanide solution (free cyanide = 0·2% KCN) was placed in the opening, which was closed by a tightly-fitting stopper through which passed one limb of a manometer, the latter thus being capable of indicating a change of pressure. To ensure all connections (*e.g.*, the contact between stopper and charcoal) being perfectly air-tight, molten paraffin wax was applied where necessary. The block of charcoal was immersed in a beaker of water to such a depth that the level of the water outside was coincident with that of the solution within. Throughout a day the manometer showed no change of pressure; also the free cyanide was completely mixed with the outside water, a solution of the same cyanide strength being found inside and outside the charcoal. After much longer periods, too, no change of pressure could be discerned.

A semi-permeable membrane of copper ferrocyanide was precipitated within the walls of an unglazed porcelain vessel. This vessel (which readily allowed the passage of pure water), containing a certain volume of potassium cyanide solution (strength 0·27% KCN), was placed in a glass cylinder of water. After 24 hours less than 0·4% of the KCN contained in the inner vessel had penetrated the membrane. To this membrane, therefore, potassium cyanide behaves as a colloid. Besides the absence of a pressure, potassium cyanide readily passes through charcoal, so that the latter does not exhibit the properties either of an animal or a semi-permeable membrane. [Since the whole of the charcoal does not behave as a membrane there appears to be no feasible reason to suppose individual cells (of the charcoal) to do so.] It can be concluded that charcoal does not stop the passage of a dissolved substance through it by means of an osmotic effect.*

At this stage the action of other porous substances than charcoal was also considered. Pumice stone was without influence in a gold-bearing cyanide solution. Several specimens of kieselguhr were available. With a high magnification a cell structure was apparent under the

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*In a paper in the *Jl. of the Chem. Soc.*, Vol. XCII., part 2, p. 233, the authors conclude 'that a semi-permeable membrane consists of a network of small bubbles of vapour communicating, on the one hand, with the pure solvent, and on the other, with the solutions, by means of capillary tubes. The passage of solvent from one side to the other then proceeds by evaporation of the liquid and the subsequent condensation of the vapour. The only substances which, when dissolved in a liquid, do not traverse freely a semi-permeable membrane separating the solution from the pure solvent are those which do not pass from the dissolved to the vaporous state.'

microscope. The kieselguhr did not behave as a precipitating agent, no matter whether a gold-bearing cyanide solution was slowly percolated over it or whether it was agitated with such a solution. Anthracitic coal and coke had no precipitating effect. These (coal and coke) were employed in quantities varying from a small amount up to 8 lb. per ton of solution, while the gold content of the solutions was also varied from a trace up to several oz. per ton of solution. The microscope showed the coke to possess a very pronounced cell structure. Lubricating oils had no precipitating effect, although it is obvious that they could retard dissolution of gold.

It has been remarked that there is a deposit within the wood charcoal. This carbon is probably more like graphite than ordinary carbon, but its physical condition and nature are unknown. Whether there is a difference of potential at the innumerable points of contact (between this deposit and the rest of the charcoal) can be a subject of uncertain conjecture only. Strips of wood charcoal and mineral graphite (about 1 in. × $\frac{1}{4}$ in. × $\frac{1}{4}$ in.) were partially immersed, 2 in. apart, in a 0·2% KCN gold-bearing solution and the circuit closed externally. An e.m.f. of 10^{-4} volt was observed. When the charcoal was replaced by a thin strip of gold, the voltage recorded was 5×10^{-4} volt. Between charcoal and gold the e.m.f. was 4×10^{-4} volt. After maintaining the circuit for many hours the graphite electrode was assayed and showed no gold. The actual figures here given are of no direct value in the solution of the problem for a variety of reasons. Nevertheless, it is possible that to this cause (namely, an e.m.f. between 'graphite' and carbon) can be attributed a small fraction of the power of the charcoal to precipitate gold from cyanide solution. The main factor to account for the fact that charcoal is a precipitating agent will be explained later. When it is remembered that the e.m.f. due to precipitated gold is very small at the commencement of precipitation, and that the e.m.f. to redissolve the gold is not great enough until the latter becomes sufficiently massive or of extensive surface—it is obvious that a small e.m.f. suffices to begin precipitation of the gold. The deposition of carbon in charcoal is probably due to the presence, during manufacture, of a mixture of reducing gases; and it is proposed now to proceed to a consideration of occluded gases.

Freshly-ignited wood charcoal, contained (in pea size) in a glass flask, was connected with a gas burette, mercury being the confining liquid. By lowering one limb a difference of about 20 in. was available for exhausting the gas. The gas (61·5 cc.* from 5 grm. charcoal) collected throughout a day showed the following composition:—

TABLE II.

Gaseous Constituent.	Volumes per cent.
Carbon dioxide	0·0
Heavy hydrocarbons ...	0·0
Oxygen	17·5
Carbon monoxide ...	20·0
Methane	0·0
Hydrogen	37·0
Nitrogen	25·5

After the removal of the above gases the charcoal was found to be nearly as efficient a precipitating agent as before the removal of the gases.

The freshly-ignited charcoal after being exposed to the atmosphere for several days and then exhausted with the gas burette, yielded a larger volume (82 cc. from 5 grm. charcoal) of gas than in the former case under the same conditions. The composition of the mixture is given in Table III.

TABLE III.

Gaseous Constituent.	Volumes per cent.
Carbon dioxide	0·2
Heavy hydrocarbons ...	0·0
Oxygen	22·3
Carbon monoxide ...	1·1
Methane	0·6
Hydrogen	55·0
Nitrogen	20·8

The charcoal left was still an active precipitating agent. The mixture of loosely held gases which are given off so readily is, therefore, not the effective part of the charcoal so far as its power of precipitating gold from cyanide solution goes.

The charcoal left after withdrawing the gases given in Table III. was slowly warmed. At any one temperature a definite volume of gas was collected, irrespective of whether the temperature was gradually rising or falling; *e.g.*, in warming the charcoal a volume of 65 cc. (at 19·3 deg. C. and 622 mm. mercury) was collected when the charcoal was heated to 70 deg. C. As the temperature rose, the volume of gas (at 19·3 deg. C., etc.) given off increased. On cooling the charcoal, contained in a vessel communicating with the evolved gas, the gas was slowly absorbed.

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*At 19·3° C. and 622 mm. mercury.

again, until at 70 deg. C. the volume of gas in the burette was, as before, 65 cc. (at 19·3 deg. C., etc.). This behaviour, typical of all the lump charcoal employed, was not shown by the same charcoal in the form of fine powder, which did not readily reabsorb gases once expelled. From this difference in behaviour it would appear that lump charcoal could retain its occluded gases to a greater extent than the powder, on exposure to the atmosphere. The gaseous mixture collected at 70 deg. C., from the lump charcoal, showed the following analysis :—

TABLE IV.

Gaseous Constituent.	Volumes per cent.
Carbon dioxide	2·3
Heavy hydrocarbons ...	0·0
Oxygen	13·2
Carbon monoxide ...	0·6
Methane	0·8
Hydrogen	9·0
Nitrogen	74·1

The charcoal was still an efficient precipitating agent. Exhausting the finely ground charcoal at the same temperature, with the gas burette, did not impair its power of precipitating gold.

The wood charcoal that had been exposed for about six months (*see* Table I.) yielded only ordinary air when subjected to the vacuum of the burette, at the air temperature. At the temperature of boiling water (94 5 deg. C.) a large volume (about 30 cc. from a grm. of charcoal) of gas was evolved consisting of ordinary air with a small portion of the oxygen replaced by CO, CO_2 and H_2.

Several forms of commercial wood charcoal that had been stored for several months were found, as in the previous case, to yield, besides air, only very small amounts of the other gases (an average of 0·16 cc. per grm. of charcoal) when subjected to the influence of the gas burette ; and the removal of these gases, as would be expected, was without effect upon the charcoal regarded as a precipitant of gold.

Freshly ignited wood charcoal, exposed to the atmosphere for 14 days, had lost the loosely held gases to such an extent that the burette removed from the charcoal (heated to 200 deg. C. by immersing the containing flask in hot oil) only 0·5 cc. gas per grm., exclusive of ordinary air. This charcoal was still as efficient a precipitant as before removing the latter gases, although not quite so effective as shortly after manufacture. To discover whether this falling off in efficiency was due to a slow evolution of gas, a quantity of the charcoal, several days after manufacture—when all the very loosely held gases were lost—was put into communication with the gas burette and left for over a week. Besides ordinary air, several cc. of a mixture of CO and H_2, chiefly CO, were collected from a grm. of charcoal which had now become a slightly poorer precipitating agent.

In the foregoing experiments only the vacuum obtainable with the gas burette was employed. In the experiments now to be described the charcoal was subjected to the powerful vacuum of a Töpler vacuum pump. A modern construction of this apparatus was available, and by suitable adjustments it was possible to collect and transfer the gases evolved to the gas burette for subsequent analysis.

The freshly-ignited charcoal, after 14 days exposure to the atmosphere, was exhausted by the pump at the ordinary temperature, the gas collected being practically air, in which 0·2cc.* (per grm. charcoal) CO replaced a corresponding volume of oxygen. Hardly any difference in precipitating power could be detected between the charcoal before and after exhaustion.

Heat, in addition to the vacuum of the Töpler pump, was next applied to the charcoal, which for this purpose was contained in a silica tube. The lower end of this tube rested in a blow-pipe furnace ; glass tubing passing through the stopper closing the silica tube made connection between the pump and charcoal. The composition of the gaseous mixture collected at the ordinary temperature has been already referred to. This gas was expelled into the atmosphere, and the gases liberated on heating collected. The evolution of gas proceeded slowly with rise of temperature until approximately 500 deg. C. was reached (using the charcoal prepared from the wood of packing cases), when there was a rush of gas. The temperature remaining nearly constant, this rapid evolution ceased after a time and gradually the liberation of gas became exceedingly slow. At this point the gas was removed for analysis and the charcoal allowed to cool out of contact with the atmosphere, after which the containing vessel was opened and the charcoal removed for experiments to be described shortly. The volume of gas collected per grm. of charcoal was about 250 cc., the analysis being given in Table V. (p. 262).

There can be little doubt that the gases determined do not exist in the charcoal in the proportions given in the table. The coefficient of dissociation (at atmospheric pressure) for CO_2 at 2500° C. abs. is 0·19 and at 2000° it is 0·04, while the corresponding figures at $\frac{1}{10}$ atmospheric

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*All volumes are referred to 19·3° C. and 622 mm. mercury.

TABLE V.

Gaseous Constituent.	Volumes per cent.
Carbon dioxide	12·4
Heavy hydrocarbons ...	0·0
Oxygen	0·7
Carbon monoxide ...	26·2
Methane	0·0
Hydrogen..	2·5
Nitrogen	58·2

pressure are 0·33 and 0·08.* The decomposition of CO into C and CO_2 ($2CO = CO_2 + C$) is at a maximum and almost complete at 450° C but decreases to almost nothing at 1000 C.* On expulsion from the charcoal the CO would, therefore, be decomposed to a considerable extent while the CO_2 would not be so affected. Also, at the temperature employed, the action of oxygen upon carbon and of CO_2 upon carbon ($CO_2 + C = 2CO$) would not be very appreciable. It would be difficult to state the extent to which certain reactions proceeded as the gases cooled off during collection. On the whole, it is probable that the gas occluded in the charcoal contains as high a, or a higher, percentage of CO than that stated in the table and a lower percentage of CO_2 than is mentioned there.

Dense, commercial wood charcoal evolved a similar mixture of gases when heated and exhausted.

The gas obtained on heating (to the temperature employed above, namely, about 500° C.) and exhausting mineral graphite had the following composition, approximately :

Oxygen	16·0% (by volume).
Nitrogen	83·5% ,,
Carbon dioxide ...	0·5% ,,

The structure of the charcoal, viewed under the microscope, was not altered by heating and exhausting, the usual cell structure with the rather bright deposit being observed. The deposition of carbon in charcoal has been occasionally attributed to the decomposition of hydrocarbon gases. Although the absence of the latter, in almost every case, does not prove that hydrocarbon gases were not present and decomposed during manufacture of the charcoal, yet it appears very probable that the decomposition of CO, at moderately low temperatures, is responsible for much of this carbon deposit. A similar explanation *may* account for the 'graphite' found in gold and silver ores, in which cases the graphite would be the result, not the cause, of reducing conditions. This supposition receives support from certain observed effects, *e.g.*, in the Mt. Morgan ore (Barberton) the graphite carries practically no gold and does not precipitate gold from cyanide solution.*

The action of the charcoal used in the above experiments was compared with that of a sample of the same stock charcoal before exhausting the gases. The usual procedure was adopted, equal volumes being withdrawn from one gold-bearing cyanide solution, and equal weights of the different samples of charcoal, in nearly the same state of mechanical division, agitated with the solutions, under the same conditions and for the same periods of time.

TABLE VI.

Nature of Charcoal.	% Gold Precipitated from Solution.
(i.) 'Packing Case' wood charcoal exposed to atmosphere for about two weeks after manufacture	93·3
(ii.) The same charcoal (i.) heated and exhausted as described in the text	17·0
(iii.) Compact commercial wood charcoal exposed to atmosphere for about a year ...	41·2
(iv.) The same charcoal (iii.) heated and exhausted ...	14·8

On reheating the charcoal to dull redness, in air, its power of precipitating was largely restored. The temperature at which charcoal is manufactured in the first instance probably determines, to some extent, the ease or otherwise with which the gases can be expelled (by variation of pressure and temperature). It was somewhat difficult to expel the occluded gases from revivified charcoal (made by heating the exhausted charcoal to dull redness in contact with air) which then again became a poor precipitating agent.

It appears, therefore, that the power of charcoal as a precipitant of gold from aurocyanide solution is due to occluded gases. Carbon monoxide and hydrogen were the reducing gases found ; and since the hydrogen is present in quantities too small (except in freshly-burnt charcoal in which it is loosely held and, as former experiments have shown, is without influence) to account for the observed effects, carbon monoxide must be the principal active constituent. Carbon monoxide or hydrogen if passed into an ordinary gold-bearing cyanide solution would not precipitate gold, [the loosely held reducing gases have been proved to be without action upon aurocyanide solution] so that the condition of these gases in the charcoal is obviously different from that of the same gases existing by themselves in the absence of charcoal, while the condition of the latter gases is similar to that of the loosely held gases in the charcoal. The absorptive power of charcoal is greater upon carbon monoxide than upon oxygen, nitrogen, or hydrogen,* that upon hydrogen being least. As the earlier Tables show, hydrogen is rapidly evolved from the charcoal whilst the liberation of CO, after the expulsion of the first loosely held volume, proceeds slowly; and it is through the loss of CO that the charcoal† gradually deteriorates in its precipitating efficiency.

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*Damour & Queneau, 'Industrial Furnaces and Methods of Control,' pp. 34—35.

**Jl. of Chem., Met. & Min. Soc. of S.A.*, Sept., 1912, pp. 84-101. *See also* 'Gels in relation to Ore Deposition,' by Hatschek & Simon, *Trans. Inst. Min. & Met.*, xxi.

To demonstrate further that the loosely held gases have little or no influence, but that the slow loss, at ordinary temperatures, of occluded gases not held quite so loosely results in the deterioration of the charcoal as a precipitating agent, a stream of CO was passed over the sugar charcoal (made by treating sugar with H_2SO_4) for several days, the charcoal being neither heated nor exhausted previous to passing CO. Under the conditions prevailing in the experiments the results of which are embodied in Table 1, the sugar charcoal now precipitated 2·3% of the total gold originally in solution.

Why the chemical activity of gases in consequence of their absorption by charcoal is increased seems difficult to explain. It has been variously stated that the gas held by charcoal is in a highly condensed condition upon its surface, and by virtue of this condensed state the gas is more active than under ordinary conditions, so that the charcoal is enabled to induce certain reactions. An investigation of the equilibrium between the gases involved and the charcoal on the basis of the Phase Rule classification would throw much light on the condition of the gases in the charcoal.

Carbon monoxide, therefore, to a large extent (and probably hydrogen to some extent) appears to be responsible for the power of the charcoal to precipitate gold from aurocyanide solution. Possibly it is entirely responsible, a condition of equilibrium between the gas and solid not being disturbed at the temperature and pressure employed. Merely heating the charcoal to higher temperatures and evacuating is, in this connection, unsatisfactory, since complicated reactions may occur that would annul the effect of the expulsion of occluded gases and still leave the charcoal an efficient precipitating agent. Also, the electrical effect previously referred to, and due to the deposition of carbon in the cells of the charcoal, may account for the precipitation of a small amount of gold. It only remains to add that, owing to their porosity, wood ashes and charcoal in tailing dumps absorb a relatively large amount of solution, and by the subsequent evaporation of this solution dissolved gold will be deposited irrespective of the presence or absence of occluded reducing gases.

In conclusion the author wishes to record his indebtedness to Prof. G. H. Stanley and Mr. M. T. Murray for their interest and advice in the prosecution of the investigation.

NOTES ON THE ASSAY OF MINE SAMPLES.

(Read at June Meeting, 1913.)

By Robert Dures (Member).

REPLY TO DISCUSSION.

Mr. Robt. Dures *(Member)*: Mr. John Watson has been at some trouble to point out the sudden changes in value that may take place in sampling underground. The sampler of experience, who knows his mine, is in a position to locate the poor zones, especially the large areas of low grade ore in the footwall of some of our mines, and as the time will soon be upon us when it will be necessary to take account of fractions of 1 dwt. in mine assaying, I still maintain that it is advisable to part mine samples that come near the borderland of "payable rock." Then, too, there are large tonnages of so-called waste rock on the surface. The assays of the samples from these dumps I contend should all be in "fine gold," considering that now 1 dwt. from waste at grass may be made to pay its way. The samples to be parted could be marked by a letter, say P, in blue pencil, and assayed in separate batches.

Mr. Watson says if all beads (which I do not stand out for) had to be parted it would mean employing more assayers or doing fewer samples. Well! certainly, get more help if the work cannot be "speeded up," and it can be in some assay offices by obtaining better equipment.

I do not think Mr. Graham Martyn's traverse of my figures calls for much reply on my part, except to thank him heartily, and my other critics, for such additional matter as they have contributed.

Mr. James Gray points out that the subject of time saving devices might have been brought forward. I am sure that an interesting and useful paper could be written at length on this matter of devices alone. A paper, too, on furnaces ought to bring out the views and experiences of many members, as there is some diversity of opinion as to what is the best type of furnace, and what is the most suitable fuel. Then, too, the question of "quartering down" is an important one, and ought to be thoroughly thrashed out, as most of the known methods are faulty, one way or another.

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* 'Dictionary of Applied Chemistry,' Thorpe.

† *Wood* charcoal is to be always inferred. Although the reaction between animal charcoal and aurocyanide *may* be attributable to the same cause, this reaction was not determined, being somewhat beyond the scope of the investigation.

One point Mr. Whitby overlooks when speaking of 30 samples being enough to see crushed, etc., is the improved machinery now in vogue as against the old hand crushing way, where with careless natives quite a percentage of rock was shot out of the mortar.

Apparently we are nearly all agreed that there is room for improvement on the old slap dash ways of handling mine samples.

Mr. Whitby, when he remarks there is very little that is new in the tables, has evidently lost sight of the avowed object of the paper, which was to draw attention to and nail down the everyday errors that crop up in the assay of samples from the mine.

THE SAND-FILLING OF MINES.

(Read at September Meeting, 1913.)

By W. A. Caldecott (Past-President) and
O. P. Powell (Member).

DISCUSSION.

Mr. H. A. White *(Member)*: It is certainly a gratifying fact, as pointed out by Mr. Pam, that the practical solution of this problem as applied on the Witwatersrand is so largely due to members of this Society. The difficulties which have been overcome were manifold, among the principal being the excessive wear of pipes and the rendering innocuous a material of distinctly poisonous character. Although the authors have not given us details of the cost of boreholes, it seems certain that they have proved the great advantage this method of conveying the sand to the stope possesses, for they entirely eliminate the expensive renewals necessary with pipes. No conceivable wear in the borehole can have any ill effect, though it would be of some academic interest if they would inform us what that wear amounts to. It would no doubt, in view of the facts quoted, be perfectly possible to utilize their method even in our deepest mines, though some modifications would be necessary where the dip of the reef is very flat, as in parts of the far East Rand. It would be a matter for consideration whether it would be preferable to utilize the vertical pressure to perform the underground distribution in pipes, involving a more fluid pulp, or whether the much greater quantity of water required to carry pulp along launders with a grade of 5% or 10%, should form an underground closed circuit. A series of experiments on the relation between grade of launder and ratio of water necessary with the fine, slime-free sand used, would form a suitable companion to that suggested by the authors on the similar problems with pipes. The figures for compression of water deposited sand under heavy loads are interesting, inasmuch as Lord Kelvin has instanced such material as having a negative coefficient of compression for small loads. This fact was impressed upon him on consideration of the well known effect produced by standing upon wet sand on the sea-shore. The pressure of the foot appears to cause the sand to dry under its weight and this is attributed to expansion of the sand, or, rather, increase of interstitial space, caused by a more unfavourable arrangement of the solid particles than the natural optimum. The details of the test quoted are not too satisfactory and it would have been interesting if the grading of the sand before and after compression had been compared.

Very little work has been done upon the question of what proportion of the vertical pressure upon settled sand is transmitted in a horizontal direction, though this is certainly some function of the angle of repose. Wilson ("Conjugate pressures in fine sand," Proc. Inst. of Civil Engineers, Vol. CXLIX, p. 208) quotes Rankine's theoretical formula

$$p = \frac{(1 - \sin \alpha)\, w\, h}{(1 + \sin \alpha,}$$

where p is horizontal pressure,
α is angle of repose,
w is weight per cubic foot,
h is height in feet,
or $w\, h$ is vertical pressure.

His experiments show a fair agreement with theory and show that ratio of horizontal pressure is least when the sand contains 12% of moisture and is then 21 of the vertical pressure.

Professor G. H. Darwin ("Horizontal thrust of a mass of sand," Proc. Inst. of C.E., Vol. 71, p. 350) points out the importance of the "history," that is, the method of stratification of the sand and shows that the most favourable case is when the strata of deposition dip away from the wall receiving the pressure. He got as low as 18% with dry sand in this manner. Boussinesq, in a note on this paper (Vol. 72, p. 262) calculates that the angle of friction may increase under pressure and suggests that the angle of friction might be proportional to the density obtained. Prof. Rankine deals with the matter in his "Civil Engineering," pp. 168 and 321 of the 1904 edition, where, however, he is chiefly concerned with the stability of foundations depending upon such material. The importance of further experiments upon this point will be more obvious when our deepest mines become vitally interested in the question.

With reference to the various surface methods adopted by the authors, it appears that the diaphragm cone arrangement has undeniable advantages : the question of trucking or pumping would usually be decided by considerations of the site of operations, because the material is in all cases at present put into trucks from the vats. Possibly a consideration of the economy of transport obtained with pumping methods may result in the system of handling sand residues now in vogue being radically altered in the direction indicated, and this would eliminate one objection raised by the authors to the suggested direct flow of residue to boreholes. But pumping certainly ensures a more thorough mixture with the cyanicide employed and the cones give the necessary delay to ensure completion of the reaction which may take as long as an hour in unfavourable cases. In this connection the cones may be credited with their safety valve in correcting fluctuations of supply and ensuring that averaging upon which calculation of the quantity of permanganate must rely.

The question of alkalinity and freedom from organic matter of the dilution water is quite as important as that of the solution associated with the sand residue. This again is steadied by the return water arrangement at the Simmer and Jack, and, no doubt, the freedom from any slight deleterious effects underground is connected with these advantages. There can be no doubt that the use of permangante as cyanicide is perfectly safe provided sufficient is present and the pulp is alkaline. It is of course well known that in acid solutions practically no reaction takes place owing probably to the fact that the HCN formed is almost completely un-ionized. As certain organic compounds are very readily attacked by weak permanganate of potash it is well that the water used for dilution should be examined in this respect.

I observe that the use of bleaching-powder has been superseded at the Simmer and Jack, chiefly, I believe, because it is more awkward to handle, and when it is considered what a small portion of the total cost is due to the chemicals employed, the use of the most convenient is doubtless justified.

The thoroughness of the methods employed in checking the destruction of the cyanide are to be highly commended, and the only point I am slightly doubtful upon is the certainty of Mr. MacArthur Johnston's method when the underground effluent is acid. Suppose a little $FeSO_4$ were present also—and this is not impossible—upon addition of alkali some of the cyanide may be converted into ferro-cyanide and lost to the test as quoted. Though this point is doubtless absolutely without importance in the cases he has to deal with, and moreover is probably provided against in the detail work, it is well to clear the matter up beyond peradventure for the methods he uses will no doubt be put into operation in other parts of the world where conditions may be very different.

In their reply the authors may possibly let us have some figures from the Robinson Deep results where I hope to see confirmation of the prediction that it would cost the reduction plant less to put the stuff under the ground than on top of the dump. I should imagine that if the change to Caldecott cone collection was justified by consideration of quality of sand sent underground, the total effect upon residues, if put to the credit of the sand-filling account would reduce the cost per ton upon that mine to a very low figure indeed.

Upon consideration of the success achieved in the destruction of cyanide by permanganate, I have suggested that experiments should be made with a mixture of solution of potassium permanganate, calcium chloride and lime water, followed a little later by a little vinegar in order to test its efficacy as an antidote in cases of poisoning.

The well known and enthusiastic Red Cross medical gentleman to whom I suggested this could only promise that if I took the cyanide he would try that remedy upon me.

A NEW METHOD OF PRECIPITATION BY ZINC.

By John S. MacArthur, M.Inst.M.M. (Hon. Member).

(Read at January Meeting, 1913).

REPLY TO DISCUSSION.

Mr. John S. MacArthur *(Hon. Member)* : It is not easy to explain how a method of precipitation which is highly successful in Portugal should not succeed in South Africa and New Zealand.

On reviewing the whole discussion, I am inclined to think that the precipitation at Caveira is facilitated by the presence of mercury and lead in solution. Mercury sometimes equals the amount of silver in quantity, and the lead though sometimes present in large quantities is generally only a mere fraction of the silver present.

The bullion precipitate tested by me contained about 35% of mercury. I do not classify mercury as foreign matter, as it is a valuable metal, and, is in fact recovered by distillation. I did not go into this detail as the matter was named by Mr. Hutton, and doubtless the mercury present reconciles Mr. Hutton's statement that the precipitate would run 35% fine metal, and my statement that the precipitate contained only about 15% foreign metal. It now seems to me that the presence of lead and mercury are a much more important factor in this matter of precipitation than I had anticipated.

The meeting then terminated.

THE ANNUAL DINNER OF THE SOCIETY.

The Annual Dinner of the Chemical, Metallurgical and Mining Society of South Africa was held at the Carlton Hotel on Saturday night, the 22nd November, and for the first time in the Society's history ladies were present.

Mr. Alex. Richardson, M.Inst.M.M., presided, and the guests included the Mayor of Johannesburg (Mr. Norman Anstey), Mr. J. Q. Braidwood (Association of Certificated Mechanical Engineers), Mrs. Braidwood, Mr. A. J. Brett, Mrs. Brett, Dr. W. A. Caldecott (Past President), Mr. W. Calder (President of the S.A. Institution of Engineers), Mrs. Calder, Mr. F. D. P. Chaplin, M.L.A., Mr. W. Cullen (Past President), Mr. J. Christie (President of the Pharmaceutical Society), Mrs. Christie, Colonel W. Dalrymple, Mrs. Dalrymple, Mr. A. von Dessauer (Geological Society of S.A.), Mr. W. R. Dowling (Past President), Mrs. Dowling, Mr. W. Elsdon Dew, Mrs. Elsdon Dew, Dr. Helen Driver, Mr. B. Gray, Mrs. Gray, Mr. H. J. Hofmeyr, Mrs. Hofmeyr, Mr. R. T. A. Innes, Mrs. Innes, Mr. E. H. Johnson (Past President), Mr. A McA. Johnston (Past President), Mr. E. L. R. Kelsey, Mr. R. N. Kotze (Government Mining Engineer), Mr. J. Littlejohn (Hon. Treasurer), Dr. H. A. Loeser (British Medical Association), Mr. T. A. Malpage, Mrs. Malpage, Dr. J. McCrae, Dr. J. Moir (Past President), Mr. J. Munro (President of the Chamber of Mines), Mr. H. Neuhaus, Mrs. Neuhaus, Mr. A. E. Payne (President of the Association of Mine Managers), Mr. S. H. Pearce (Past President), Mrs. Pearce, Mr. S. R. Potter, Mr. B. Price (South African Institute of Electrical Engineers), Mr. Fred Rowland (Secretary of the Society), Mr. C. B. Saner (Past President), Professor G. H. Stanley (Vice-President), the Right Rev. the Bishop of St. John's, Mr. H. Taylor, Mrs. Taylor, Mr. J. E. Thomas (Vice-President), Mrs. Thomas, Mr. J. Wevell (President of the Rand Pioneers), Mr. H. A. White (Vice-President), Mrs. White, Professor J. A. Wilkinson (Chairman of the South African Red Cross Society), Mr. J. A. Woodburn, Mrs. Woodburn.

Apologies for inability to attend the dinner were received from His Excellency the Governor-General, General Botha, the Hon. F. S. Malan, Sir Lionel Phillips, M.L.A., Sir George Farrar, Mr. W. Toomey, Mr. and Mrs. G. Imroth, Messrs. G. Nathan, Senator S. Marks, Mr. and Mrs. W. L. Honnold, Mr. and Mrs. Warrington Smyth, Mr. and Mrs. A. Aiken, Mrs. Chaplin, Mrs. A. E. Payne, Mrs. Bernard Price, Dr. H. B. Currie and Dr. Stanwell.

The Society's Activities.

The Mayor, in proposing "The Society and the Mining Industry," recalled that in 1890 there was a severe depression in Johannesburg. Many low-grade propositions were in a very sad state, but the introduction of the cyanide process made all the difference, and the discovery of the uses of cyanide in those days carried the fate of the whole Witwatersrand area. Those low-grade propositions were to-day employing thousands of white miners and many more thousands of natives, and upon them depended Johannesburg as a commercial and an industrial centre. He realised that the Society, in addition to the practical assistance it had rendered to the mining industry, had also interested itself in other directions by contributing prizes in aid of research work. It had also assisted the Seymour Library, and inaugurated an exhibition which came to the assistance of inventors who were not always able to bring their inventions to fruition without the aid of the Society. Those aids which the Society had rendered to some of the smaller men were, he understood, invaluable. The members of the Society were also, he learned, engaged in assisting ambulance work along the Reef, and their work was much appreciated by the miners themselves. They had greatly aided the mining industry by the results of their chemical researches. There was one other industry to which he thought they could render very great assistance. He referred to the agricultural industry, which needed the aid of the chemist almost as much as the mining industry, in order that the farmer might be assisted in determining the value of the various grades of his soil. They might assist the farming industry in research work, as it was assisted to-day in regard to cattle diseases. He believed that with assistance of that sort South Africa would yet be one of the great granaries of the world. It might also be one of the great cattle-raising centres of the world. There was no reason, so far as he could see, why this country should not develop as an agricultural centre, and he hoped he should live to see the time when South Africa would compete with our fellow-colonists in Australia and New Zealand, and supply the manufacturing centres of the world with our foodstuffs and our cattle. (Applause).

Johannesburg with its 125,000 white inhabitants, was practically dependent to-day upon the mining industry—(hear, hear)—and anything they did for the benefit of that industry they did for the benefit of Johannesburg in particular and of the country generally. They as a community realised the assistance rendered by scientific men in the development of that great industry upon which they all indirectly and directly lived. Touching on the relationship between the Municipality and the mining industry, he said that the Municipality had always recognised the importance of the dependence the city had on the prime industry of the country. His predecessors in office had very carefully managed financial affairs from the outset.—(Hear, hear.) About nine years ago the Municipality raised a loan of 5½ millions, which had since been increased to 5¾ millions, on very reasonable terms. The loan was floated at 96¼, and at a rate of 4%. This loan, according to the exigencies of the situation in Johannesburg, dependent as they were upon the mining industry, the life of which was considered in financial circles in London a comparatively short one, was therefore made redeemable within 30 years; consequently at present they had practically paid back 1¼ millions of that loan. They had put to reserve and to depreciation of their various assets a further sum of half a million. They had invested that redemption in buying back their stock; and in addition to the improved position that existed to-day, they had available trading concerns that were run in the interests of the people of the city. In those trading concerns they had 2¼ millions of their loan invested. Those were bringing them a very handsome return on the investments. If put on the market to-day as ordinary trading concerns, they would be fully sufficient to pay off the remaining loan on the Municipality; therefore, he considered that the financial position of Johannesburg to-day was essentially a sound one from a public point of view.—(Applause.) He recognised the importance of the Society, to which he wished every possible success.—(Applause.)

The President: Mr. Mayor, ladies and gentlemen,—let me preface my remarks by seizing the opportunity of extending most hearty congratulations to Mr. Norman Anstey on his election as Mayor, and of wishing him a most successful year of office.—(Applause.)

It gives me genuine pleasure to respond to the toast of the Society and the Industry, because the work of the Society has always excited my keenest interest, and having been closely connected with the Industry for twenty-three years, I have, for more reasons than one, a very friendly feeling for it. The Society started nineteen years ago with forty-two members, all resident on the Rand; and it has grown steadily ever since, until it now has over one thousand members, representing all the civilised parts of the globe and some of the uncivilised ones.—(Hear, hear). This widespread co-operation in the search for technical perfection assists our members in maintaining a wide breadth of view on technical subjects; and it is also of service to the whole community, inasmuch as there can never be too many minds helping to round off and establish a scientific truth or to pulverise a fallacy. We hold that there are few devices so efficacious in preserving the sanity of science as healthy discussion, and we are at all times anxious to encourage the sincere expression of views which will keep sanity predominant. I think that some people are unnecessarily reluctant about placing their professional experience upon record, and it seems to me that this unwillingness is a mistake; for when a man puts a fence round his knowledge he bars out a great deal more than he closes in, and though it may sound paradoxical it is none the less true that a man's knowledge only becomes his real asset when he has shared it with others.—(Hear, hear.) Of course there are those who object to anything in the nature of vigorous discussion; but even if the worst should happen and someone's sensitive toes should be trodden on in the struggle for enlightenment, he may unreservedly solace himself with the reflection, that it would have been very much worse had he been allowed to go on sleeping.

Now, with regard to the Industry—which is the sole reason for the existence of Johannesburg and our presence in it—I am afraid that there is little new that I can tell you; but it may be interesting to recall the feeling of apprehension, international in its scope, which arose when during our late industrial unrest the flow of gold from here to Europe was threatened with a serious diminution. This apprehension is not difficult to understand. Gold is the wind of commerce and the tide of trade. All over the civilised world the wholesale transactions of commerce are conducted by means of credit instruments, that is, without the intervention of metallic money. In England, 99% of it is conducted in this way. Now this enormous superstructure of credit is based on gold; how enormous it is you will be able to infer from the statement of an economist, that if all the banks were at the same moment compelled to meet their obligations in actual coin they would just be able to pay out 4d. in the pound!—(Laughter.) It follows from this that a small shrinkage in the gold basis causes a proportionately large shrinkage in the vast edifice of credit which rests upon it, and by so doing reduces credit facilities, discourages enterprise, and hampers trade. As this goldfield furnishes 40% of the new supplies of the metal, it readily becomes apparent that our output exercises an influence far beyond that usually supposed; in fact it is not too much to say that our mines stand for the economic comfort of civilisation.—(Applause.) Apart from this aspect, the importance of the Industry in the stimulating influence which it has had, and will have, on the development of South Africa can hardly be overrated; and it, therefore, should be the desire of any one who wishes to see this country advancing, to assist the growth of the Industry in any way he can.

I confess that at this stage of our mining history it is difficult to bring forward any reasonable suggestions which if applied might prove beneficial, but I should like briefly to touch upon two points. The first has reference to the conventional flat contract system in use in many of our mines. That this method for the payment of miners has never given unbounded satisfaction is common knowledge, and it is probably equally well known the perfect system has still to be discovered. However, after studying many methods and witnessing a few of them in operation, I find myself most in sympathy with the formula: Whites on day's pay and natives on contract; a system the merits of which have been amply demonstrated in the diamond mines of Kimberley.—(Hear, hear.)

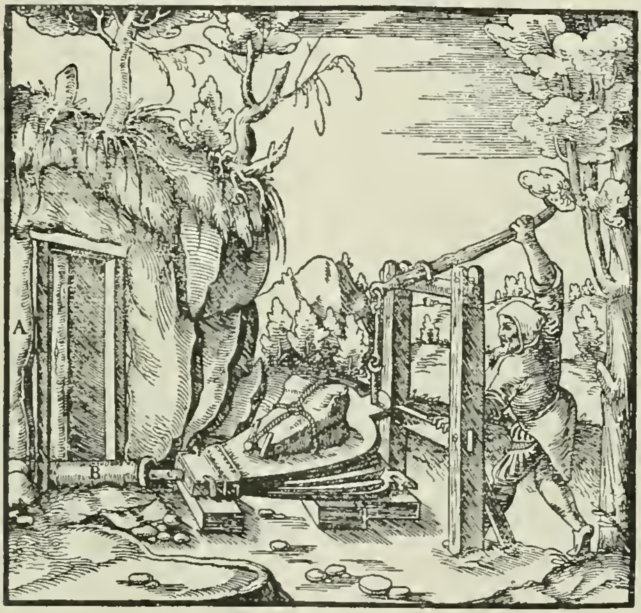
Regarding my other point, I should much like to indulge in a lengthy explanation,—(No, no.)—but as no one, I think, after dinner, wants anything elaborate, I shall try to put it in a nutshell. Owing to our geographical position we are to a great extent an isolated community, at any rate we are not in close direct touch with any other large mining field from which instructive lessons could be drawn, and our own conditions vary little from mine to mine. This concentration has a tendency to occasion a certain inbreeding of outlook, and makes it difficult for us to see our difficulties as others see them; not that others have the slightest compunction in painting fancy pictures of them for our benefit. But to come to the kernel of the matter, it has often occurred to me that it would be a good thing if now and then one or other of our knowledgeable engineers were induced to adopt the rôle of post-graduate—shall I say super-post-graduate?—and deputed to investigate the practice of other fields, in order that the first-hand information so gathered might be utilised in helping the Industry to solve any of the difficult problems that confront it. Such an investigation need not cost much, and it might be productive of excellent results.—(Hear, hear.)

This concludes all I have to say to you this evening, except to thank the Mayor for the kind way in which he gave the toast, and you, ladies and gentlemen, for the very cordial way in which you received it.—(Applause).

Prof. G. H. Stanley *(Vice-President)*: Mr. Chairman, Ladies' and Gentlemen, I am honoured to-night in having been allotted the pleasant duty of proposing the health of "Our Guests and Kindred Societies."

I am sure we are all glad that so many distinguished people have been able to accept the invitation to attend what I feel is a sort of annual "thanksgiving day" of this Society for the progress of the past year, and only regret that of those invited all could not attend. Quite a number of them, however, occupy high places in the political world, and judging from what we read in the newspapers, had very pressing business elsewhere! To-night, also, for the first time, we are honoured by the presence of ladies: in this connection, gentlemen, words fail me and I am forced to leave the duty of welcoming them in the capable hands of Prof. Wilkinson. Many of our guests represent kindred societies, others, public bodies and controlling financial houses and others again are of outstanding importance in the official and social spheres, or members of the "fourth estate," we try our best to obtain a representative selection—our friends all—to rejoice with us on these occasions. We heartily welcome them all and through them extend our greetings to the Societies they represent.

In these days human knowledge covers such a wide field that the number of societies and institutions dealing with different branches, and even with cognate branches, seems almost appalling, and to the formation of new societies there seems to be no end. With so many societies, each specialising in some particular branch, there may be some danger, amongst the members of each, of overlooking the work of other societies and losing grasp of the proper correlation of effort. Too great specialisation is not a good thing; it is necessary to keep as wide an outlook as possible, and we try to remember that our work touches that of other societies at many points, as theirs does ours, and some little overlapping of effort is unavoidable. Still, when we recognise that we are all in pursuit of knowledge and can undoubtedly help one another in our several fields, the "brotherhood of science" which even knows no nationality, is brought home to us very forcibly and the gathering of representatives of our local Societies here to-night is symbolical of the co-operation and friendship with which each pursues the common aim. Looking down the list of guests, I note the name of Mr. Kotzè, Government Mining Engineer, representative of the majesty of that Department which "shapes our ends" in the mining sense. and I fear that some of our members sometimes have to think of him and of what *he* may think in certain eventualities in his official capacity: to-night I hope the cares of his office are forgotten: and yet the illustration on our menu cards, representing an early method of mine ventilation, is very likely to recall them. Indeed it has an interest for most of our guests: Mr. Payne of the Mine Manager's Association may also feel hurt at our forcing upon his notice at such a time one of the current problems of his profession, and I am afraid it may similarly affect the Engineers, Mr. Calder, Mr. Price, and Mr. Braidwood, and cause a few cold shivers in the quarters from which the wherewithal has to be provided. Still, I think we are all agreed that mine ventilation has to be made efficient, and although as a community we may congratulate ourselves on the progress which has been made—I do not mean since the time of the illustration—but within the last few years here, I think we also admit there is more to be done.



A METHOD OF MINE VENTILATION IN THE OLDEN DAYS.
(*a*) Tunnel; (*b*) Pipe; (*c*) Nozzle of Double Bellows.
From "*De Re Metallica*," by Agricolae.
(First printed in Latin in A.D. 1556.)

You will also notice on your cards the motto "Capaci Occasio." Recently I overheard some discussion as to what it really meant, and one gave as a translation "of capacity equal to the occasion." Well, that may not be very correct, but we like to think that the meaning, at any rate, is something like that; and in connection with this problem of ventilation or any other similar problem that may arise, we hope we shall be, with the co-operation of our good friends the kindred technical societies, equal to the occasion. Similarly, it seems to me an extremely appropriate motto to find a place on a dinner invitation: I am sure we hope your capacity has been equal to the occasion. Personally, being a bit of a heretic, I always feel that the order of the proceedings on such an occasion as this is wrong and should be reversed; that is, the speeches should come first and the dinner afterwards, and this to my mind has some obvious advantages. It always seems to me that an ordinary healthy animal after having had a substantial meal doesn't want to do anything very particularly but go to sleep, and I have heard a good many humans express the same feeling with great conviction. Observe what a task the present custom throws upon the caterer: if he feeds the company too well they are unfitted for the subsequent proceedings, if not well enough they are again unfitted, for a different reason, they will not feel sufficiently cheerful—he has to keep a very nice balance. If it is contended that a surplus should be provided and the onus put upon the diners, it necessitates a careful watch and nice restraint upon the appetite for fear that when the capacity is equal to the occasion, paradoxical as may seem, incapacity may be the result! Then again, think of the effect upon the length of the speeches—perfectly obvious! and as I have no desire to mar the enjoyment of our guests by speaking at length myself, I now give you the toast of "Our Guests and Kindred Societies," coupled with the name of Mr. Payne.

Mr. A. E. Payne, replying to the toast, said that there had been some doubt at first as to the actual cause of phthisis, as to whether it was of gaseous or of other origin; but whatever its origin, they had all made up their minds about one thing, and that was that the disease had to stop.—(Applause.)

Prof. J. A Wilkinson proposed the toast of "The Ladies," and Mr. E. L. R. Kelsey replied.

Mr. F. Drummond Chaplin, M.L.A., in proposing the health of of the Chairman, congratulated the Society on its activities, and the success of the function just concluded.

The Chairman: Mr. Drummond Chaplin, ladies and gentlemen, it is most kind of you to take the toast of my health. All I can say is, that I hope you will look back upon this evening with as pleasant recollections as I shall have of my occupancy of the Chair.

An admirable musical programme was rendered by Mrs. Stanley Anderson, Messrs. J. W. Birrell and Mr. L. R. Glenton.

The proceedings concluded with God Save the King.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Sensitiveness to Light of Azides.—"According to this article, silver, mercurous, lead and cuprous azides are decomposed by the action of light. Experiments show that not sub-azides but mixtures of unaltered azide and finely divided metal in a colloidal state remain. This is interesting in view of proposals to use azides in place of mercury fulminate in detonators."—Wöhler and Krupko, from *Berichte*, *46*, 1913, pp. 2,045-2,057; *Journal Society of Chemical Industry*, July 15, 1913, p. 698. (T. D.)

Electrolysis of Copper Solutions —"In the electrolytic deposition of copper from nitric acid solutions it is practically impossible to deposit the whole of the metal upon either a rotating or stationary cathode under ordinary conditions: and in considering the results of experiments in which a copper-coated cathode was caused to rotate in a nitric acid solution with no current passing, the probable cause of the failure of the current to deposit the last traces of copper from a nitric acid solution suggested itself.

The object of the preliminary experiments was to determine the difference in the rate of reaction of dilute nitric acid with copper at rest and in motion. For this purpose a platinum cathode was thickly coated with copper by electrolytic deposition. After drying and weighing it was rotated in 50 cc. of nitric acid (sp. gr. 1·108) containing about 10 gm. of HNO_3 and less than 0·1 mgm of HNO_2. When the rotation was stopped the cathode was at once removed, washed, dried, and reweighed. It was then immersed in a similar solution, but was allowed to remain at rest for the same length of time. The beaker was immersed in a vessel containing water at room temperature. All the conditions were kept as constant as possible, except that the metal was rotating in the one case and at rest in the other. The following remarkable results were obtained:—

i. Rotated for one hour, weight of copper dissolved	6·5 mgm.
Rest do. do. do.	780 6 ,,
ii. Rotated for two hours, weight of copper dissolved...	15·4 ,,
Rest do. do. do.	3,486·3 ,,

It is well-known, but often overlooked, that nitrous acid plays a very important part in the solution of copper in nitric acid, and that the lower acid, or nitrite, accumulates in the solution. In solutions containing only traces of nitrous acid the action is very slow at first, but gradually increases as more nitrite is formed. If this is allowed to remain on the surface of the metal the general reaction is rapidly accelerated, but if it is hurried away into the mass of the solution the reaction is retarded. This is the most obvious explanation of the results given above.

Now in the ordinary electrolysis of nitric acid solutions of copper the solution is boiled in the first instance to decompose nitrous acid and nitrite, and to expel the products of their decomposition; but nitric acid is again reduced to nitrous acid by the current during electrolysis, and all the conditions for the dissolution of copper from the cathode are present. At first the current deposits the metal very much faster than it is redissolved, but finally, when only traces of metal are present in the solution, dynamic equilibrium is reached, and no further permanent deposition of the metal can take place. This furnishes another argument in favour of the rotating cathode over the stationary one when purely nitric acid solutions are used.

The following experiments were then made as a further test: 5 gm. of electrolytic copper were dissolved in 50 cc. of 1·2 nitric acid, and the solution evaporated to one-fifth its bulk to decompose nitrous compounds. It was then made up to 500 cc., and the free nitric acid in 50 cc. determined. This was found to be 236 mgm. The remaining solution was electrolysed with a rotating cathode in 50 cc. lots. Additions of free acid were made as shown in the table, and the other conditions were kept as constant as possible. The measured volume with additions was made up to 100 cc. in each case.

The potential difference between the terminal of the rotor carrying the cathode and the anode = 3·2 volts. Current = 1·5 amps. Duration of experiment = 40 minutes.

TABLE OF RESULTS.

	1	2	3	4
Experiment.	Weight of Copper Deposited.*	Free Nitric Acid present.	Nitrous Acid Found.	Free Sulphuric Acid present.
1	474·3	236	33·0	—
2	497·4	236	28·8	—
3	497·3	236	16·3	4,500
4	455·0	1,236	33·5	—
5	495·5	2,236	30·9	—
6	495·7	3,236	31·3	—
7	424·8	4,586	38·1	—
8	314·7	6,326	60·3	—
9	494·0	6,326	14·2	4,500

It will be seen from the table that the more or less perfect deposition of the copper depends upon the concentration of nitric acid and of nitrous acid in the solution, the smaller deposit corresponding to the greater concentration of the higher acid, while for the same concentration of the 'ic' acid the smaller deposit corresponds to the greater concentration of the 'ous' acid. From experiments 5 and 6 it would appear that up to 3 gm. of HNO_3 per 100 cc. of solution the deposits are good providing the nitrite keeps low; but beyond that the deposit is poor.

Before the introduction of the rotating cathode the method adopted by the author was to evaporate the nitric acid solution with sulphuric acid before electrolysis, as it was found impossible to deposit the whole of the metal without getting rid of the nitric acid, on account of the low current density used. But with the rotating cathode it is sufficient to keep the nitric acid low and add from 2 to 3 cc. of strong sulphuric acid. The usual plan is to dissolve about 0·5 gm. of the metal in 10 cc. of 1·2 nitric acid, boil down to decompose nitrites, and make up to 100 cc. with addition of 2 cc. of concentrated sulphuric acid. After 40 minutes' electrolysis the exhausted solution will very rarely give any colouration with sulphuretted hydrogen. To prevent any re-solution of the metal when the circuit is broken the following simple plan has been found very effective: a wash-bottle with a long rubber tube attached to the mouthpiece is arranged so that when the rotation is stopped a jet of water can be directed upon the cathode, while the current is gradually broken by lowering the beaker. The fixed water-bottle with the flexible tube allows the operator to use both hands in manipulating the apparatus.

The action of the sulphuric acid is probably twofold: (*a*) its ions help to carry the current and thereby to check the formation of nitrous acid; (*b*) it unites with nitrous acid to form a nitro compound which is much less effective in bringing about re-solution of the metal. Comparison of the results of experiments 2, 3, and 8, 9 makes the first part of this explanation clear. The second part was suggested by the late Dr. Edward Divers in discussing the results of some experiments on the reactions of metals and alloys with nitric acid, which the author has carried out."—J. H. STANSBIE, *Transactions of the Faraday Society*, July, 1913, p. 11. (H. A. W.)

SAMPLING COAL DELIVERIES. — "Anthracite larger than pea size is now bought by the United States Government on a guaranteed maximum percentage of ash on the dry coal. If this is exceeded, the Government has the option of refusing delivery or deducting 10% from the bill. Anthracite peas and smaller sizes are bought on a guaranteed maximum percentage of ash and minimum heat value, both calculated on the dry coal. Coal which is not up to specification may be refused, or 25% deducted from the price at the purchaser's option. Bituminous coal is bought on a guaranteed maximum percentage of ash, volatile matter and sulphur, and minimum heat value, all calculated on the dry coal, and on a guaranteed maximum percentage of moisture in the coal as received. The determination of moisture is only made when the sampler suspects an undue percentage, and in such cases he is required to collect a special moisture sample with the utmost speed and without the refinements adopted to make the ordinary sample uniform, as such refinements would, by loss of moisture in the process, introduce errors, besides which the slightly more representative character of the sample would be negligible. These moisture samples are of about 30 to 50 lb., and are rapidly crushed to $\frac{1}{2}$ in. and quartered down to 3 lb. The ordinary sample is collected with the utmost care to make it representative and without regard to moisture losses in the process, as the price to be paid depends directly on the heat value of the dry coal, and is subject to increase or decrease according to an agreed scale for every 1% that the ash is less or more than that named in the contract. It is laid down in the contracts that the gross sample shall not be less than 600 lb. in the case of slack and small anthracite, nor less than 1,000 lb. with larger coal, and it is shown in this bulletin that there is rarely any advantage in collecting a gross sample larger than 1,500 lb. Detailed directions are given for mixing and quartering such quantities. When prepared by hand, the pieces of coal and impurities are crushed to the approximate size indicated in the table below before each reduction:—

Weight of Sample to be Divided.	Size to which Coal and Impurities should be Broken before each Division.
1,000 lb. or more ...	1 in.
500 lb.	$\frac{3}{4}$ in.
250 lb.	$\frac{1}{2}$ in.
125 lb.	$\frac{3}{8}$ in.
60 lb.	$\frac{1}{4}$ in.

The $\frac{1}{4}$ in. mesh sample is then reduced to about 3 lb. by quartering or, preferably, by the use of a mixing and reducing machine."—S. POPE, from *U.S. Bureau of Mines Bulletin*, 63, p. 68: *Analyst*, July 1913, p. 330. (T. D.)

A NEW CHEMICAL CAUSE OF RUSTING OF IRON. —"NH_4NO_3 is said to attack iron energetically producing ferrous and ferric hydroxides. Part of the iron dissolves as a complex ion which cannot be detected by the ordinary reagents. Alkalinity hinders the reaction."—VAUBEL, from *Chemiker Zeitung*, 37, 1913, p. 693-4; *Journal Society of Chemical Industry*, July 15, 1913, p. 701. (T. D.)

RAPID MODIFIED CHLORPLATINATE METHOD FOR THE ESTIMATION OF POTASSIUM.—"This method is applicable in the presence of chlorides, sulphates, phosphates, nitrates, carbonates, borates and silicates, salts of sodium, barium, calcium, strontium, magnesium, iron and aluminium, and is especially suited for the estimation of potassium in potassium salts and mixed fertilizers and in organic fertilizers after an appropriate modification to remove ammonia and other organic bases.

The method follows:—Prepare the solution in the usual way, acidify slightly with hydrochloric acid, add chlorplatinic acid solution slightly in excess of that necessary for the complete precipitation of the potassium present and evaporate the solution on the steam bath to a syrupy consistency, *i.e.*, until solidification occurs on cooling. Flood the cooled residue with a small quantity of alcohol of at least 80% strength, grind thoroughly with a pestle made by enlarging the end of a glass rod, and allow to stand for one-half hour. The alcoholic solution should be coloured if an excess of chlorplatinic acid has been used. Pour the liquid through a small filter, using suction, and before adding more alcohol rub up the residue again with the pestle. Now continue the washing by decantation with small portions of alcohol until the wash liquid becomes colourless. Three or four washings usually suffice. Transfer the precipitate to the filter and wash two or three times with alcohol.

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*Weights are expressed in milligrams.

Dissolve the precipitate in hot water, washing it through the filter into a beaker of convenient size. To the hot solution add about 1 cc. of concentrated HCl and approximately 0·5 gm. magnesium ribbon (which has been previously washed in water) for every 0·2 gm. potassium present, stirring the solution and holding the magnesium at the bottom of the beaker by means of a glass rod. When the magnesium has practically dissolved add a few cubic centimeters of dilute hydrochloric acid and allow the flocculent platinum to settle. The supernatant liquid should be perfectly clear and limpid like water if reduction is complete. To make sure add more magnesium, in which case the solution will darken if reduction be incomplete. Now add concentrated hydrochloric acid and boil to dissolve any basic salts, filter, wash thoroughly with hot water, ignite and weigh. From the weight of platinum thus obtained calculate the percentage of potassium."—W. B. Hicks, *Journal Industrial and Engineering Chemistry*, Aug. 1913, p. 650. (T. D.)

Anhydrous Alumina as a Desiccating Substance.—"Anhydrous alumina, obtained by heating hydrated alumina at a low temperature, is stated to be a better desiccating material for gases than the substances hitherto employed (phosphoric anhydride, sulphuric acid, calcium chloride, zinc chloride and bromide). When a certain quantity of phosphoric anhydride and aluminium oxide is exposed to the action of a current of air saturated with steam, the phosphoric anhydride only absorbs a scarcely perceptible amount of water, whereas the aluminium oxide increases 18% in weight.

A tube containing aluminium oxide can be used indefinitely to desiccate a gaseous current. It suffices to regenerate it, from time to time, by running a current of air, at 300° to 400° C., through it."—*Chemical Trade Journal* (from *Metaux et Alliages*), July 26, 1913, p. 87. (T. D.)

Combination Specific Gravity Bottle and Dilatometer.—"This apparatus has been used for determining the changes in specific gravity and volume which sugar solutions exhibit under various conditions. It can be used for various laboratory purposes.

The apparatus consists of a narrow tubular body holding about 30 cc., connected at the bottom with a graduated capillary tube and contracted at the top to a neck. The latter is slightly funnel shaped and is ground to receive a thermometer. The displacement of the thermometer is about 7 cc. leaving a capacity of about 23 cc. for the instrument after stoppering. The end of the capillary tube is ground and a small cap fitted. The scale on the capillary upon which the changes of volume are measured, is graduated so that 1 division is equal to 0·001 cc.: by means of a magnifying glass, readings can be made to 0·0001 cc.

A few weighings, after filling the instrument with air-free distilled water to different points of the scale at different temperatures, are sufficient for constructing a table of water constants for each scale division and temperature.

A full-size sketch of the apparatus is given in the original article."—*Journal American Chemical Society*, 8, Aug. 1913, p. 955. (T. D.)

Extraction of Nitrogen from Air.—"Air at a high pressure is pumped into water. Owing to the solubility of nitrogen being greater than that of oxygen the proportion of nitrogen in the dissolved gas is greater than in the original air. By reducing the pressure the dissolved gases are liberated and by repetition of the process the nitrogen can be obtained in greater degree of freedom from oxygen."—E. M. Wiley, *American Patent*, 1056244, March 18, 1913; *Chemiker Zeitung*, Aug. 23, 1913, p. 464. (T. D.)

Process for the Absorption of Nitrous Gases.—"In this process fatty acid esters of alcohols such as amyl acetate are used as absorbents.

Mixtures of air and nitrogen oxides such as are obtained in the oxidation of the nitrogen of the air, are absorbed in the organic liquid and the nitrous oxides liberated therefrom in concentrated form.

Amylacetate at ordinary temperature absorbs about 20% of its own weight at NO_2 and N_2O_3. Since by passing in warm air or other gases, the nitrous gases are readily driven off, it is suited for the concentration of nitrous gases.

The recovery of the oxides of nitrogen can be carried out in this way or in other ways, as for instance by the direct action of water. In the latter case they can be obtained in part directly as commercial nitric acid."—*German Patent*, 27935, Class 12i, July 17, 1913; *Zeit. Angewandte Chemie*, Aug. 12, 1913, p. 481. (T. D.)

Constant Temperature Apparatus.—"The authors describe the apparatus and a sketch is given of a form specially suitable for heat test work. The apparatus, which is based on the fact that the temperature of steam given off by water boiling under constant pressure is constant, consists of a boiler (about one-third full of water) a reflux condenser, a water-jacketed pressure bulb, and a screw valve. The internal pressure is regulated by the valve and the temperature may be maintained indefinitely at any temperature between about 25° C. and 145° C."—F. H. and P. V. Dupré, *Analyst*, July 1913, p. 308. (T. D.)

Potassium Permanganate in the Quantitative Estimation of Some Organic Compounds.—"Potassium permanganate has been most generally used in the volumetric estimation of iron. Some uncertainties formerly existed since it was impossible to obtain a chemically pure article, and insufficient data were at hand as to proper methods of preparation and standardisation of its solutions.

At present these objections have been largely overcome and almost all of our text-books on quantitative analysis contain an extended treatise on proper means of preparation and standardisation of volumetric permanganate solutions.

One of the most commonly known organic compounds that is quantitatively determined by the use of volumetric permanganate is oxalic acid. Now oxalic acid and iron are determined in acid solution, but the procedure most applicable for the oxidation of all types of aromatic compounds as well as carbohydrates and hydrocarbons is with alkaline permanganate. Oxidations in acid solution are less energetic than those with alkaline $KMnO_4$, and in the latter case the final product of a completed decomposition of the organic compound is oxalic acid instead of CO_2 and H_2O.

Among the substances mentioned in the literature (see Note) as being oxidised to oxalic acid are propylene, isobutylene, amylene, acetone, fatty acids; butyric, lactic, succinic, and tartaric acids, dextrose, sucrose, glycerol, and phenol. Now when an organic compound is oxidised to oxalic acid, a further oxidation to CO_2 and H_2O readily follows upon acidifying and warming the solution. Such a procedure forms the nucleus of a method for the determination of the compounds previously enumerated. Tocher made use of this method, and found that phenol could be determined. His method was substantially as follows:—Dissolve 1 gm. phenol in 1,000 cc. distilled water, and take 10 cc. for titration. Add 3 to 4 gm. $NaHCO_3$ together with a little distilled water. Then add 50 cc. $KMnO_4$, and boil for 5 min. Set aside to cool, and gradually add dilute H_2SO_4 to decide excess: warm to 60° C., and titrate the excess of $N/10$ $KMnO_4$ with $N/10$ oxalic acid.

(Note.—'Oxidation of Organic Compounds with Alkaline Permanganate,' Eduard Donath and Hugo Ditz, *Journ. Prakt. Chem.*, 1899, [2], lx., 566; through *Journ. Chem. Soc.*, 1900, [1], lxxviii., 197; 'Contribution to the Knowledge and Determination of the Carbohydrates,' J. König, W. Greifenhagen, and A. Scholl, *Zeit. Nahr. Genussm.*, xxii., 705; through Abstr. *Journ. Am. Chem. Soc.*, 1912, vi., 901; 'Volumetric Determination of Phenol,' Jas. F. Tocher, *Pharm. Journ.*, lxvi., 360.)

This method was found to be open to the following objection:—That the manganese dioxide formed as a result of the action of $N/10$ $KMnO_4$ upon the phenol did not reduce readily enough with consequent solution upon direct titration with $N/10$ oxalic acid. Thus the solution was full of oxide which not only obscured but rendered the end-point of little value, in that the oxide was not completely reduced before the permanganate end-point was obtained.

The following modification of Tocher's method was found to give good results:—

Dissolve 0·4 gm. phenol in 1,000 cc. distilled water. Place 50 cc. $N/10$ $KMnO_4$ and 3 to 4 gm. $NaHCO_3$ in a 500 cc. glass-stoppered Erlenmeyer flask. Add 25 cc. of the phenol solution with gentle rotation. Boil 5 to 10 min. (with stopper removed). Cool flask to about 60° C. Acidify with dilute H_2SO_4, let stand about 2 min.; cool to room temperature. Dilute with distilled water, add 5 cc. 20% KI, and titrate the liberated iodine with $N/10$ thiosulphate solution, using starch as indicator. The number of cc. of $N/10$ thiosulphate substracted from the number of cc. $KMnO_4$ originally added = No. cc. of $KMnO_4$ consumed by the phenol.

1 cc. $N/10$ $KMnO_4$ = 0·000336 gm. phenol.

If a glass-stoppered Erlenmeyer flask is not available an ordinary Erlenmeyer may be used, and its contents transferred to a glass-stoppered bottle before acidifying. Any oxide adhering to the Erlenmeyer is easily removed by the addition of a little distilled water acidified with H_2SO_4, and containing a few drops of 20% KI.

In considering the nature of the oxidation with $KMnO_4$ in acid and alkaline solutions it is observed that each molecule of $KMnO_4$ in acid solution liberates 2·5 atoms of oxygen according to the following equation:—

$$2KMnO_4 = K_2O + 2MnO + 5O,$$

or—

$$2KMnO_4 + 3H_2SO_4 = K_2SO_4 + 2MnSO_4 + 3H_2O + 5O.$$

Now in alkaline solutions the two molecules of MnO are immediately oxidised to $2MnO_2$ at the expense of 2 atoms of oxygen, so that we actually have $2KMnO_4 = K_2O + 2MnO_2 + 3O$. Hence, for each molecule of $KMnO_4$ used, only $1\frac{1}{2}$ atoms of oxygen are available for our oxidation process. This fact must be recognised in providing sufficient $KMnO_4$ to readily complete the oxidation process, and it would necessarily enter into a calculation of the value of $N/10$ $KMnO_4$ in terms of phenol if the MnO_2, or rather its hydrated form, were filtered from the solution before acidifying and adding the KI. But since the procedure is not lengthened by a filtration the MnO_2 is reduced to its manganous form with the liberation of free iodine, and we must calculate our factor by considering that reaction proceeds as in acid solution with 2·5 atoms of oxygen available per molecule of $KMnO_4$, although such is not literally the truth.

To completely oxidise phenol, 14 atoms of oxygen are required, according to the following equation:—$C_6H_5OH + 14O = 6CO_2 + 3H_2O$. Since only 2·5 atoms of oxygen are available per molecule of $KMnO_4$ then 5·6 molecule of $KMnO_4$ would be required for every molecule of phenol, and the factor for $N/10$ $KMnO_4$ in terms of phenol becomes:—

$$\frac{\text{Mol. wt. phenol}}{2{\cdot}5 \times 2 \times 10 \times 5{\cdot}6 \times 1000}$$

Experiments with a phenol solution containing 0·0005 gm. of phenol per cc., as determined by the Koppeschaar bromine method, resulted as follows:—

Ex. No.	Gm. phenol taken.	Per cent. phenol found.
1.	0·0050	99·65
2.	0·0050	100·18
3.	0·0075	99·65
4.	0·0076	99·86
5.	0·0100	99·86
6.	0·0100	99 72

Now, when the cresols were run in the same manner as phenol it was found that they were not completely oxidised and that they varied slightly as to the rate with which oxidation proceeded; hence, any permanganate method for their accurate determination must depend upon definitely fixed conditions.

Likewise, it was obvious that commercial creosote and guaiacol could not be determined by this procedure, since they are mixtures of several more or less related phenols that are not present in like proportion in different specimens. However, with single solutions of several common phenols and closely related compounds, fairly gratifying results were obtained. Pyrogallol, pyrocatechol, resorcinol, and hydroquinone, from all of which the CH_3 group is absent, were readily and completely oxidised.

Benzoic acid was very slightly attacked, while under similar conditions salicyclic acid and salol were completely oxidised. Thus it would seem that the phenolic OH group predisposes towards a complete oxidation, and that many uninvestigated phenols and closely related compounds would give analogous reactions. In making up solutions of the several phenols, sufficient $N/2$ NaOH was added when necessary, to ensure ready solution. The following table is self-explanatory:—

Substance.	Percentage found.	
Pyrogallol... ...	100·4	100·1
Pyrocatechol ..	100·2	99·9
Resorcinol... ...	100·5	100·4
Hydroquinone ...	99·49	99·56
Salicylic acid ...	99·79	100·2
Salol	99·77	99·91

The alkaline permanganate method is especially applicable for the quantitative estimation of the above compounds when they occur individually in very small amounts in single solutions, or in conjunction with substances not readily oxidised."—C. M. PENCE, *Chemical News* (from *Journal of Industrial and Eng. Chemistry*), Aug. 15, 1913, p. 74. (C. T.)

ANALYSIS OF FERRO-TITANIUM.—"The following method for the determination of silicon, titanium, aluminium, iron, and manganese will be found very simple and accurate, especially so with regard to the complete separation of titanium and aluminium.

The method of estimating the aluminium by the long fusion with sodium carbonate, &c., is a long and tedious process, and the results obtained are not at all reliable.

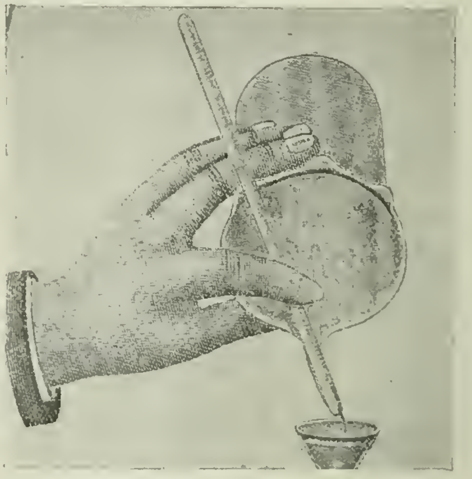
Method. —Weigh out 0·25 gm. of the finely powdered alloy, and fuse with about 10 gm. of potassium bisulphate in a deep platinum basin. Fuse until no grit can be felt with a platinum rod. Cool, and put basin and contents into a porcelain evaporating basin. Extract with water, and add 10 cc. of concentrated H_2SO_4. Allow to stand on edge of hot plate for an hour. Filter into 600 cc. beaker. Wash with hot water, and ignite residue in platinum crucible. Fuse contents of crucible with a little potassium bisulphate, and extract as before with water and a few drops of H_2SO_4. Filter, and add filtrate to main filtrate. Ignite residue in a tared platinum crucible, cool and weigh. The increase in weight is SiO_2, with a little Fe_2O_3 and TiO_2. HF the residue, and weigh; the loss is SiO_2—calculate to silicon. Dissolve the residue in a little HCl; transfer to small beaker. Add NH_4OH, boil and filter; ignite in platinum crucible and weigh as Fe_2O_3. Calculate to Fe. Any difference in the weight of residue after volatilisation of SiO_2 will be TiO_2; calculate to Ti.

Titanium.—Boil up combined filtrates from the SiO_2, and add ammonium hydrate until slight precipitate appears; clear with a few drops of hydrochloric acid, and add about 15 gm. sodium thiosulphate dissolved in water. Boil for fifteen minutes, and filter through pulp filter. Wash with acetic water. Transfer precipitate to tared platinum crucible, ignite strongly, and weigh as TiO_2. Calculate to Ti.

Iron and Aluminium. —Boil down the filtrate to fairly low bulk, and add about 200 cc. of hot water; boil down again. Continue the addition of water and the boiling until clear. Oxidise with bromine water, filter off any TiO_2 that may have got through the filter, and ignite same. Boil up the filtrate and add ammonium hydrate; filter into conical flask. Ignite the precipitate, and weigh as $Fe_2O_3 + Al_2O_3$. Dissolve up the combined oxides in hydrochloric acid, reduce with stannous chloride, and titrate the Fe with potassium dichromate in the usual way. The difference after calculation will be *aluminium*.

Manganese.—To the cooled filtrate add bromine and ammonium hydrate, boil and filter. Ignite, and weigh as Mn_3O_4. Calculate to Mn."—A. R. SCOTT, *Chemical News*, August 1, 1913, p. 52. (C. T.)

MODIFIED BEAKER FOR ANALYTICAL WORK.—"This article illustrates a modified form of beaker for analytical work, having its rim indented to hold a rod. This modification enables the beaker and rod to be held securely with one hand as is shown in the accompanying illustration."—*Chemiker Zeitung*, July 26, 1913, p. 898. (T. D.)



Modified Beaker for Analytical Work.

SOUTH AFRICAN COAL FOR GAS MANUFACTURE.—"The best South African coals for gas making are those found in Natal, no good gas-making coal being yet obtainable in bulk in the Transvaal. Of the Natal coals, that from the Burnside colliery is the best. It yielded per ton, 10,500 cb. ft. of gas of 16 candle-power, 5½ gallons of tar, and 900 lb. of excellent coke containing 86% of combustible matter. The English coal hitherto employed by the South African Lighting Association at Port Elizabeth (Aldwarke Main washed nuts) gave under similar conditions 12,400 cb. ft. of 16 candle-power gas, with 10 gallons of tar and 800 lb. of coke, but costs 38s. per ton at the works. The Natal coals were not washed or picked."—J. W. BROWN (*Jl. Gas Lighting*), *Journal of the Society of Chemical Industry*, Sept. 15, 1913, p. 859. (J. W.)

METALLURGY.

SLOW-SPEED CHILEAN MILLS.—"The following data were obtained through correspondence with actual operators of the Lane type of slow-speed Chilean mills. The use of the mill was being investigated in connection with the remodeling of a cyanide plant in Nevada. The present plant consists of a 100-ton dry-crushing and leaching installation, which did not prove successful for various reasons, but chiefly because the sets of Humphrey rolls employed did not crush fine enough. Clay in the ore prevented leaching, and so it was planned to instal a grinding machine to be fed by a coarse roll product, and to classify the pulp from the grinder into sand and slime and treat these separately in accordance with the best modern practice.

All Sliming Mill.—However, extensive tests proved that the best extraction could be obtained by 'sliming' everything, and it was decided to erect this type of mill. In the meantime numbers of letters had been written to users of the Lane mill, which was one type of grinder under consideration, and replies were received from most of these giving results with some details. These all tend to show that the Lane would be an efficient machine where an extremely fine product was not desired.

It is necessary that the mill be set on a good concrete foundation which has been carefully leveled. One manager stated that by settling the pulp and using the water over and over again he had a loss of only 180 gallons per ton of ore treated. It is apparent from this and other letters received, that the capacity and fineness of product is directly proportional to the quantity of water used up to certain limits. Another writer stated that he had milled 7,000 tons without any repairs and the tires were still in good condition.

Unfortunately for my purpose, all the replies were from operators that used the mill chiefly as an amalgamator. In one plant, however, the mill was fed with a 1-in. product of a Dodge crusher and the pulp from the Lane mill went directly to cyanide vats.

All of the operators heard from advised feeding the Lane directly from a crusher, and those who were still using stamps stated that they contemplated discarding them and feeding the Lane mill from a crusher. One operator stated that he had not been satisfied and could not get the capacity required, and that the mill made too much slime for his purpose. Unfortunately, he did not go into details as to his experience with the mill.

Only two operators used screens; the fineness of product was entirely regulated by height of discharge. All agreed that the Lane type of mill was an excellent amalgamator, a good fine-grinder, and that the costs of repairs was low. Others stated that the installation cost less, the water consumption was less, and the first cost was lower than stamps of the same capacity. It seems evident that there is a field for Lane mills as intermediate grinders between crushers and tube-mills, and I would be glad to know if it has been used for that purpose in modern cyanide plants.

Tabulated results obtained from the operators heard from are given below :—

Lane Slow-Speed Chilean Mill Data.

Data.	1	2	3	4	5	6	7
Size of mill, in feet	7	10	10	7	7	10	10
Time in actual operation ...	A	2 yr.	3 yr.	G	8 mo.	18 mo.	K.
Capacity, tons in 24 hours ...	24	40	55	14	20	30	45
Horse-power required	7	10	12	?	7	10	8
Gallons of water used per minute	M	B	D	?	?	15	few
Height of discharge, inches ...	4	4	7	?	6	7	7½
Mesh of screen, if any	30	8	...	...	...	...	...
Character of ore	med.	hard	soft	hard	?	hard	med.
Mesh of discharged pulp ...	44	40	E	H	I	J	L
Time shut down for repairs ...	5 dy. p. mo.	C	2 hr.	?	...	...	...
Cost of repairs per ton	5c.	5c.	F	?	3c.	5c.	4c.
Size of feed, inches	½	1	½	½	½	1	1

Remarks.

A—Total crushed is 8500 tons.

B—Same as a 10-stamp mill.

C—Repairs made while cleaning up.

D—Have an unlimited supply of water. No record kept of consumption.

E—90% is minus 100 mesh.

F—Lower than stamps.

G—90 shifts to date.

H—80% is minus 100 mesh.

I —95% ,, ,,

J —68% ,, ,,

L —73% ,, ,,

K—Milled a total of 7000 tons to date of writing.

M—16,000 gallons per 24 hours.

Where not otherwise stated, no screens were used.

[1] In Arizona. Considers Lane better than stamps. Feed direct from crusher to Lane mill.

[2] In southern Nevada. An excellent fine grinder and amalgamator.

[3] In Idaho. An ideal amalgamator; 81% caught in the mill itself.

[4] In California. Not satisfied. Grinds too fine. Capacity limited.

[5] In Idaho. Better than stamps as a crusher; less wear and less water needed.

[6] In Nevada. Ore extremely hard; free milling; 80% of pulp through 60 mesh.

[7] In Texas. An excellent machine for fine grinding.

The above data were obtained in August 1912, from correspondence with companies actually operating the mill.

—Erich J. Schrader, *Mining and Scientific Press*, July 26, 1913. (A. McA. J.)

The Hall Ore Desulphurising Process.—"This process is based upon the principle of removing the 'fixed' sulphur atom of a sulphide by distillation and without permitting any considerable portion of the sulphur thus discharged to pass into any combined forms, such as SO_2 or SO_3, H_2S or COS, the sulphur being removed and discharged in the elemental condition. Analyses of the fumes discharged have been made over continuous runs of several hours, and the total amount of combined sulphur gas of any condition has been found to average under 0·25% by volume without admixture with any extraneous air. The rate of decomposition in the Hall process conducted in multiple-hearth furnaces appears to be from 100 to 125 lbs. of ore per sq. ft. of hearth each 24 hours, which makes it compare favourably with the capacity obtained in the best conducted roasting furnace. The cinder discharged is in excellent condition. Tests made to determine how complete a desulphurising could be produced show percentages of total sulphur remaining in the cinder of less than 1%.

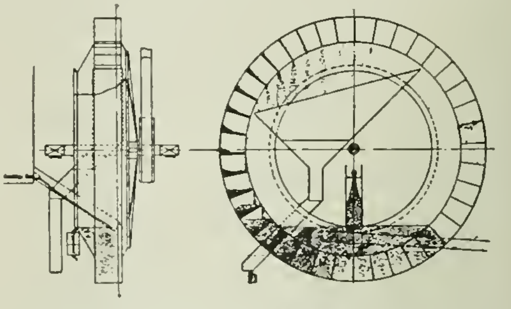
The distillation is obtained by the direct application to the ore of a burning gas flame of slightly reducing or, at least, non-oxidising character, accompanied by sufficient H_2O either as water of formation (from combustion of hydrogen) or by addition of small amounts of steam. This water is decomposed by the hot ore; the nascent hydrogen combining with any free oxygen that may enter the furnace through the atomizer through which the gases are admitted and the nascent oxygen going to the metal. It appears that nascent hydrogen has more affinity for oxygen than it has for sulphur. Various ores have been experimented with, including pyrites, pyrrhotites, copper concentrates, crude blend, zinc concentrates and even chemically pure FeS. Producer gas as a fuel requires a large amount of extraneous water, whilst gasified fuel oil appears to be ideal, containing substantially no nitrogen and having extremely high thermal values. The temperature in the furnace must be slightly above 700° C. and below 900° C. The amount of coal fuel required (gasified) is calculated at less than 10% of the weight of ore. The sulphur is extracted from the fumes by simply washing. The solids precipitated from the fumes by washing have been found to run from 98% to 99½% S."—W. A. Hall, *Engineering and Mining Journal*, No. 1 July 5, 1913, p. 35, and No. 2 July 12, 1913, p. 50. (T. D.)

Solder for Aluminium.—"This solder is an alloy composed of Al 1 pt., Zn 1 pt., Sn 12 pts. Part of the tin may be replaced by lead and sulphur may also be added."—*Journal of the Society Chemical Industry*, No. 15, Aug. 15, 1913, p. 795. English Patent 2036, 25/1/13. (T. D.)

Coating Iron with Aluminium.—"By the process of a Japanese inventor, S. Uyeno of Tokio, iron is coated with aluminium by dipping into a bath of the molten metal after being first galvanised. The aluminium bath has a temperature of 600° to 700° C. The temporary zinc or tin surface comes away in the first dipping, but for the best results the article to be coated is treated two or three times. During immersion the surface of the article is swept by a steel brush. The aluminium coating given is claimed to have extraordinary adhesion, so that it will not come off and cannot even be stripped off by mechanical means, and it does not tarnish under the action of air water or heat." *Chemical Trade Journal*, No. 1366, July 26, 1913, p. 88. (T. D.)

The Annealing of Gold.—"In the first part of the paper the effect is shown of the presence of small quantities of impurities on the temperature at which gold can be annealed. This is the first investigation of the kind, and although the exact quantitative effects in the case of metals other than gold are doubtless different from those recorded in the paper, indications are afforded of the general results which may be expected. The effects are remarkably great. For example, the addition of only 0·002% of hydrogen raises the temperature of annealing of gold from 150° to over 300°, and 0·05% of copper raises it to 250°. Further additions do not raise the temperature proportionately. It is further shown that the annealing of gold begins by the complete softening and recrystallisation of some of the laminated crystals of which the metal consists, the other laminæ remaining unaltered until more time has elapsed, or the temperature has been raised. Accordingly partially annealed metal consists of alternate strips of hard and soft material, having weak places where its rigidity and resistance to rupture are little higher than those of fully annealed material. The paper is illustrated by curves and photo-micrographs."—Dr. T. Kirke Rose, *Mining Journal*, Aug. 30, 1913, p. 841. (H. A. W.)

Dewatering Wheel for Pulp.—"At the Winona stamp-mill, Michigan, it was necessary to re-use the water, and the tailing had therefore to be separated from it. This lead to the development of the 12 ft. diameter wheel now in use. As shown by the drawing, the dewatering wheel consists of a sheet steel water-tight wheel with radial partitions along the periphery forming pockets in which the sand is caught and lifted out of the water and discharged at the top of the wheel over an apron upon the belt conveyor. The sand is run from settling vats through spigots into the bottom of the wheel. The water overflowing from the wheel is carried to the settling vats and re-used."—R. B. Seeber (from *Lake Superior Mining Institute*); *Mining and Scientific Press*, August 30, 1913, p. 339. (A. McA. J.)



Elevations of Dewatering Wheel.

Calculation of Extraction in Cyaniding.—*Sampling Mortar Discharge not Satisfactory*—"Sampling mortar-discharge pulp is not satisfactory even when water is used in crushing, and when solution is used it is futile. Unless special arrangements are made to collect the mortar-discharge pulp into one solid stream, a correct sample of it cannot be taken. The head sample ought to be taken before the ore has entered the primary crusher, but after preliminary breaking, so that there will be no large boulders to interfere with the accurate cutting out of a portion of the stream. Having the material in this appropriate form, a fair sample can be taken without regard to the size of contained metallic particles. In the cutting down of this sample every care can be taken to avoid difficulties with metallics, but when these do appear, every assayer knows how to deal with them.

In considering the sampling of the product from stamp mills, George Simpson, Jr., expresses the opinion that a fair sample of the mill head can be obtained from the pulp leaving the mortar boxes, but this I cannot altogether agree with, as has already been mentioned. Experience with trying to get a fair sample from mortars has discouraged such attempts even when water is used in crushing. A good sample cannot be taken from the lip of the mortar because the pulp is too widely spread and a cut of the entire stream cannot be had at one time, a prime requisite for a fair sample. In mills making use of plate amalgamation, the stream before beneficiation is not gathered into a compact body which can be sampled with any reasonable degree of accuracy. In other cases where water is used in the batteries and there is facility for gathering the whole stream from all batteries into one body, a sample may be taken with some claim to accuracy, exactly how much depending on the character of the ore and the sampling device. It is true that most of the mechanical sampling appliances in use are not accurate because only a portion of the stream is taken at any one time, the cutter or entry slot of the sampler often being excessively narrow. In sampling a stream of pulp, as well as with a stream of dry ore, the requirement is that the sample must take the whole stream for part of the time, and not a part of the stream part or all of the time. It would be comparatively simple to make a machine to comply with all the requirements, but it is seldom done. Satisfactory accuracy is not obtainable over long periods of time by hand sampling. The average employee of the class usually available for such work does not have the necessary interest and cannot be depended upon to carry it out intelligently.

Solution Crushing Prevents Mortar Sampling.—When solution is used for crushing, the amount of metal dissolved during the crushing period and the amount introduced with the formerly-used solution, presents an obstacle to accurate sampling that I have found insurmountable. The argument that the erratic extraction results appearing from month to month will be equalized and corrected in the yearly summary, is correct in instances when samples are correctly taken, but it is doubtful if a mortar-discharge sample would ever show correct results, especially when solution is used for crushing. Much complication might result from the attempt to assay a pregnant cyanide solution entering the mortars, determine its proportion to the dry ore crushed, assay both the pulp and solution issuing from the mortars, and calculate the correct content of the dry ore treated. Opportunities for error are so many that it seems most unwise to make an attempt which will almost surely result in error.

Formula for Extraction Percentages without Ore Weights.—For estimation of percentage extractions it is necessary to know the weight of the ore and the content of both feed and tailing. Under some circumstances, the weight of ore milled may be omitted, but in those cases the content of feed, tailing and product must be known with accuracy. Having this information, the formula quoted by Mr. Welton,

$$\text{Per cent extraction} = \frac{\dfrac{A - B}{C - B} \times 100\,C}{A}$$

where A is the assay of the feed; B that of the tailing, and C, that of the concentrate or product, is applicable and can be used with advantage.

It is not improbable that this formula might be applied with good effect in such cases where weights of ore handed are not to be easily secured. By no means should it be forgotten, though, that in making such calculations the content, or assay value of all the factors used must be accurate if correct results are expected. Sampling of the head, tailing and product must be as nearly exact as it is possible to make it.

Percentage Calculations Made by Many Methods.—All possible precautions as to weighing and sampling having been taken, there is still room for some difference of opinion about the method of calculating extractions. When simple, continuous processes are followed, this difficulty does not arise. Thus in concentration, amalgamation, and in a great measure the process of continuous slime agitation, the ore is not retained in process of treatment for great lengths of time and, only one treatment being given, tailing samples taken on a certain day will approximately correspond with head samples taken the same day. Calculated extraction percentages over monthly periods, in such cases, may be considered reasonably accurate. Dealing with combination processes, however, the solution is not so easy. Consider, for example, the system followed by so many mills treating silver-bearing ore, where concentration followed by separate sand and slime treatment is practised. Concentration is quickly completed, the head, tailing and production checking up each day. Then follows slime treatment, the feed of which is part of the tailing from the concentrators. Three to ten days are occupied in carrying out the details of the process, consequently the daily percentage calculations cannot be correct and even over a month they are likely to be erratic. When, however, one considers the sand treatment, lasting from 10 to 40 days, it will be readily comprehended that over a monthly period extraction percentages must be calculated by some other method than the simple use of mill-feed and tailing assays, which cannot have any definite relation to each other. There are, however, methods which may be applied under such circumstances which will give an idea, and a close one, of what is actually going on in such a plant. Such estimates are usually required in monthly reports, but their most important use is to the metallurgist in charge of operations. He must know what he is doing at all times so that he may be ready to correct any error or change his treatment to suit a changing ore. This situation has already been discussed, and a proposed solution is to use the mill feed as 100%, and the feed to the cyanide plant as a different head. If concentration extracts 20%, then 80% goes to the cyanide plant. If the latter extracts 80% of what it receives, it amounts to 64% of the mill feed, and 64% plus the 20% extracted by cyanidation gives 84% as total extraction.

That it is not possible to secure accurate samples of either feed or tailing is a theory in which I cannot place confidence. If the proper mechanical appliances are installed to take samples in the right way, and these mechanical means receive vigilant supervision from persons conversant with their objects and requirements, a good and accurate sample may be obtained. Believing that it can be done and knowing that many operators do not do it, it is natural to wonder why. There are plenty of examples of mills which installed sampling and weighing apparatus and then discontinued their use. The sampling plant of the Goldfield Consolidated mill was not rebuilt after having been destroyed by fire, and it is to be supposed that the operators did not consider it essential.

Does Correct Sampling Pay?—I believe that actual sampling and correct extraction calculations pay—in real money. I believe that the losses accompanying the other method are more costly than proper sampling would be. By the calculation method, losses inside the mill cannot be determined. Stealing may occur without detection. Leaks may be in the tanks and pregnant solution may be lost without suspicion. Precipitate may fall on the floor and be rubbed into it, or may be washed away, and no intimation of such loss will come through the figures. Careless operators may let tanks run over and waste the solution; it will never show and no one will know of it except those who actually see it. It puts a premium on carelessness. Whenever a calculation can be used in all kinds of cases and always check up correctly, even with bad work, there is something serious the matter with it. Besides these facts, I do not believe that correct percentages are obtained by the calculation method even where serious direct losses do not occur. I believe that extractions of silver from sulphide and other complex ores rarely run high in the nineties, and that the average extraction would be much nearer to 80% to 85%. Tha accurate sampling and calculation of extractions, recognition of losses and earnest endeavours to avoid them and strong efforts actually to raise extractions to the point now said to have been reached, would be a wise and economical course to pursue, I fully believe."—HERBERT A. MEGRAW, *Engineering and Mining Journal*, Sept. 6, 1913, p. 441. (A. R.)

CHENHALL'S PATENT TIN EXTRACTION PROCESS.—"So much has lately been heard of 'losses of tin' during the concentration processes, which the present system of dressing the tin ore involves, that one need scarcely wonder that a metallurgist of Mr. James W. Chenhall's repute should enter the arena and endeavour, with every appearance of success, to devise a process which should substantially increase the percentage of extraction at a cost more than covered by the saving of by-products, which are saved more extensively and in a purer state. The Chenhall process is a chemical one, by which the metals contained in the ore are made soluble in acid solutions, and recovered therefrom by means of precipitation. The ore as raised from the mines is first crushed to a suitable size, then mixed with fine coal dust or other carbonaceous material, and heated in a closed furnace until the oxides of the metal are decomposed, leaving the metals in a metallic form distributed throughout the roasted ore in the form of fine metallic grain, care being taken to prevent reoxidation of the metal, or metals, during the cooling of the ore. The gas disengaged during the roasting is collected and stored, and when the flow of gas from the mixture of coal and ore almost ceases, that which has been stored is returned to the furnace, so that during the whole of the time the ore is in the furnace it is in an atmosphere of coal gas. The coal mixed with the ore, being converted into coal gas, keeps the ore open, and allows the gas in the furnace to permeate the whole of the ore, as well as acting as a deoxidising agent. The cold roasted ore is afterwards submitted to treatment with acid solutions. When tin is the predominant metal, the first solution consists of sulphuric acid and water, and should there not be sufficient iron in the ore to produce iron sulphate, a little iron is added; about 85 per cent. of the tin contained in the ore is dissolved out by means of this solution. The ore is now washed with water, and the final solution (hydrochloric acid and water) is added, by which the remaining tin is dissolved out. When any copper is present in the solution, which is always the case when it is present in the ore, it is precipitated from the solutions by means of metallic iron (wrought scrap). Any insoluble material in the liquor is allowed to settle, and the clear liquors treated with sulphuretted hydrogen, which at once precipitates the tin in the solutions in the form of sulphide, which is collected, dried, and heated, slowly at first and then to a dull red heat, which converts the sulphide into oxide of tin, and the sulphurous acid conducted to chambers for the production of sulphuric acid. The waste liquors, after the tin has been extracted, are evaporated. The first solution for the manufacture of sulphate of iron, and the second chloride solution for the manufacture of red oxide of iron (rouge): by this means the excess of acid necessary through the presence of iron in the ores is converted into a saleable product, thus recovering the value of the acid so used as a valuable by-product. The acid solutions are continually passed over fresh lots of roasted ore until practically all the acid is combined with the metals, and the ores receive a sufficient number of fresh acid solutions on them until the whole of the soluble metals are dissolved out. The amount of acid consumed depends on the character of the ore to be treated, ores from different mines varying in the amount of soluble materials they contain. Tungstic acid, when in the ore, remains in the residue, and is extracted by means of an alkaline solvent, and recovered from the solution as very pure tungstic acid. The loss of tungstic acid with the present methods of recovery of the wolfram in tin ores is 75% of the wolfram in the ore. The plant needed for this process is that required for the manufacture of the two acids (sulphuric and hydrochloric); suitable closed furnaces in which, during the roasting of the ores and their cooling, the air is excluded, and the presence of carbonated hydrogen gas is maintained during the roast: the means for effectively leaching the roasted ores, precipitating the tin, and the recovery of the by-products, including the acid used, when not sold in the form of by-products. The cost of the process will vary with the metallic contents of the ore, and the amount of by-products there is to recover from the ore. The roasting will cost from 4s. 6d. to 5s. 6d. per ton, and the leaching a further 1s. 6d. to 2s. per ton. The cost of the production of the necessary acid (sulphuric) will be, say, 45s. to 50s. per ton, and the hydrochloric about 80s. to 85s. per ton. A large percentage of the acid will be recovered for further use in the process."—W. CHENHALL, *Mining Journal*, Sept. 6, 1913, p. 854. (H. A. W.)

MAGNESITE CRUCIBLES.—"The need for a crucible which shall be more inert to its charge than either fireclay or silica, and without reducing action, such as the graphite has, is probably one which sooner or later arises in every assay office. Theoretically, the need is met by magnesite, and the following is merely a brief account of the search for data on this subject. Crucibles made according to a description in the *Wisconsin Engineer*, November, 1912, were satisfactory for work in the electric furnace of the resistance type. The directions are, to moisten the calcined magnesia with water, and tamp it around a wooden core set in a crucible turned from a bar of Acheson graphite. The core is drawn and the crucible set in an oven heated by gas, until thoroughly dry. It is then covered with a thin graphite cap, buried in a granular carbon resistor, and heated to a temperature a little above that in the work proposed. After so baking the magnesia linings are readily removed from the crucibles. For use, they are replaced in the graphite crucibles, charged as desired, covered with a magnesia disc, and are then ready for the furnace. Commercial calcined magnesite was found by O. P. Watts, author of the paper referred to, to vary greatly in the amount of iron oxide contained. This is detrimental, because in baking the linings, it distils out, is reduced to iron in contact with the inner wall of the crucibles, and this iron is then likely to corrode both the crucible and the lining by carrying carbon from one to the other. Any considerable amount of carbon dioxide indicates insufficient calcining, and such material will show excessive shrinkage in baking. If the shrinkage is excessive, the magnesite may be fused before using in the electric arc This calcined magnesite will probably not shrink enough, so a little (2% to 10%) raw material may be mixed with it to give it suitable working qualities. The extreme temperature at which such magnesia linings may be used in graphite crucibles in a resistance furnace is fixed, not by the fusing point of magnesia, but its decomposition by the graphite. On account of this action, the granular resistor should never be allowed to come in contact with the magnesia linings. I have been informed that magnesite apparatus is made to order by the Royal Berlin Porcelain Co., and that it can be obtained through the Wilson-Maenlen Company, of New York (presumably from other agents for the Royal-Berlin crucibles, etc.. also). I have not used any personally, but am told it is entirely satisfactory, except for the time consumed in getting it. The magnesite crucibles are said to be rather more porous than are the fireclay crucibles, resembling in texture the sand crucibles. This is hearsay only, however. For muffle use, where a basic crucible is required for a basic charge, I have made satisfactory crucibles by machining magnesite bricks. These work well under drill press or on the lathe, although the material must be handled far more tenderly than iron or steel, so use slow speeds and light cuts. These machined crucibles must be set down in a clay or silica crucible and protected by magnesite powder or some other similar material, so as to take the strain of the tongs off the magnesite. Wet charges cannot be used in them except with the utmost care in heating. It is better to avoid them altogether. Chrome brick, the alternative to magnesite, which naturally comes to mind, is not a feasible proposition, because of the hardness of the material, which immediately puts the tools out of business.—DONALD M. LIDDELL, *Engineering and Mining Journal*, Sept. 13, 1913, p. 503. (A. R.)

MINING.

THE LARGEST ELECTRIC MINE HOIST.—"Electrical developments are proceeding at a very rapid rate in connection with the South African mining industry, and there is much plant there which deserves careful attention on account of its technical interest, and also on account of the magnitude of the power involved. What is probably the largest electrically operated mine hoist in the world is now, the *Electrician* understands, in course of erection in connection with the undertaking of the Crown Mines (Ltd.) This company is equipping its shaft with a 4,000 h.p. hoist which will operate on the mains of the Victoria Falls Power Co. without any attempt at equalisation of the maximum and minimum power demand. The important nature of this proposal will be realised when it is remembered that with a hoist of this description the motors will require something like 7,000 h.p. during acceleration, resulting in a load of 9,000 h.p. on the transmission line. The electrical equipment, which has been built by the General Electric Co., of Schenectady, consists of two 2,000 h.p. continuous current motors, one being connected at either end of the drum shaft of the hoist itself, which has been built by Messrs. Fraser & Chalmers, of London. The drums which the motors will drive are of the cylindro-conical type, the smaller diameter being 12 ft., and the larger 20 ft. 8 ins. The speed of the drum is 53·5 revolutions per minute. The voltage at which the continuous current motors operate is 500 volts, the motors running in series, and being controlled on the Ward-Leonard system. These hoist motors will receive their power from a motor-generator set converting the 50-cycle three-phase power to continuous current. This motor-generator consists of a 5,000 h.p. three-phase induction motor, operating on the 50-cycle 2,000 volt circuit at 375 revolutions per minute. It is direct coupled to two separately excited continuous current generators, each of 1,650 kw. capacity. These generators are connected in series, and have compensating and commutating field windings to prevent sparking. In order to supply the necessary current for exciting the fields of the two hoist motors and the two generators a 60 kw. shunt-wound exciter is provided. The magnitude of this equipment will be more fully realised when it is mentioned that the total weight of the motor-generator set will be 138,000 lbs., while the total weight of each hoist motor will be approximately 142,000 lbs., exclusive of shafting and bearings. When it is remembered that this mammoth electric hoist is only one of a hundred or more electric hoists operating on the lines of the Victoria Falls Power Co., the magnitude of the electrical interests involved in the development of the South African mining industry will be realised."—*Science and Art of Mining*, Sept. 27, 1913, p. 73. (A. R.)

INDIAN GOLD MINING METHODS. — "*Modern Equipment—Serious Accidents—American Motors.*—In paying a recent visit to this Kolar gold mining field I was shown over some of the interesting surface workings, where all the most up-to-date crushing and extracting machinery is used, but it was stated that visitors are not permitted to go underground because of the recent appalling number of serious accidents that have been persistently recurring, due to the phenomena of air blasts and quakes. The cause of these is not entirely understood, but is supposed to be due to the failure of pillars to withstand the tremendous pressure from above, the mines now reaching to depths of 2,500 ft. to 5,000 ft. below surface levels. Scarcely a month goes by without such accidents occurring. I visited the hospital at Kolar, which is maintained by the companies in this gold field for the employes, and found that it was full of those who had been injured by these distressing occurrences. The problem of protecting the mines and their employees from these accidents is a most serious one and much scientific attention is now being paid to it.

Air Compressors — Crushing and Extracting Machinery.—The use of air compressors is a highly important feature of these mines, as it is only by reliance on them that the thousands of employees working at great depths below can secure air to keep themselves alive. Both American and British air compressors are used. The usual type runs at moderate speed with a final compression of 60 lb. to the square inch and delivering about 2,300 cub. ft. of free air per minute. Such compressor has two stage cylinders, with a stroke of 48 in., the air being taken into the low-pressure cylinder through a cooling chamber which reduces it about 8° below the temperature of the atmosphere. Improved cages have lately been introduced at the mines for rendering the handling of men up and down safer and more expeditious. Twenty men can be raised or lowered each trip. The brakes and reversing gear are actuated by steam. The cages, which can hold a gross load of 12,500 lb. are double decked, run in wooden guides, and are fitted with safety catches and hooks.

All of the crushing and extracting machinery by the cyanide process is of British manufacture. These companies, with their large earnings, are progressive in expending money for improvements and do not hesitate to scrap machinery when a more efficient or suitable kind can be procured. For instance, the management of the Ooregum mine, which has produced about $10,000,000 worth of gold since it began operations, has just decided upon the expenditure (most of it during the current year 1913) of $125,000 on the complete reorganisation of its milling plant, in order to get a closer separation of the fine ore and the coarse ore.

The coarse ore will go to a portion of the existing stamp battery and be crushed through a screen of probably 30 holes per square inch and the fine ore will be put through Chilean mills with screens of 900 holes per square inch. It is calculated that about 20 stamps and two Chilean mills of the latest fast-running pattern will do the crushing necessary at this stage. The pulp from the battery and Chilean mills will pass to diaphragm cones, from which the underflow will go to two tube mills, subsequently joining the overflow, and the total product passing over amalgamating plates in a separate building where the amalgam will be finally dealt with and converted into bullion. The tailing from the amalgamating plates will return and be raised to secondary diaphragm cones, placed above the first two cones so that the overflow feeds directly into the first cones and thus back to the tube mills and amalgamating plates. The overflow from the secondary cones will pass to a double diaphragm cone from which the overflow feeds directly into the sand agitators; the overflow from the double diaphragm cone will pass to two or more Dorr thickeners. As regards sand treatment, the sand will be mixed with barren cyanide solution in the agitators, in which practically the whole of the gold will be dissolved, after which the mixture will be pumped to a sand filtering vat from which the gold-bearing solution and washes will be removed through a vacuum chamber and sent to the storage vats of the precipitation plant. The residues from the filtering vat will be mixed with barren solution and pumped to a residue dam. The slime from the Dorr thickeners will go to slime agitators, from which after solution of the gold it will be pumped to the pulp storage vat of a filter plant. The discharge slime cakes will then be pumped to a residue dam, while the gold-bearing solution will go to the storage vat of the precipitating plant.

The annual output of the great Mysore mine is gradually approaching $5,000,000, the amount last year being the largest in the long history of the undertaking. The increase last year was due to the treatment of slime by the experimental plant, the result of which was considered sufficiently favourable to justify the installation of a large permanent plant. A further decrease in costs is expected to result from various improvements which are now to be carried out.

Fire Protection.—In view of the large amount of timber used in the Kolar gold mines for supports, considerable attention is given to organisation and equipment for dealing with outbreaks of fire. The precise arrangements vary at different shafts and in different mines, but the general scheme followed involves the installation of water pipes, hydrants, and hose connections in all the principal shafts, the fixing of numerous fire doors for regulating or stopping the air currents and for isolating various sections of the mine, and, finally, the organisation of special patrols for inspection of the workings. All the main vertical shafts and some of the incline shafts have perforated pipes or sprinklers at or near the collar of the shaft and connected with the main water supply. Fire hydrants are also installed at the shaft top and connected with the water supply, and where the pressure is insufficient pumps have been installed and are kept in readiness. For the supply of water below ground pipes are carried down the principal shafts and connected with the water supply. At the various levels hose connections are provided and a large supply of fire hose (up to 3,000 ft. in some cases) is kept in readiness to be sent down to any point required.

In some cases cisterns with automatic float valves, and connected to the rising main, are provided at a number of levels below ground and the pipes of these fitted with hose connections. Where the water is taken right down from the surface, reducing valves are introduced at various points in order to keep the pressure within reasonable limits. In some cases additional pipes have been laid along certain levels from the mains in the shafts and to a large extent the various pipes from the underground pumps have been arranged so that special fittings, which are kept on hand, can be readily attached for making hose connections. Considerable progress has already been made in the way of providing fireproof doors for isolating various sections of the mines and controlling the air currents. In the Ooregum mine a noticeable feature is the provision of iron doors, beneath each chute-pass in all of the principal shafts, on to which stuff can be run so as to completely block the shaft. At various levels below ground small portable chemical fire extinguishers are kept in readiness, also fire buckets filled with water, and in the case of electrical hoists and pumps, buckets of sand. Other supplies are kept on surface in special stores ready to be sent below whenever required.

Rescue Appliances.—The question of rescue appliances for enabling men to enter a place charged with injurious gases has received considerable attention on the part of the mining board of the Kolar gold field. Some years ago some of these appliances were found to be unsatisfactory."—HENRY H. BAKER.—*Mining and Engineering World*, Sept. 27, 1913, p. 558. (A. R.)

SELECTIVE MINING.—"'Selective mining' means working all payable ore, and avoiding all unpayable ore as far as possible. This has a great bearing on working costs, and, of course, many critics will tell you that working rich ore and poor ore together will be of advantage to shareholders, as the mine will be worked to a greater extent than by selective mining. These critics lose sight of the fact that the loss arising from the poor ore has to be made good by the rich ore, thus reducing the total profit proportionately. They may say that lower working costs are obtained in the way advocated by them, but if they calculate the cost at per ounce recovered, instead of at per ton milled, they will realise their error. On the other hand, it must be admitted that it is impossible always to prevent stoping poor ore with rich ore—as, for example, when both are intermixed in the same face. In such case I approve of their being worked together, as if the poor ore is left in the mine after it has been broken, the loss would perhaps be greater than if it had been extracted and treated. But whenever the unpayable ore *in situ* can be left in the mine without being stoped, I contend that it would be most unwise to stope it with the only object of feeding the mills and without any chance of increasing the profits. I hope, and I think, working costs will be reduced in the near future without resorting to the milling of unpayable rock, and I believe that sand-filling, by rendering the mines safer, reducing in the long run the cost of maintenance of the workings (old or new), and by facilitating ventilation through a reduction of the area through which the air can be directed to or from the working faces, will also be recognised in the near future as a money-saving device. Another reason which makes me support selective mining is that ways and means may yet be found of turning to account the unpayable ore left in the mines. We all think that improvements will take place, as we cannot say that we have yet attained perfection in either mining or metallurgy. Next generation may bring about a revolution in all our present processes; who can tell the contrary? In those circumstances it would be foolish to work to-day at a loss rock which will perhaps in future yield a profit."—E. J. Renaud.—*South African Engineering*, Aug. 1913, p. 32. (A. R.)

Disadvantage of Sprays in Drilling.—"The existing Queensland laws in general require the use of water for the prevention of dust in underground drilling. It is stated, however, in the annual report for 1912 of the Under-Secretary for Mines that the rule is so framed as to permit variations to some extent, at the discretion of the inspector. The necessity of this provision became apparent in several of the deep mines during the last year. It was found that the increased humidity produced uncomfortable conditions and even in some cases increased the danger of accidents. For instance, the excessive use of water at the drills caused the filling in one of the Mount Morgan stopes to become saturated with water and this was shown by inquiry to have been the chief cause of an extensive collapse of timber."—*Engineering and Mining Journal*, Sept. 13, 1913, p. 500 (A. R.)

Mining Methods at Cobalt.—"In the *Canadian Mining Journal* for August 1, Reginald E. Hore writes about methods adopted at Cobalt for mining the narrow lodes of rich silver ore. In the early days open-cuts were general. The rock on one side of the vein was broken and removed, and the vein-matter picked off the wall and bagged. Many open-cuts were worked to depths of 50 or 100 ft. It was not for a long time that shaft-sinking and driving were commenced, and even then it was usual to leave the vein on one wall and collect the ore in bags. In this way much fine rich ore was lost in the broken rock and removed with it to the dumps. Things have greatly improved since then. The present practice in development is to sink vertical shafts in the veins, and drive at intervals never more than 100 ft. and often 50 or 75 ft. Other veins are reached by cross-cuts, and developed by winzes and drifts in a similar manner. At most mines there is a central hoisting shaft. Instead of leaving the vein on the wall, it is now usual to keep the vein well within the working face. The rock on each side for a few feet contains enough silver to pay for milling. In some mines, care is taken to keep the rich ore separate, while at others everything is mixed and hoisted to the surface where it is picked, jigged, and tabled.

In stoping, some mines use the shrinkage system, but at others stulls and lagging are put at short intervals so that little ore shall remain in the stopes. At the Crown Reserve, square-set timbering is employed. Good examples of the shrinkage system are provided by the Nipissing and Coniagas mines. At Nipissing, drifts are run 14 ft. high and 5 ft. wide. If the walls consist of milling rock, or if there are two or more veins close together, the drifts may be wider, in some cases as much as 12 ft. The veins are kept well within the breast. The drilling for the whole 14 ft. is done from one set-up, so each set-up is for an advance of 5 or 6 ft. with a height of 14 ft., and a width of 5 to 12 ft. A cut 8 ft. high is taken 5 or 6 ft. in advance of the remaining 6 ft., which is drilled by uppers. The level is protected by a lagging of poles laid on caps supported by posts. Chutes are built at intervals of 25 ft. Before stoping a large block, rises are cut into the level above. These are cut by two men using small stoping drills. A man-way is constructed by timber partitions, and the remainder of the rise is filled with ore as it is broken. The miners work on a platform which protects the man-way and extends across the broken ore. To maintain the desired working space, ore is drawn off from a chute at the bottom of the rise. This chute is afterwards used for ore from stoping. When the rise is completed, it is used to provide a stoping face, and the ore is broken on each side. The rise provides ventilation, and it is used to lower steel and as a man-way. The high-grade ore is not kept separate. The cars are taken to surface on a cage.

At the Coniagas, the methods of development and mining are similar to those at Nipissing, with some variations. The drifts are run 8 ft. high, and are afterwards enlarged by a cutting-out drift, giving 8 ft. additional height. The lagging is supported by stulls set in hitches. Where the drift is wide, posts are used to support stulls in the middle. Chutes are built at intervals of about 25 ft. Stoping is begun by breaking some ore carefully on to the lagging, and is then continued by the shrinkage system. In order to avoid pop-holing and sledging of large pieces of rock, care is taken to break the rock comparatively small. Much high-grade ore is picked underground. In the drift-stopes, a sorter examines the ore as it is loaded into the cars and bags the best quality. In the stopes there are no sorters, but the miners themselves sort as much high-grade ore as possible. It will be seen that while at Nipissing the sorting is done at the surface, at Coniagas it is largely done underground."—*Mining Magazine*, Sept. 1913, p. 222. (A. R.)

Effect of Incombustible Dusts on the Explosion of Gases.—"The authors have carried out many experiments to test the question whether finely divided magnesia or shale dust acts catalytically in propagating flame or merely behaves as an inert substance.

Various opinions are held as to the action of incombustible dusts. On the one hand the opinion is held that these dusts act the same part when mixed with coal dust, as inert gases do when mixed with explosive gas mixtures; on the other hand that the dust might promote the combustion of fire damp on its surface and the experiments of the late Sir F. Abel have been quoted as proving that danger from such catalytic action is by no means remote.

Abel found that a 3% fire damp mixture passing a lamp flame at a velocity of 600 ft. per sec. produced no result, but when the mixture was caused to convey calcined magnesia in suspension, long flares were produced within a few seconds and the inflammation speedily spread through the gallery with feeble explosive effect. Abel attributed these results to the 'localising and intensifying' of the chemical action by the heated particles, which he imagined might carry the flame along by passing in a state of intense incandescence into regions of unburnt gas. Later he considered the action was catalytic. In Abel's experiments the dust cloud also conferred on the current *before it reached the flame* the power of propagating flame backwards.

Experiments at Eskmeals failed to show any in-increased inflammability of gas mixtures due to suspended magnesia when the magnesia, gas and air were thoroughly mixed by a powerful fan before entering the gallery. The 'flares' observed when the mixtures were brought above the lower explosive limit roared, rapidly charred wood and distinct explosions were heard.

Experiments by the Home Office Committee at Eskmeals showed that flame traversed explosive mixtures more rapidly when dust-free than when laden with magnesia dust.

The authors found that by introducing dust it was possible to make an explosive mixture of gas and air non-explosive.

Under no conditions that could be produced was it possible to make a non-explosive mixture explosive by inducing combustion of a portion of the mixture by a heated platinum wire.

On the other hand it was found easy to illustrate the quenching effect of magnesia on an actual explosion by filling a long horizontal glass tube with an explosive mixture of gas and air and placing a spoonful of magnesia in the tube. A fairly rapid flame on reaching the dust threw it up and was promptly extinguished.

Later important details were discovered respecting Abel's experiments and his experiments were then repeated with the same results. Abel had taken no precautions to mix his gas and air, consequently the gas entering through a pipe in the bottom of the gallery flowed upwards through the air current and filled the upper portion of the gallery with a feebly explosive mixture.

On pouring magnesia into the gas and air current just after the point of entry of the gas, the cloud of dust made the flares more luminous and every now and then a particle of dust passing through the lamp flame was whirled upwards white-hot into the explosive mixture above and ignited it."—DIXON and CAMPBELL, *Journal Society of Chemical Industry*, July 15, 1913, p. 684. (T. D.)

AZIDES FOR DETONATORS.—"The author discusses the use of azides of the heavy metals for detonator composition and concludes that lead azide is the most suitable on account of greater density, cheapness, greater compressibility and comparative insensibility to shock and temperature elevation, when compared with fulminate of mercury."—NEITZEL.—*Zeitschrift fur das Gesamte Schiess und Sprengstoffwesen*, June 1, 1913, p. 210. (T. D.)

MINING CONDITIONS ON THE CONTINENT.—"Upon the return of the South Wales mining students from their tour of the colliery districts of Saxony, Mr. Henry Davies, in an interview, said one of the surprises of the tour was encountered at Freiberg, where the celebrated State smelting works are established. Here, nothwithstanding the fact that the same kind of work is carried on as at Landore, there are green meadows, waving cornfields, fruit trees, and dense foliage to be found closely adjoining the works. All the fumes from the various processes were carefully collected and repeatedly analysed, so that there was not the slightest escape of anything which could be turned to any value, with the result that the country gained not only in the recovery of valuable by-product, but in the prevention of poisonous fumes which, near Landore, have been such a source of trouble to farmers and others. It was stated that these German scientists are so careful in their recovery of all waste material that may possess the slightest value that even the coats of workmen engaged in jewellers' and goldsmiths' shops are taken to the Freiberg laboratories and burnt, in order that the gold dust may be recovered. There was ample evidence on all sides to show that the mine owners interested themselves very closely in the home life of the workmen, and efforts were made in many directions by the owners themselves to build either model villages or model homes for the workmen. The houses were invariably rented at a very low rate. The hours worked by the miners were longer than those in South Wales, and the conditions under which the work was carried on were not equal to those which obtain in the best Welsh collieries. In some of the collieries the conditions astonished the visitors, as in many cases a Saxon miner would be found discharging his duties in an absolutely nude condition, clad, as Mr. Davies put in, in nothing beyond a smile. In one colliery at least one hundred men were thus found, the cause being of course the high temperature prevailing in the working places." *Science and Art of Mining*, Sept. 13, 1913, p. 51. (A. R.)

HOW DETONATORS SHOULD POINT.—"It is commonly asserted that the maximum effect of a detonator is exerted at its business end, and therefore that this should point towards the bottom of the hole where it is desirable to produce the heaviest blasting effect. This point has been proved on a large scale in practical quarry work, says *Coal Age*, March 22, 1913, by priming half the holes in a uniform ledge of 28 holes with the detonator pointing towards the mouth of the hole. The other holes were loaded with the detonator pointing towards the bulk of the charge. The ledge in which the detonator pointed outward was thrown off to within 2 ft. of the bottom of the hole. There the rock was broken in places but was not thrown out. In the other half of the ledge, where the detonator pointed toward the bulk of the charge, the rock was thrown off clear and clean and nearly 2 ft. below the bottom line of the hole, taking off about 30% more material than the same charge did with the detonator pointed the wrong way."—*Engineering and Mining Journal*, May 3, 1913, p. 900. (T. D.)

MISCELLANEOUS.

WATERPROOFING OF CONCRETE STRUCTURES.—"While the committee has not arrived at sufficient conclusions to formulate definite specifications for making concrete structures waterproof, certain results have been reached which may be of assistance in securing impermeable concrete.

The work of the committee was found to be divided naturally into three branches:—

(1) Causes of permeability, and methods of avoiding them.

(2) Waterproofing by the addition of other substances.

(3) Subsequent treatment of exposed surfaces.

In the laboratory and under test conditions using properly graded and sized coarse and fine aggregates, in mixtures ranging from 1 cement, 2 sand and 4 stone to 1 cement, 3 sand and 6 stone, impermeable concrete can invariably be produced. But the fact remains, nevertheless, that the reverse often obtains in actual construction, permeable concretes being found even with 1 cement, 2 sand and 4 stone mixtures and are of frequent occurrence where the quantity of the aggregate is increased. This the committee attributes to:

(*a*) Defective workmanship, resulting from improper proportioning, lack of thorough mixing, separation of the coarse aggregate from the fine aggregate and cement in transporting and placing the mixed concrete, lack of density through insufficient tamping or spading, and improper bonding of work joints, etc.

(*b*) The use of imperfectly sized and properly graded aggregates.

(*c*) The use of excessive water, causing shrinkage cracks and formation of laitance seams.

(*d*) The lack of proper provision to take care of expansion and contraction, causing subsequent cracking.

Properly graded sands and coarse aggregates are rarely, if ever, found in nature in sufficient quantities to be available for large construction, and the effect of poorly graded aggregates in producing permeable concrete is aggravated by poor and inefficient field work. Even if the added expense could be afforded of screening and remixing the aggregates so as to secure proper granulometric composition to give the density required to make untreated concretes impermeable, it is seemingly a commercial impossibility on large construction to obtain workmanship even approximating that found in laboratory work.

The committee finds that in consequence of these conditions substances calculated to make the concrete more impermeable—either incorporated in the cement or added to the concrete during mixing—are often used.

While it has been impossible for the committee to test all of the special waterproofing compounds on the market, it has investigated a sufficient number of these, as well as the use of certain very finely divided mineral products, to form a general idea of the value of the different types. The committee finds :

(*a*) That the majority of patented and proprietary integral compounds tested have little or no immediate or permanent effect on the permeability of concrete and that some of these even have an injurious effect on the strength of mortar and concrete in which they are incorporated.

(*b*) That the permanent effect of such integral waterproofing additions, if dependent on the action of organic compounds, is very doubtful.

(*c*) That in view of their possible effect, not only upon the early strength, but also upon the durability of concrete after considerable periods, no integral waterproofing material should be used unless it has been subjected to long-time practical tests under proper observation to demonstrate its value, and unless its ingredients and the proportion in which they are present are known.

(*d*) That in general more desirable results are obtainable from inert compounds acting mechanically than from active chemical compounds whose efficiency depends on change of form through chemical action after addition to the concrete.

(*e*) That, assuming average quality as to size of aggregates and reasonably good workmanship in the mixing and placing of the concretes, the addition of from 10% to 20% of very finely divided void-filling mineral substances may be expected to result in the production of concrete which under ordinary conditions of exposure will be found impermeable, provided the work joints are properly bonded and cracks do not develop on drying, or through change in volume due to atmospheric changes, or by settlement.

While external treatment of concrete would not be necessary if the concrete itself, either naturally or by the addition of waterproofing material, was impermeable, it has been found in practice that in large construction, no matter how carefully the concrete itself has been made, cracks are apt to develop, due to shrinkage in drying out, expansion and contraction under change of temperature and moisture content and through settlement.

It is, therefore, often advisable on important construction to anticipate and provide for the possible occurrence of such cracks by external treatment with protective coatings. Such coating must be sufficiently elastic and cohesive to prevent the cracks extending through the coating itself. The application of merely penetrative void-filling liquid washes will not prevent the passage of water due to cracking of the concrete. The committee has, therefore, divided surface treatment into two heads :

(*a*) Penetrative void-filling liquid washes.

(*b*) Protective coatings, including all surface applications intended to prevent water coming in contact with the concrete.

While some penetrative washes may be efficient in rendering concrete waterproof for limited periods, their efficiency may decrease with time and it may be necessary to repeat such treatment. The committee, therefore, believes that the first effort should be made to secure a concrete that is impermeable in itself and that penetrative, void-filling washes should only be resorted to as a corrective measure.

While protective extraneous bituminous or asphaltic coatings are unnecessary, so far as the major portion of the concrete surface is concerned, provided the concrete is impermeable, they are valuable as a protection where cracks develop in a structure. It is therefore recommended that combination of inert void-filling substances and extraneous waterproofing be adopted in especially difficult or important work of any nature.

Considering the use of bituminous or asphaltic coatings, the committee finds :

(*a*) That such protective coatings are often subject to more or less deterioration with time, and may be attacked by injurious vapors or deleterious substances in solution in the water coming in contact with them.

(*b*) That the most effective method for applying such protection is either the setting of a course of impervious brick dipped in bituminous material into a solid bed of bituminous material or the application of a sufficient number of layers of satisfactory membranous material cemented together with hot bitumen.

(*c*) That their durability and efficiency are very largely dependent on the care with which they are applied.

Such care refers particularly to proper cleaning and preparation of the concrete to insure as dry a surface as possible before application of the protective covering, the lapping of all joints of the membranous layers, and their thorough coating with the protective material. The use of this method of protection is further desirable because proper bituminous coverings offer resistance to stray electrical currents.

In conclusion, the committee points out that no addition of waterproofing compounds or substances can be relied upon to counteract completely the effect of bad workmanship, and that the production of impermeable concrete can only be hoped for where there is determined insistence at all times on good workmanship on all structures." — AMERICAN SOCIETY OF TESTING MATERIALS, *Engineering Magazine*, Sept. 1913, p. 899. (J. A. W.)

LIQUID FUEL.—"This deals with the origin, composition, production, uses, and future supply of petroleum. The author considers that the greater quantity of petroleum has been formed by the slow decomposition of marine vegetation (sea weeds, etc.). A table from A. Beeby Thompson's 'Petroleum Mining' is given, showing the specific gravity and ultimate composition of the various kinds of petroleum, whilst in another table the specific gravity, flash point, calorific value, and actual evaporative power of various kinds of liquid fuel are given; heavy tar oil is shown to be considerably inferior to a good petroleum fuel oil as a fuel. It is stated that air carburetted with petrol vapour gives good results in the Bone and McCourt system of flameless surface combustion, and experiments have also indicated that fuel oil, finely atomized with air in the correct proportion, and then blown into the tubes containing the granular materials will also prove efficient if the granular material be previously heated."—V. B. LEWES, *Jl. Roy. Soc. Arts*, lxi., pp. 666, 690 and 702; *Journal Franklin Institute*, Sept. 1912, p. 344-5. (J. W.)

EXTINGUISHING FIRES WITH SAWDUST.—"The article describes tests made on the extinguishing of fires with sawdust. Fires in volatile lacquers, varnishes, gasoline and similar substances were extinguished. The tests were made in tanks containing the inflammable material and were conducted out of doors while a wind was blowing at the rate of 19 miles an hour. The tanks were three in number and of different sizes.

Both softwood (pine) and hardwood (oak) sawdust were used, dry and moist. A portion was also mixed with sodium bicarbonate. The extinguisher was applied by means of a snow shovel after the inflammable material had been thoroughly ignited. The fires were all extinguished in 8 to 25 sec. using the sawdust. The sawdust containing bicarbonate (1 to 3) extinguished the fires in 8 to 9 sec. Sand and ordinary fire extinguishers operated much more slowly.

The conclusions are that sawdust spread on the surface will readily extinguish fires in inflammable liquids contained in moderate sized tanks or small fires in these liquids on the floor. Gasoline fires in tanks were extinguished without difficulty, but there may have been some question as to whether sawdust would have worked equally well on very large tanks. The efficiency of the sawdust is undoubtedly due to its blanketing action in floating for a time upon the surface of the liquid and excluding the oxygen of the air. Its efficiency would be greater on viscous liquids than on others The sawdust itself is not very easily ignited, and when it does become so, burns without flame. The character of the sawdust appears to be of little or no importance, nor is the amount of moisture contained in it. The effect of the carbon dioxide from the bicarbonate mixture was noticeable in that the flames occasionally made an effort to shoot sidewise and up from under the blanket of gas. It is probable that the bicarbonate mixture would have shown up to still greater advantage had it not been for the wind which must have blown the gas away."—*Engineering and Mining Journal* (from *Metal Industry*, March 1913), June 7, 1913, p. 1149. (T. D.)

THE USE OF EXPLOSIVES IN AGRICULTURE.—"One of the chief problems in successful agriculture in India is the maintenance of a proper physical condition of the soil. Many of the soils are extremely heavy, being composed of fine clay and silt; such soils are easily puddled and compacted, thus becoming relatively impervious to air and water. Some soils that are of good tilth on the surface have a layer of hard compact soil below, which is impervious to water. No matter how deep the soil may be ploughed or trenched, or how thoroughly it may be pulverised, a layer forms just below the depth the cultivator may reach when ploughing, which is firmly packed into an impervious mass. This condition of compactness becomes more pronounced as time goes on, until the lower layers are too firm for the best root growth, or for the penetration of air and water, long before the plant reaches maturity. Frequently when heavy rains occur shortly after ploughing, the lower soil assumes a puddled compact condition within a very short time; under such conditions neither air nor water can penetrate deeply enough for purposes of successful growth and the plants are injured by exclusion of air through over saturation with water. One of the consequences of the above unfavourable physical condition is that the nitrogen in the soil is not available and this can only be remedied by improved conditions for aeration. In some instances certain good results have followed the use of surface drainage to carry off the surplus water from excessive rain, thus preventing the soil from packing together more firmly than would otherwise be the case, but the lower layers still remain compacted. On account of the foregoing difficulties explosives of the nitro-compound class (such as dynamite and gelignite) have been used at varying depths from the surface to break up the sub-soil, the resulting cracks and fissures being beneficial in improving the drainage of the land. More water can thus be held in the soil by reason of the additional depth of soil spaces and the movement of water becomes more rapid, both downward during the rainy season and upward during the dry season. The importance of the use of explosives for the above purposes cannot therefore be overestimated. It is of considerable economic value, when we consider the vast amount of labour and time required for deeper ploughing or trenching to break up the lower compact layers, which, as above shown, are so detrimental to plant growth. Experience has shown that the immediate effect of the explosion is that a pocket of pulverised earth is formed with numerous cracks and fissures radiating from it in all directions. The soil is affected at an equal radial distance all round, and the hard compact layer of earth is effectually destroyed. We have no evidence as to what length of time these improved conditions last, but it is estimated the effects should be beneficial for several years."—ANON, *Indian and Eastern Engineer*, July, 1913, p. 9. (H. A. W.)

EXPLOSIONS IN AIR COMPRESSORS.—"The increased use of air compressors, liquefying plant, and, to a certain extent, large gas engine units, has brought up the question of a danger which may be present where such machinery is employed. A recent article in *The Times Engineering Supplement*, which deals with this risk with air compressors, is of interest. The cause of such explosions as have taken place is probably connected with the presence of an inflammable gas in the cylinder of the compressor or in some of its accessory portions. Where this is present in quantity sufficient to form an explosive mixture, and when in some way or other this is raised to a sufficient temperature to cause ignition, certain unpleasant results may follow. So far as it is known at present, these inflammable gases may have their origin, either in the oil used to lubricate the cylinder or else carbon monoxide may be formed *in situ* by the oxidation of a deposit of carbon. It is thought that the latter gas is responsible for a large proportion of the explosions which have taken place. Deposits of carbon may form from the lubricating oil, or from dust which enters from the inlet port, or even from both. In the case where it is due to the lubricating oil, a suitable variety has not been chosen, and by careful attention to this point, the risk may be undoubtedly reduced. In the case where it has its origin in dust, the compressor has generally been working in the neighbourhood of a colliery. It is known that such deposits of carbon, however they may be formed, will absorb oxygen when a quantity of air is passed over them, and that the rate of absorption increases with temperature, so that the temperature of the deposit of carbon, once it reaches a certain quantity, or thickness may increase until the ignition point is reached. The ignition temperature of carbon monoxide is about 1,200° F., and this temperature may easily be produced, locally, by the heating effect caused by the sticking of the outlet valve. Explosions of this nature, in addition to those which have occurred under certain conditions from the presence of lubricating oil in excess in large unit gas engines, which, in one case at least entailed disastrous consequences and loss of life, are disconcerting phenomena. They are specially interesting to the chemist, as he alone will probably have the necessary data to trace their origin, and also from the fact that such machinery is entering largely into operations of an electro-chemical or kindred nature. It is also evident that the use of such machinery will tend to increase in the future, and it is, therefore, necessary wherever the chemist meets with the same (especially in large units) that the possibility of such phenomena occurring must receive serious attention, and every means taken to reduce the chance of their occurrence to a minimum"—*Chemical World*, (from *Times Engineering Supplement*), Aug., 1913, p. 242. (T. D.)

Dangers of Coal.—"The authors commence by describing an apparatus and a method by which coals may be tested and classified as regards their liability to self-ignition. Such tests would furnish definite data as to the safety or danger of storing and transporting any particular type of coals. Coals which when heated to 150° C. in a stream of oxygen show no tendency to increase of temperature are absolutely safe; those which show a slight tendency to heat up locally but fail to ignite within an hour are safe enough for storing and transporting on board ship. Those which ignite within an hour and below 150° C. are relatively dangerous coals. The temperature of self-ignition, and the time of heating necessary, determine relatively the degree of danger. In general the heat value of the coal is not an index as to its tendency to self-ignition; brown coals are much more dangerous in this respect than soft coals. The presence of finely-divided particles in a coal pile increase this danger greatly. The coals tested show that the greater the percentage of oxygen in the coal the greater the liability to self-ignition."—Dennstedt and Schafer, *Zeit. Angew. Chem.*, xxv., 2625; *Journal of The Franklin Institute*, Vol. clxxvi., No. 1, July, 1913, p. 94. (J.G.)

Chloride of Lime in Sanitation.—"The research department of the Hooker Electrochemical Company, on the advice of Dr. L. H. Baekeland, President of the American Institute of Chemical Engineers, recently undertook to collect all available data relating to the use of chloride of lime in sanitation, and the amount of material brought together was so large and appeared of such great value that it was decided to publish it in book form. Sanitary officials in this country should be grateful for this decision, more particularly as the book contains short abstracts of a large number of articles on sanitation which have appeared in various journals and which are referred to in the text. These abstracts are arranged under convenient headings and the book is well indexed. For its treatment of the subject of the campaign against the house fly alone it is an important contribution to the literature of sanitation, which should be in the hands of all who are in charge of work connected with the public health."—Albert H. Hooker, *Chemical News*, July 18, 1913, p. 34. (F. W. W.)

The "Pyro" Boiler-cleaning System.—"The difficulty of satisfactorily removing hard scale from the interior of steam boilers has induced the expenditure of much ingenuity and from time to time of large sums of money on the part of investors. These attempts met with more or less success, but notwithstanding that, the old practice of hand-chipping with hammers is still usually adopted. The latest scientific method of removing scale has left the beaten track, and instead of applying scraping and percussion, as is usual, heat in a very intense form is utilised. The apparatus was brought out by Mr. Adolph Schror, and the process has been termed the 'pyro' boiler-cleaning system. The plant required is extremely simple, consisting of an oxyacetylene blow-pipe with two or three jets, a portable acetylene generator, and a steel cylinder of compressed oxygen, together with the necessary length of flexible tubing. When any scale is to be removed, all that is necessary is to play upon it with the blow-pipe flame for a few minutes (time depends on the thickness of scale) until the scale cracks and falls off, due to the rapid expansion under the intense heat. The operation is easier if the scale is slightly damp. In an actual operation it has been found satisfactory. Care has to be taken to prevent the flame from playing on the bare plate for any length of time. When flat pieces of hard scale, some of them $\frac{5}{8}$ in. thich, were held on the bare hand, with a piece of white paper between, and the flame was directed on the scale, this fell to pieces, while the paper was not scorched nor the hand burned. A distinct feature of the system is that it can be easily applied to almost inaccessible parts of a boiler, where hand-chipping is quite impossible. After cleaning off the scale the inside of the boilers is painted with 'corrosanti' paint, which it is claimed preserves the plates."—Anon, *Eng.*, xcv., No. 2,463, p. 377; *Journal of The Franklin Institute*, Vol. clxxvi., No. 2, August, 1913, p. 235. (J.G.)

The Natal Wattle Bark Industry.—"The amount of Natal wattle bark exported in 1912 was 52,700 tons (of 2,240 lb.) valued at £283,000, as compared with 49,400 tons valued at £288,400 in 1911, an increase of 7% in tonnage, and a decrease of 2% in value. At the beginning of the year the price in Durban was about £6 5s. per ton. The price fell as low as £5 5s. about the middle of the year, but afterwards advanced again. It is said by *The Board of Trade Journal* that the prices realised last year do not adequately remunerate the growers, and there has been some movement towards reducing the acreage under wattles. In some cases, year-old trees have been ploughed in, and maize has been sown in their place. In other cases, when plantations have been cut down for stripping, the next crop of young trees has been destroyed. Now that tick fever has been conquered, the tendency is for agriculturists to turn their attention to cattle rearing and dairying. Changes of this description, however, will not reduce the export of bark during the next few years, for existing plantations are not likely to be generally disturbed until their growing crop has matured and been harvested, and by that time conditions may have changed again in favour of the grower."—*The Oil and Colour Trades Journal*, August 16, 1913, p. 576. (J.W.)

GRAPHITE IN MADAGASCAR.—"A report by the Governor-General of Madagascar states that numerous deposits of graphite have been discovered on the island during the past three years, namely, in the districts of Tananarivo, Manjakandriana, Vatomandry, Andovoranto, Maevatanana, Betafa, Antsirabo, Ambositra, Fianarantsoa, Fort Carnot, and Ambalavao. The line of outcrop extends along nearly the whole island from Ambre to Fort Dauphin, and nearly 400 claims have been located. The graphite is disseminated in the form of scales interstratified in gneiss, and seems to replace mica. Close to the graphite outcrops and parallel thereto is an extensive intrusion of white quartz, traversing the beds of gneiss, and apparently having some definite connection with the graphite, the two being associated in all cases. The graphite deposits form masses attaining a thickness of 20-30 yd., the net thickness of graphite being about 1-2 yd., and they consist of weathered gneiss from which the graphite can be easily separated by hand washing, by which means it can be concentrated to 80% or 85%. The scaly crystals of graphite have a greasy feel. The output in Madagascar in 1911 amounted to over 1,500 tons, and this will probably be doubled in the present year. Works are already being constructed to treat the graphite, some of them (Ouberbie) employing electric fans for classifying the product, whilst others (Soc. Franco-Belge, Syndicat Lyonnais, Lonys Frères, and Cabrol & Cie) using water as the classifying agent. Railway communication already exists between Tananarivo and Brickaville, and an extension to Tamatave on the one side and Antsirabe on the other would afford a powerful stimulus to the graphite industry."—*The Oil and Colour Trades Journal*, August 23, 1913, p. 677. (J.W.)

THE SOUTH AFRICAN WHALING INDUSTRY.—"H.M. Trade Commissioner for South Africa reports that the industry was first established in 1908 by a Durban company, and that the large profits obtained attracted others, so that at the end of 1912 there were 24 companies, with a capital of £1,100,000, operating on the coasts of South Africa, half on the east coast and half on the west. Most of those working on the west are Norwegian companies with floating factories, and there are four floating factories on the east coast. The general opinion in the history is that the business is being overdone, and the question of asking for its regulation by the Government has been raised. Few of the East Coast companies made any profit last year, and none paid more than 10% dividend. The kinds of whale captured off the Natal coast include the right whale, the sperm, blue fin, humpback, and rorqual. They migrate northwards from May to July, and southwards from September to November. From November to May only tooth whales are found on the south-east coast. The sperm whales run in schools, and generally keep a hundred miles off the land."—*The Oil and Colour Trades Journal*, August 16, 1913, p. 576. (J.W.)

DUSTLESS BREAKERS.—"Another step in the improvement of the physical conditions surrounding their employes has been taken by the Lehigh Valley Coal Co., Wilkes-Barre, Pa. Suction fans to relieve the air of coal dust have been installed in all of their dusty breakers, and are being installed in breakers not classed as dusty. In the new breakers, such as the one at Mineral Spring, where dust is controlled by dampening the coal, no suction fans are used. The suction fans, which are the result of much experimenting, are from 5 ft. to 14 ft. in diameter, and absorb from 10,000 cub. ft. to 100,000 cub. ft. of dust-laden air per minute, and cause the fresh air to rush in through the windows. Immense galvanised iron tubes reach all parts of the breaker where dust is generated, or where employes are gathered. One intake is at the top of the breaker where the coal is dumped after coming from the mine, another is at or near the rolls where the coal is crushed, etc. The tubes conduct the dust-laden air to the fan which whirls it up into a tower where it is moistened by a spray of water. The rush of air causes the moistened dust to travel past wide sloping shelves, down that tower and through another that is erected alongside for the purpose, until it finally drops to the ground in a quiescent condition. With the 14 ft. fan at the Dorrance colliery, Wilkes-Barre, the tubes are from 2 ft. to 3 ft. in diameter, and the big main tube reaching directly to the fan is at least 7 ft. in diameter. The fan runs at an average rate of 130 revolutions per minute."—*Colliery Engineer*, Aug., 1913, p. 29. (A. McA. J.)

TIMBER PRESERVATION.—"The Powellising process is a method of wood treatment developed in England and consists essentially of impregnating the wood with a solution of saccharine. This is performed by the usual open-vat treatment method. The saccharine has the faculty of hardening the wood considerably, rendering it impervious to attacks by decay and by boring insects. Furthermore, it probably fire-proofs the wood to a certain extent, or at least does not render it more inflammable, as does creosoting. This effect of hardening and fireproofing the timber treated should render the Powellising process especially available for shaft guides."—*Engineering and Mining Journal*, Oct. 4, 1913, p. 642. (A. R.)

Reviews and New Books.

THE CYANIDE PROCESS OF GOLD EXTRACTION. Fifth Edition. By James Park. Cloth, octavo, 350 pages, with many illustrations. London: Charles Griffin & Co. Price 8s. 6d. net.

"Mr. James Park, professor of mining in the University of Otago, New Zealand, is a most skilful expositor of the science and practice of mining and metallurgy. The consequence is that his text-books have enjoyed a wide popularity throughout the world. So small a book naturally does not give the minutiæ of practice sought by the expert, but nevertheless it is wonderful how much detailed information is contained within its pages. The new edition has 350 pages of text, 110 more than the previous edition issued in 1906, and the increase in the number of illustrations is in the same ratio. The advance in slime-treatment accounts for a large part of the additional matter. The author has included descriptions of typical modern installations of plant in various parts of the world, drawn from the best sources. The chapter on analysis of cyanide solutions has been extended. It is of interest to remind readers that Mr. Park experimented with the air-agitation of slime years before F. C. Brown erected what is now called the Pachuca at Komata Reefs, in New Zealand, and with vacuum-filtration long before the Moore, Butters, and Oliver filters were known, so that he watches the development of these methods and processes with a paternal eye."—*Mining Magazine*, Sept. 1913 p. 227. (A. R.)

Avery, Alfred H. Electric Lighting, a Practical Guide to the wiring of Houses and the Installing of Electric Light Plants. Fully illustrated by Line Drawings. 12mo., pp. viii.-150. *Cassell.* Net 2s.

Bloxam, Charles London. Chemistry, Inorganic and Organic; with Experiments. 10th edition. Re-written and revised by A. G. Bloxam and S. Judd Lewis. Roy. 8vo., pp. 890. *Churchill.* Net 21s.

Cochran, J. A Treatise on the Inspection of Concrete Construction. 8vo. *Spon.* Net 18s.

Cordeoro, F. J. B. The Gyroscope. Cr. 8vo. *Spon.* Net 6s 6d.

Cumming, A. C., and Kay, S. A. A Text-book of Quantitative Chemical Analysis. 8vo., pp. 394. *Gurney & Jackson.* Net 7s. 6d.

Dommett, W. E. Motor Car Mechanism. Illustrated. Cr. 8vo., pp. 208. *Scott Greenwood.* Net 3s. 6d.

Earl, T. C. Gold Dredging. Illustrated. 8vo., pp. 224. *Spon.* Net 20s.

Edwards, J. Hall. Carbon Dioxide Snow: Its Therapeutic Uses, Methods of Collection and Application. Cr. 8vo., pp. 94. *Simpkin.* 3s. 6d.

Hall, A. D., and Others. The Soil Solution and the Mineral Constituents of the Soil. (Royal Society). 4to., pp. 22, swd. *Dulau.* 1s. 6d.

Hall, G. L. Elementary Theory of Alternate Current Working. Illustrated. 8vo., pp. 202. *Electrician.* Net 3s. 6d.

Hicks, W. M. A Critical Study of Spectral Series. Part III. The Atomic Weight Term and Its Import in the Constitution of Spectra. (Royal Society). 4to., pp. 98, swd. *Dulau.* Net 4s. 6d.

Kershaw, John B. C. Electro-Thermal Methods of Iron and Steel Production. 8vo., pp. 264. *Constable.* Net 8s. 6d.

Price, G. M. Handbook on Sanitation. 3rd edition. Cr. 8vo. *Chapman & Hall.* Net 6s. 6d.

Snyder, H. The Chemistry of Plant and Animal Life. 3rd edition, revised. Cr. 8vo. *Macmillan.* Net 6s. 6d.

Treiber, E. Foundry Machinery. Cr. 8vo., pp. 148. *Scott Greenwood.* Net 3s. 6d.

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Prout, W. M., *l/o* Pretoria ; Pankop, Pienaars River.
Schlesinger, B., *l/o* Boksburg : Main Reef West G. M. Co., Ltd., P. O. Box 2, Maraisburg.
Thorburn, A., *l/o* Germiston ; Nourse Mines, P. O. Box 32, Denver.

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OF THE
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Vol. XIV. DECEMBER, 1913. No. 6.

Proceedings
AT
Ordinary General Meeting, December 13th, 1913.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, December 13th, 1913, Mr. Alex. Richardson (President) in the chair. There were also present:

21 Members: Prof. G. H. Stanley, Messrs. H. A. White, T. Donaldson, H. Meyer, C. Toombs, F. W. Watson, John Watson, Prof. J. A. Wilkinson, A. F. Crosse, W. R. Dowling, A. McA. Johnston, Dr. J. Moir, (Members of Council), A. S. Allam, W. Beaver, H. C. Boydell, F. D. Phillips, W. S. V. Price, D. F. Scott, G. Hildick Smith, and A. L. Spoor.

12 Associates and Students: Messrs. E. G. Baskett, W. Human, R. W. Irwin, L. T. Leyson, E. J. Moynihan, S. E. Willows-Munro, D. Ordman, H. Rusden, P. Scatterty, B. C. Smith, W. J. Sutherland, and W. Waters.

10 Visitors, and Fred. Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

NEW MEMBERS.

Mr. F. D. Phillips and Mr. A. L. Spoor were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected, as follows:—

Gill, Harold Warren, B.Sc. (Lond.), A.I.C., S. A. School of Mines & Technology, P. O. Box 1176, Johannesburg. Lecturer on Chemistry.

Lee, William, Klipval-Pongola G.M. Co., Ltd. Sulphur Springs, Piet Retief. Mine Manager.

The Secretary announced that since the last meeting of the Society the following Associate had been admitted by the Council:

Lass, W. P., Alaska Treadwell G. M. Co., Ltd., Treadwell, Alaska, U.S.A. Cyanide Superintendent.

GENERAL BUSINESS.

The Secretary announced that the second competition for the shield presented by this Society for mine surface workers would be held on the following morning at the Crown Mines. The Red Cross Society, and members of the Society would regret that for that shield only six teams had entered, and of those two had scratched, leaving the following four to compete for the shield: the Crown Mines, Knight Central, Luipaardsvlei Estate, and De Beers.

For the Underground Workers' Shield, presented by the Society, ten teams had entered, and the competitions would be held at the Meyer and Charlton Mine to-morrow week. It would be a good way underground, and he doubted whether there would be very much room for sightseers, but to-morrow the Red Cross Society would welcome all members of this Society who cared to be present.

EVENING CLASS EXAMINATION RESULTS.

The Secretary: I have pleasure in notifying that, as a result of the Evening Class Examinations just concluded at the S.A. School of Mines and Technology, the following students have won the prizes offered for competition by the Society:

Inorganic Chemistry— Stage I. ...	No award
Inorganic Chemistry— Stage II. ...	T. L. Thorne
Practical Inorganic Chemistry Stage I. ...	E. G. N. Bradley
Practical Inorganic Chemistry Stage II. ...	B. C. Smith
For the prize of £5 and the Society's bronze medal— Stage III. ... ... (who has obtained a 1st class in both the theoretical and practical examinations, gaining over 75% in each)	E. Rosenberg
Metallurgy—Stage I. ...	T. L. Thorne
Metallurgy—Stage II. ...	L. H. G. William

Metallurgy of Gold ...	B. C. Smith
Assaying I.	A. S. Allam T. K. Prentice
Assaying II.	E. Rosenberg

ATTEMPTED ASSASSINATION OF SIR LIONEL PHILLIPS.

The President: I am sure you will all heartily join with me in expressing the greatest abhorrence at the dastardly outrage of which Sir Lionel Phillips, one of our Honorary Vice-Presidents, has been the victim. On behalf of the Society I should like to convey our deepest sympathy to Sir Lionel and Lady Phillips, and to wish him a speedy and complete recovery.

The proposal was agreed to by acclamation.

SEYMOUR MEMORIAL LIBRARY.

Mr. Andrew F. Crosse (*Past-President*): I am not satisfied with the way in which the Seymour Library is going. It is not run as one would wish. We want to refer to the newest books on technical subjects. During the last two months I had occasion two or three times to go to the library and found that the editions available were ten or twelve years old. Yesterday I wanted a book on acetylene, and the newest book they had was published fourteen years ago.

This Society is a subscriber. I do not know what is being done, and I wish the members who are on the Committee would give a statement about the library, so that we may know its position. A technical library is of no earthly use unless we have the newest books. I want seriously to bring this matter before you. Something should be done to support the industries of the country; and in one of the biggest mining centres of the whole world, we ought to have a library where we can obtain books containing the information we require.

I am sure you will all agree we do not want student's books; they can get their own books; we want books of reference on mining and practical chemistry. If this country is going to advance in any branch of science we want the newest information, whether it be from America, England, Germany, France, or anywhere else to help us along; we cannot all afford to go and acquire the information in Europe.

Mr. H. A. White (*Vice-President*): I am in hearty sympathy with some of Mr. Crosse's remarks. There is no doubt we have not the money to get all the books we would like to get for the library. I may say that Mr. Crosse has only recommended, to my knowledge, one book in the past three years, and that book is on order. If he will take the trouble to recommend to the Book Committee any books which he has found himself to be suitable for that library I have no doubt whatever they will be purchased as far as our funds will permit. At the present moment we have quite a fair sum of money in hand which we are about to expend on books, and a large order for books has just gone home. If Mr. Crosse has anything special in his mind I would ask him to make his recommendations before the next meeting of the Committee, on Wednesday.

Mr. A. F. Crosse (*Past-President*): What does the Government give to the library?

Mr. H. A. White: £150.

Mr. A. F. Crosse: I want the Government to put in your hand plenty of money, so that you can spend it.

Mr. H. A. White: I may say that at recent meetings of the Council, which unfortunately have not had the advantage of Mr. Crosse's presence, some of these things have been threshed out.

The President: I take this opportunity to congratulate the management of the Seymour Memorial Library on the good work which it has done. This fine collection of technical books is a credit to the town; and it deserves to be made use of a great deal more than it is.

ANNUAL DINNER.

Mr. John Watson (*Member of Council*) proposed a vote of thanks to the Dinner Committee for the excellent manner in which the arrangements for that function had been carried. This was agreed to.

THE SOLUBILITY OF IODINE IN SODIUM IODIDE SOLUTION.

By Harold W. Gill, B.Sc.(Lond.), A.I.C. (Member).

A short time ago I pointed out* that the waste solutions obtained in iodometric analysis could be readily worked up and the whole of the iodine present recovered as sodium iodide, if the standard solutions of iodine were prepared by dissolving the iodine in sodium iodide solution in place of potassium iodide as is more usual. In many laboratories where much iodine is used the saving effected is well worth considering. As no data were apparently available I therefore undertook experiments in connection with the solubility of iodine in solutions of sodium iodide. The sample of sodium iodide used was manufactured by Messrs. May & Baker, of London, and was sold as B.P.: it was free from potassium and also from sodium iodate. The iodine used was Merck's pure.

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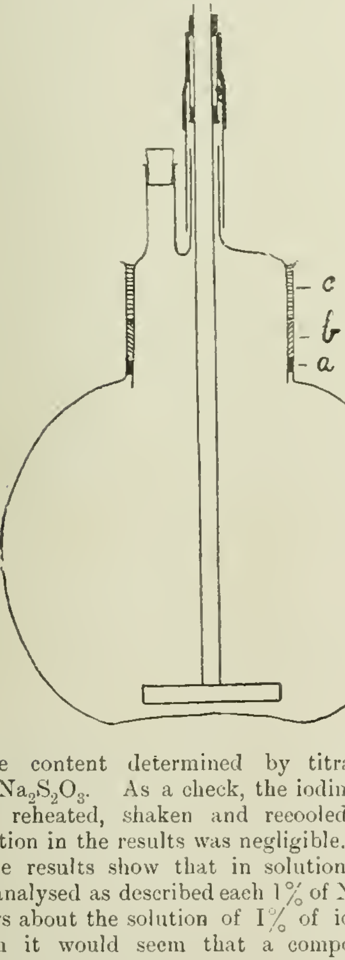
* *Analyst*, 1913, Vol. 38, p. 409.

Table I. shows the variation in the solubility of iodine in sodium iodide solution with the concentration of the sodium iodide. Solutions containing 5, 10, 15, and 20 per cent. respectively of sodium iodide were prepared and a large excess of iodine added. The whole was then warmed to about 60° C and well shaken for about 5 minutes. The flasks were then allowed to cool down for 4 hours at 25° C in a thermostat which was continuously stirred by an electrically driven stirrer. Quantities of about 10 gm. of solution were then weighed out and their iodine content determined by titration with $N/2$ $Na_2S_2O_3$. As a check, the iodine solutions were reheated, shaken and recooled but the variation in the results was negligible.

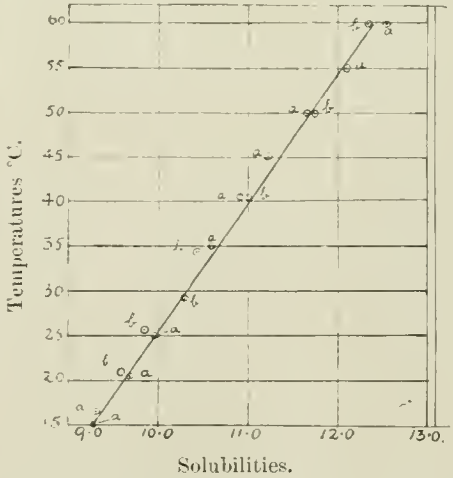


The results show that in solutions prepared and analysed as described each 1% of NaI present brings about the solution of 1% of iodine from which it would seem that a compound $NaI_2 \cdot 2H_2O$ exists in solution.

Tables II. and III. give the specific gravities of solutions of sodium iodide and of iodine in sodium iodide respectively for various concentrations at 25° C. The measurements were carried out using a 25 cc. specific gravity bottle.

Tables IV. (*a* and *b*) show the variation of the solubility of iodine in sodium iodide solution with temperature. The results only show fair agreement this being due (*a*) to the difficulty of obtaining strictly saturated solutions and (*b*) to the volatility of the iodine.

In a trial experiment it was found essential to keep both thermostat and the solution to be experimented upon efficiently stirred. It was also obvious that neither rubber nor cork stoppers would long withstand the action of the warm iodine vapour. The experiments were therefore carried out in the apparatus indicated in the sketch. The flask (500 cc.) was closed with a stopper made by sealing off a piece of wide glass tubing and fusing into it two glass tubes one in the centre for the stirrer the other at the side to enable solutions to be introduced or withdrawn as required. The stopper was fixed in place by means of a rubber band "a", then a layer of asbestos saturated with paraffin "b" and above this a layer of sealing wax "c". The apparatus having been arranged as described was filled with solution and a large excess of iodine added. The whole was placed in a thermostat a few degrees above the required temperature and stirred for about two hours. The thermostat was then cooled until the desired temperature was attained and stirring continued for at least six hours. The solution was then allowed to settle and about 10 cc. pipetted out and quickly transferred to a stoppered flask, weighed and then titrated with N. $Na_2S_2O_3$ solution. Stirring was then continued for another hour and another reading obtained. The curve given herewith represents the solubility of iodine in 100 gm. of 10% sodium iodide solution (*a*) and the solubility of iodine in 50 gm. of 20% sodium iodide (*b*). (See Table IV.) It will be observed that the



curves are coincident within the limit of experimental error.

Table I.—*Variation in the solubility of iodine in sodium iodide solution with concentration of the sodium iodide.*

100 gm. of x% sodium iodide solution dissolve y gm. of iodine at 25° C.

x	y	$\frac{y}{x}$
5	4·99	·998
10	9·96	·996
15	14·93	·995
20	20·02	1·001

This table shows that the amount of iodine dissolved is directly proportional to the amount of sodium iodide present and that a 1·5% solution of anhydrous iodide is capable of taking up sufficient iodine to prepare a 0·1 N. solution.

Table II.—*Specific gravities of solutions of sodium iodide at 25° C compared with water at the temperature.*

Concentration of NaI. Per cent.	Specific Gravity.
5	1·0369
10	1·0720
15	1·1072
20	1·1458

Table III.—*Specific gravities of solutions of iodine in sodium iodide at 25° C. compared with water at the same temperature.*

Concentration of NaI. Per cent.	Specific Gravity.
5	1·06985
10	1·1415
15	1·2162
20	1·2998

Table IV. (a and b).—*Variation in the solubility of iodine in sodium iodide solution with temperature.*

100 gm. of solution at t° C dissolve x gm. of iodine.

IVa.—Concentration of NaI 10%.

t	x	t	x
15	9·27	35	10·61
16·5	9·30	40·5	10·88
20·6	9·67	45	11·245
25	9·97	50	11·67
30	10·365	55	12·10
		60	12·57

IVb.—Concentration of NaI 20%.

t	x	t	x
21	19·24	40·5	22·01
25·75	19·69	50·0	23·55
29·75	20 63	60·0	24·70
34·5	20·86		

Summary.

I. The amount of iodine dissolved (at all temperatures) by a solution of sodium iodide is directly proportional to concentration of the sodium iodide.

II. A solution containing 1·5% of NaI will dissolve sufficient iodine to prepare a 0·1 N. solution.

III. It is probable that the compound NaI_2 $2H_2O$ exists in iodine-sodium iodide solutions.

The author expresses his thanks to Professor J. A. Wilkinson for facilitating the carrying out of the above experiments.

Mr. Andrew F. Crosse *(Past-President)*: This is a very interesting paper. There is a book that many of us refer to, "Comey's Dictionary of Chemical Solubilities." I think if our Society would communicate with other Societies and start the ball rolling it is about time that a new book was brought out on that question. It is not the sort of book which it would pay a man to write, but it is the sort of book Societies should take in hand and bring out. It is 21 years old and I think it is high time we had something new.

Professor J. A. Wilkinson *(Member of Council)*: In reply to the last speaker I beg leave to inform him that a book was published not many years ago on this subject, which may be called a modern supplement to Comey. This book by Dr. Seidell can be found in the Seymour Library. Also the tables of constants published by the International Committee can be obtained there, should Mr. Crosse require the most up to date work. At the same time I agree that it would be an excellent thing if the handy book by Comey could be revised and re edited.

Mr. Andrew F. Crosse *(Past-President)*: Professor Wilkinson will agree with me it is a very important question of almost every day work and I am glad to receive this information, because I think many of us have not heard of the new work.

Dr. J. Moir (*Past-President*): I hope the Secretary will insert these remarks in the *Journal* and send a copy to Dr. Comey, who is still quite alive; he may be induced perhaps to bring out a further edition.

THE DETERMINATION OF GOLD IN THE PRESENCE OF IRIDIUM AND ALLIED METALS IN MATERIALS SUCH AS BLACK SAND.

Prize Papers by Mr. Jas. Gray, F.I.C., and Mr. Chris. Toombs (Members of Council).

(*Published in July Journal, 1913.*)

REPLY TO DISCUSSION.

Mr. Jas. Gray, F.I.C. (*Member of Council*): The discussion which was conspicuous by its absence warrants no reply from me and I merely desire to correct a remark of the President's at-the September meeting in announcing that the papers were open to discussion. He said (this *Journal*, 1913, 119) *inter alia* "the Council has received several requests from members, including Messrs. Toombs and Gray, to throw these papers open to discussion." To dispel any false impression which this remark may have made, I would state that when the matter was raised in Council I said that I had *no objection* to the paper being discussed—which is quite a different story.

Mr. C. Toombs (*Member of Council*): There being no discussion to reply to, I will touch briefly on a few of the points in my paper which to me seem the most important. Only the bare experimental results of some of these were given to the Adjudicators, it being understood that they were *au fait* with the finer points of gold assaying, and I had hoped that some discussion either constructive or destructive would take place on them.

It is an open secret I believe, that the Iridium Competition arose through a shortage of gold recovered from black sand bought on supposed gold assay value. Now, one of the points dealt with in my paper is that of the "parting" operation, and I have shown that when dealing with an accurately weighed piece of fine gold one can have a plus or minus according to how the various operations are carried out after simple cupellation. The ordinary method of parting on the Rand consists of detaching the bead from the cupel and flattening with two or three strokes of the hammer, then dropping into routine parting acid without putting through rolls or annealing. To my mind, here is one cause of the shortage experienced by the smelting works, and I venture to assert that working with 1 A.T. of material going 15 to 20 oz. gold per ton the retained silver will amount to close on 5 dwt. if the parting is done in routine assay office manner; and it is essential for great accuracy whatever process is used that a check piece of fine gold should be treated in exactly the same manner.

Another point of interest was the failure to recover the total fine gold after simply cupelling, annealing, and giving two nitric acid treatments. I have had the opportunity to give this point a little further attention and I find that the loss is largely due to cupel absorption. I have examined several different makes of cupel, as well as those made under my own supervision, and I have not yet found a cupel which does not absorb gold. The absorption varies inversely with the amount of silver present—the greater the ratio of silver to gold the smaller the absorption, silver being absorbed freely, and consequently the gold absorption is only a question of dilution. The ordinary method for testing cupels for absorption is to finely crush one or two which have been used for cupelling some important sample, such as "screens," fusing the lot with the necessary fluxes, cupelling, etc. Now the absorption is so small and magnesium oxide being rather refractory in fusing the result generally shows "nil" or the merest trace.

My method is, I believe, original and may be of interest—I hold the theory that absorption if any is only on the surface of the cupel to test this theory I scraped off $\frac{1}{16}$ in. from the surface of about 18 cupels, which had been used for screen samples, and fluxed with plenty of borax, litharge, and barren slag. I then took off the next $\frac{1}{16}$ in. and gave same treatment, also a third $\frac{1}{16}$. The result proved my theory to be correct, a large quantity of silver and an easily weighable quantity of gold being obtained from the first layer of the cupel and practically nothing from the second and third layers. The weight of gold divided by the number of assay tons of sample originally fused gives the absorption per assay ton fused. I also found that the absorption varied with the value of the sample, *i.e.*, a lot of cupels used for 6 dwt. material gave a lower absorption figure than those used for 9 dwt. material. I further tested cupels to determine if the absorption was largely around the finishing point, but found that it took place all over the surface of the cupel.

Another point of interest in my paper is the failure of methods of evaporation in lead dish with or without scorification, even when the gold was fixed by previous precipitation. This is a method highly spoken of in text books, also cropping up now and then as an infallible method in original papers; indeed I must plead guilty myself to having used it as such, and spoken of it as such in a paper I read before this Society about seven years ago.

A further point of interest is the very complete manner in which cuprous chloride precipitates gold from solution, and I have found that it gives complete precipitation, practically instantaneously, from a diluted *aqua regia* solution of gold and I would now use this reagent in preference to ferrous sulphate for that purpose.

Coming to iridium the most interesting point is the effect that roasting produces in giving much higher values for that metal than if assayed without roasting, and from this I am of the opinion that iridium is not present as the metal or as osmiridium but probably associated intimately with arsenic and nickel. Iridium would also seem to have a greater affinity for iron than for lead, the iron nails used in fusing taking up 70% to 80% of the metal present, although when iron in the shape of filings was used as a flux, iridium was found in almost full proportions.

That this affinity exists I have proved by placing iron nails in the fused mass for a few minutes before pouring, and obtained low iridium results.

My investigation was carried out with a view to a suitable method for assaying the gold and iridium contents of ordinary sale samples of black sand going from 10 to 20 oz. gold, and iridium from 3 to 4 dwt. to 2 oz. per ton; and such samples should have no terrors to an assayer having a passing acquaintance with elementary chemical manipulations, such as filtering and incineration of filter papers. Recently I have handled a few uncommon black sand samples in the shape of concentrate from black sand before and after cyaniding. One sample contained 10 oz. approx. iridium or platinum metals per ton with only about 7 oz. gold, and another, iridium 10 oz., gold about 1 oz. per ton. The method in vogue is to add a large excess of silver as a collector for the iridium or platinum group metals. My experience with this method was that these metals were in such a fine state of division after parting as to be in almost a colloidal state, and no filter paper would hold them and complete settlement for washing by decantation was absolutely impossible. Highly satisfactory results were obtained as follows:—

A weighed excess of fine gold, varying with the amount of gold in the sample under examination together with the necessary 2½ times silver for parting, was added when fluxing, and the routine carried out as described in my paper. The fine gold acts as the collector of the platinum metals, and with care in parting these can be retained on the surface of the gold, a check piece being essential. The gold and iridium are separated by solution of the former in dilute *aqua regia*, the iridium being left in an apparently granular form easily filtered, or in fact washed by decantation, though the former is preferable. I have found this method gives excellent resnlts and can recommend its use to anyone handling similiar material.

If I may make a few remarks in conclusion, I would say that the iridium competition from our Society's view has been extremely disappointing and not in keeping with the splendid record of the past. The prize certainly was meagre, this probably kept the best busy public professional men out of the field, but surely it does not account for the unexpected silence of the numerous assayers employed on the Rand, and the quotation from Huxley used some little time ago by Mr. J. Watson is very applicable here, "the worker who takes all and gives nothing is but a drone in the hive." Personally I am pleased I undertook the investigation—I am not much richer in worldly goods, but I have now that which money cannot buy, a knowledge and understanding of some of the finer points of gold assaying which I did not possess before.

My thanks are due to our Past-President, Mr. E. H. Johnson for many useful hints and for his kindly encouragement, and to Mr. R. A. Cooper for his assistance during the investigation, also to the General Manager of the East Rand Proprietary Mines, Mr. W. T. Anderson, for permission to carry it out.

Mr. Andrew Crosse *(Past-President)*: I wrote the Secretary some weeks ago saying I would rather not discuss these papers for various reasons but I would read a short original paper of my own at the next meeting.

THE DETERMINATION OF THE ACIDITY OR ALKALINITY OF WATERS—A STUDY IN INDICATORS.

(*Read at September Meeting, 1913.*)

By Jas. Moir, M.A., D.Sc. (Past-President)

DISCUSSION.

Mr. John Watson, F.I.C. (*Member of Council*): Our Society is once more indebted to Dr. Moir for an interesting paper, on a subject which is of importance chiefly to the chemical, yet also, indirectly, to the metallurgical and mining members. The author holds the record, I believe, of having contributed more papers to this Society than any other member—long may they continue. Since the vote of thanks and preliminary discussion, there has been no contribution on this paper. Perhaps it is because Dr. Moir is known to live up to the motto of his native land—*Nemo me impune lacessit.*

It is important, as he says, that the right indicator, or class of indicator should be used.

In Alkalimetry the chemist makes up his standard acid (hydrochloric, nitric or sulphuric), and checks it by testing a weighed quantity of pure dried sodium carbonate, dissolved in sufficient distilled water. Of course, the author knows this as well as I do, and I merely mention it to show the different behaviour in this simple operation of three of our best known indicators, methyl orange, phenolphthalein and litmus. Pure dry sodium carbonate contains 58·5% soda (Na_2O) or *available alkali*. When the acid is correctly standardised a drop of *methyl orange* used as indicator shows after agitation in the cold, or very slightly warm, the above result on titration, the colour changing from lemon yellow to salmon.

With *phenolphthalein* as indicator, titrating cold, the pink colour fades away to colourless with just half the quantity of standard acid, indicating 29·25% soda, half of the soda being converted into chloride, nitrate or sulphate: the other half remaining as sodium bicarbonate ($NaHCO_3$), which is neutral or colourless to phenolphthalein in the cold. Litmus, properly prepared, will indicate 58·5% soda in pure sodium carbonate; but the titration must be carried out at the temperature of *boiling* water to get rid of carbon dioxide. I mention these facts for the benefit of such learners, students, shiftsmen. etc., who may not understand much about indicators or alkalimetry.

The First Table.—In this the author gives the behaviour of 29 different indicators with, (*a*) an acid mine water from the New Goch, (*b*) with Johannesburg tap water as supplied to Plein Street Laboratory. He tells us that the mine water really contains about 65 to 70 parts of free sulphuric acid per 100,000. These figures mean that the indicators requiring 6·6 cc. to 7·1 cc. are the correct ones for the purpose.

The Mining Engineer.—I may be old-fashioned in my methods; but if a mine manager or resident engineer had sent me such a water to test for acidity I should use methyl orange as indicator. This, according to the author's test, would show the said water to contain only 24·50 parts H_2SO_4 per 100,000. The mine officials mentioned probably want to know to what extent such water would corrode the exposed parts of pumps or wire ropes, and I take it that methyl orange would indicate such acidity. In referring to phenolphthalein (p. 115, 2nd col.), the author has somewhat altered his paper by saying that this indicator is also the proper kind for use in calculating the amount of soda required to prevent scale from boiler water. There are at least five or six different ways of making such a titration, and I am not sure that phenolphthalein for this purpose is superior to methyl orange.

The subject of water purification is a large and important one. In this connection I should like to draw the author's attention to a paper:—"Some Recent Methods of Technical Water Analysis," by Prof. H. R. Procter (*Jl. Soc. Chem. Ind.*, 1904, p. 11). In that paper Prof. Procter discusses the methods of Hehner for temporary or bicarbonate hardness with methyl orange; he also discusses the methods of Pfeifer and Wartha (*Zeits. angew. Chem*, 1902, 198) for temporary and permanent hardness, who use alizarine as indicator, and in the case of permanent hardness use a mixture of equal parts of *N*/10 sodium carbonate and sodium hydroxide (hydrate) solutions. About 1903 we started on the City and Suburban mine a water purifier made by Breda, of Berlin. At first I made the daily tests with soap solution, then I adopted a *N*/10 soda solution (mixed) as above, after reading Prof. Procter's paper. I used a drop or two of methyl-orange as indicator, not alizarine. Of course, the object of adding NaOH is to get better precipitation of the magnesia salts. Daily tests were made of the raw and purified water, and occasionally analyses for lime, magnesia and sulphuric acid.

The Cyanide Manager looks at the water reaction from the protective alkali point of view, as the author points out. I make up from time to time a solution of *N*/10 sodium hydrate for the cyanide department, where it is used for testing mill circuit water and this solution I always standardise with phenolphthalein as indicator.

Crown Mines Water, Shaft No. 3.—In the case of this water, the author says the true acidity is about 115 parts per 100,000. 100 cc. of such water would require 11·73 cc. *N*/5 NaOH solution. Will the author kindly tell us in his reply how he arrives at the above figure (115 parts). also how he gets the 65 to 70 parts of free sulphuric acid per 100,000 in the New Goch mine water? I find his figure 12·5 cc. with nitro-benzeneazo-guaiacol gives 122·5 parts H_2SO_4 in the Crown Mine water, against 115 parts per 100,000, said to be the true acidity. Out of the 29 indicators given by the author in his first table there is, apparently, not one to be exactly relied upon. Under such conditions, Hamlet might have said: "To be honest, as these dyes go, is to be one indicator in a thousand."

Alkalinity of Potable Waters.—The late Mr. John Pattinson, F.I.C., for many years public analyst of Newcastle-on-Tyne (and other northern counties) once said at a meeting of the Society of Chemical Industry, that he had generally found the surface moorland waters of Northumberland to be alkaline. I cannot say what indicator he used. According to Thresh, authorities differ on this point, some recommending lacmoid and others phenolphthalein. Prof. Mohr and Dr. Wanklyn recommended a specially prepared solution, or tincture of cochineal (carminic acid) as indicator in testing the reaction of water.

Litmus and its preparation.—Litmus gives the range required according to the author's figures for the New Goch Mine water. Had the author prepared his litmus as recommended by Lunge, ("Sulphuric Acid and Alkali," 1st Edn, Vol. I., p. 45); Sutton, ("Volumetric Analysis," 8th Edn., pp. 33 and 34), or Fresenius ("Chemical Analysis Quantitative," 7th Edn., pp. 197 and 198), he would probably have got a much sharper end reaction. The last stages of the titration should be carried out after boiling to get rid of carbonic acid.

I quite agree with the author that no hard and fast rule can be laid down as to the proper indicator to use, until one knows, definitely, for what purpose the information, as to acidity or alkalinity, is required.

Indicators in Alkalimetry.—For some 4½ years, I was chemist in the caustic soda department of the Allhusen works, which then became, and I believe still are, the largest high strength caustic soda works in the world. We occasionally had complaints that our 77% caustic only tested 76% and so on. On investigating, I found I remember in one case that the buyer's chemist had used phenolphthalein as indicator, whereas the custom of the British alkali trade is to calculate all the soda existing as carbonate as available. The indicator used is, therefore, methyl orange. With phenolphthalein, at ordinary temperatures, only half the soda existing as carbonate would be indicated.

Effect of Alumina with Indicators.—In Nov., 1890, Prof. Lunge contributed a paper to the Society of Chemical Industry (see p. 1018 of their journal), wherein he dealt with the estimation of available soda in commercial caustic soda. He referred to a contention by Messrs Cross & Bevan (well known specialists on the chemistry of cellulose, etc.), that, on account of the presence of alumina, methyl orange should not be used as indicator. The writer made 21 analyses of Newcastle Chemical Company's caustic soda, and 18 analyses of other makes, which averaged:

	Newcastle Chemical Company's Per cent.	Other Makes, 17 Samples Per cent.
Sodium aluminate ... ($Na_2Al_2O_4$)	0·07	0·17
= Alumina (Al_2O_3) ...	0·04	0·10

In one exceptional case I found a Lancashire brand to contain aluminate equal to 1·73 Al_2O_3%. This I omitted from the average as being exceptionally high. Messrs. Cross and Bevan recommended that commercial caustic soda should be tested with phenolphthalein as indicator, that the solution should be boiled, cooled and back-titrated – a much more complicated method than the ordinary.

The writer read a paper on this subject to the Newcastle section (*Journal of the Society of Chemical Industry*, 1890, p. 1107). Some years before I had tried experiments on caustic "bottoms," *i.e.*, the chocolate coloured residue, amounting to perhaps 6 to 9 cwt. from each 12 ton pot of high strength caustic. In testing these "bottoms," we found litmus to be the best indicator; thus, at a time when Dr. Lunge had written of litmus as "the beatified indicator," we still found it to be the best in the field for this special purpose.

Caustic bottoms contain from 4% to 10% $Na_2Al_2O_4$ and the insoluble in water varies from 6% to 14%. Our litmus was carefully prepared and neutralised as recommended by Lunge and Sutton. I see that Dr. Lunge in the latest edition of "Sulphuric Acid and Alkali" (Vol. II., Pt. I., p. 115) quotes my paper and confirms my contention that methyl orange could be employed for ordinary caustic soda without hesitation.

Strength of Indicators.—Having used methyl orange since 1880, I prefer to keep to a very weak solution or tincture of it. I gm. dissolved in methylated spirit can be made up to 2 litres (2,000 cc.) and 1 or 2 drops of this is sufficient for most purposes.

Phenolphthalein we used to prepare of the same strength; now it is made considerably stronger, viz., 1 gm. per 100 cc. and this may be necessary for say cyanide work. The quality of methylated spirit we get here is much inferior to what we had in the North of England. We used to dissolve these indicators in methylated spirit and dilute with water 2 to 1; here I find I must use ⅔ spirit to ⅓ water, or the indicator is re-precipitated.

H_2CO_3 as an Acid.—I could say a good deal on this point, in support of Dr. Moir's work on the weaker class of indicators. The greater part of my first 24 years were spent on the magnesian limestone (Permian formation) of Durham county. There one sees abundant evidence of the solvent action of carbonic acid in water on the dolomite. Sunderland water is specially referred to by Dr. Wanklyn as very extraordinary on account of its hardness.

Re sand-filling, once a cyanide is decomposed by CO_2, we must remember that hydrocyanic acid is extremely volatile. It has a greater volatility than ethyl or sulphuric ether.

This contribution has run to a greater length than I intended. If I have spoken chiefly of three of the oldest and best known indicators of the present generation, I quite realise that the next generation of chemists may, possibly, do just as good, or better, work with 3 or 4 of those less-known indicators whose names and constitution are given on Dr. Moir's valuable table.

THE UNION PATENTS BILL.

A. L. Spoor and W. E. John (Members).

Whilst matters of law are not, as a rule, of very engrossing interest to technical men, it is believed that the subject matter of the present paper is one of paramount importance both to the members of this Society and to the mining industry. We have before us at present a law in its plastic condition, one upon which the fortune and future welfare of any member may sooner or later be vitally dependent. It surely, therefore, behoves us as members of the largest scientific society in this sub-continent to do our utmost to have the new Patents Act for the Union passed with the fewest possible blemishes.

The following discussion of the principal points of interest in the proposed Act may perhaps be made somewhat clearer if the general nature of patents is first briefly considered.

A patent is a document granted by the Government and purporting to give the owner of a patentable invention the exclusive benefit of his invention for a limited period. The patent right so granted is in effect the Government's part of a bargain between the Government and the inventor, in which the latter makes his invention known to the public in return for receiving such right. It is a purely artificial creation and as such has its limitations and drawbacks; but on the whole, there would appear to be no better form of return which could be given to the inventor. The inventor's reward is automatically adjusted since it depends primarily upon the value of the invention to the public and the skill with which the invention is exploited.

The Bill which is the subject of this paper has been drafted by a Government Commission consisting of Lord de Villiers, Sir James Rose-Innes, and Mr. W. P. Schreiner, K.C. with Capt. E. Christian as Secretary. Whether it represents what the Government will ultimately present to Parliament and what amount of revision it will receive in Parliament are matters for conjecture. The Bill is substantially modelled on the present Transvaal Patents Proclamations, which for the last eleven years have, on the whole, been found to work well. Some features, including transitory provisions to cover the change from Provincial to Union administration, have been borrowed from Australia; and certain innovations, some useful and others the contrary, have been taken from the present British Act. It is of course proposed to have one Patent Law for the whole of the Union, and to grant a single patent covering the whole territory in place of the four patents now necessary for that purpose.

Patent Office.—There is to be a central Patent Office, the most likely situation for which is Pretoria, and provision is also made for sub-offices in the other Provinces at which applications can be lodged. The benefit expected from the sub-offices is not very apparent; they have not been found necessary in such extensive countries as the United States and Australia.*

Provincial Patents. — Existing Provincial patents will remain in force (unless surrendered); their periods and renewal fees will be unaltered, but in some other respects they will be subjected to the new Act. They may be extended to cover the Union or such portion of the same in which the invention is new. This extension is to be made by the grant of a new patent coexistent with the original Provincial patent. As however it is said that the Crown (from which the grant ultimately proceeds) cannot make two grants from the same thing, an interesting question arises as to the validity of this procedure.*

Commissioner of Patents.—A Commissioner of Patents is to be appointed, and it is recommended that he should be a man of high scientific and legal attainments. This recommendation will, it is thought, be heartily endorsed by all interested in patent matters.†

Definition of Invention.—Coming to the Bill itself, attention is first directed to the definition of invention (presumably "Patentable Invention") viz.:—

"'Invention' means any new and useful art, process, machine, manufacture or composition of matter, or any new and useful improvement thereof, capable of being used or applied in trade or industry, and not known or used by others in the Union, or in any foreign country (save as provided by sections 50, 51 and 52 respectively of this Act) and not patented or described in any printed publication in the Union or any foreign country before the application for a patent in respect of the same."‡

To define what should be patentable is admittedly no easy matter, but it is submitted that the above definition falls short of solving the difficulty.

"New and useful art, process, machine, manufacture or composition of matter, or any new and useful improvement thereof, capable of being used or applied in trade or industry" is sound. Its categorical enumeration of the six broad classes leaves little doubt as to the technical subject matter which falls within the scope of patent protection; while it rightly excludes any thing having no industrial applicability.

In "known or used by others" the word "others" has no context to contrast with. Presumably it means others than the actual inventor or his nominee. Moreover the phrase is indefinite particularly as to time. In the corresponding United States definition the words "before his invention or discovery" are added.

The addition of the words "any foreign country" however is thought to be open to serious criticism. It involves placing Union Patentees in a position of uncertainty probably unequalled in the world. Very many years ago the United States statute contained similar words, but their obvious injustice and tendency to discourage invention soon resulted in their elimination. The British Law again takes no account whatever of knowledge or use outside the Realm. The ultimate object of the Union Patent Law is to increase the sum of industrial knowledge available to the people of the Union, and it is in no way concerned with the personal knowledge or practices of people in other countries. As the draft now stands a Union patent could be defeated by knowledge or use of the invention among the head hunters of Borneo. It might be supposed at the first glance that the distinction between knowledge by others in the Union and elsewhere is merely a territorial one. It is in fact very much more. Prior knowledge or use by others in the Union very properly stops the patent because otherwise the patent would stop such others from exercising their pre-existing right. The same reasoning does not apply to persons abroad because the patent cannot touch them. Of the excepting sections mentioned, Section 50 is the usual one permitting an invention to be published at or during international or industrial exhibitions without prejudicing the patent subsequently applied for. It is believed to have no practical value whatever whilst provisional protection can be obtained. Section 51 is new and provides that "prior publication shall not invalidate the patent if the patentee proves that the publication was derived from him and that as soon as he learnt of it he protected his invention with due diligence."

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*Section 16.

*Sections 6, 7, 8, 9.

†Section 13.

‡Introductory 4.

Sections 52 and 84 are to the effect that a patent or description more than 50 years old shall not itself debar a patent for the same invention. It has been suggested that these are dangerous clauses because they permit re-patenting of all sorts of things which have previously been patented in this country or elsewhere for more than 50 years, and so prevent the public from using such old inventions. It is thought that the danger is not a real one. A publication that is more than 50 years old is for all practical purposes dead and buried. If it is the only source of knowledge of the invention, the latter is lost to the public. On the other hand an old invention of any value would be known apart from its original publication, for instance by current literature or public knowledge or practical use, and would thereby be rendered unpatentable to anyone.

Another excepting section, mention of which is omitted, is Section 61. The first part of this relates to fradulent applications. The second part waives the general prohibition regarding an invention already patented elsewhere in case the application was made here by the same applicant within 12 months of the date of the grant of the prior application. Publication during that period also is not fatal. "Date of grant" is ambiguous; it may mean date of application for, or date of issue of the patent, the difference between which is always some months and may be years.

One of the peculiarities of a society like this is the unrivalled opportunity it gives speakers to inflict their pet fancies on a temporarily defenceless audience. We accordingly put forward the following definition for the criticism it no doubt deserves, with however the secret consolation that after all the last word will be ours.

"Invention means any new and useful art, machine, process, manufacture or composition of matter or any new and useful improvement thereof capable of being applied in trade or industry.

"An invention shall not (save as provided by the necessary excepting sections) be considered as new if at the date of the application for a patent in the Union in respect thereof,

"*(a)* it is publicly used or manufactured by others in the Union; or,

"*(b)* it is a matter of common knowledge in the Union or abroad; or,

"*(c)* it is described in a printed publication in the Union or abroad in such a manner as to make it reasonably available to the public in the Union; or,

"*(d)* it has been patented in the Union or abroad within 50 years prior to the application;

"Provided that use of the invention by others or its publication in print without the knowledge and consent of the inventor shall not prejudice the application for a patent or provisional protection or letters patent obtained thereon; if such use or publication was derived from the inventor, and if in the event of his learning of such use or publication he thereafter made such application with reasonable diligence."

Applicant.—According to the present Transvaal law the actual inventor must apply for the patent, either alone or conjointly with others It is proposed in future to allow the assignee or agent or attorney of the inventor also to obtain the patent. At the present time in the Cape and Natal an apparently valid patent may be obtained by a resident who has received knowledge of the invention from another country, whether from the actual inventor or not. This somewhat curious practice is a relic of the period when British inventors did not exist and all new inventions were brought into England by travellers from Flanders and other foreign countries This was done often "at peril of their lives" as the old writers put it and accordingly merited reward; but at the present day the practice cannot logically be supported and will not continue under the Union Act.*

Restricting Invalid Patents.—The Commissioners report that they carefully considered what procedure is most advisable to prevent or check the issue of invalid patents. Whilst they consider that as an abstract question the best course is that of an official examination for novelty and patentability as is done in Great Britain, United States, Germany and some other countries, they are satisfied that local conditions render such a course out of the question. They accordingly recommend that full opportunity should be given for anyone to oppose the grant before issue of the patent.

The following is the armoury of weapons with which the Act provides an intending opponent, and against which the inventor should be forearmed, viz.:

(*a*) Invention obtained from opponent.

(*b*) Invention obtained by fraud

(*c*) Prior patent in the Union.

(*d*) Complete specification covers wider invenvention than the provisional specification, the additional matter having meanwhile been protected by the opponent.

(*e*) Invention not novel or not capable of being protected.

(*f*) Alleged true and first inventor not such.

(*g*) Invention contrary to law public order or morals.

(*h*) Invention described is purely hypothetical.

(*i*) Complete specification insufficient.

(*j*) Title of the invention does not truly indicate the subject matter of the invention.†

This is a formidable enough list which should be capable of preventing the issue of any but the soundest patents. Unfortunately its purgative virtues are carried to the point of vice. For instance, in the last item the very existence of the patent is made dependent upon the accuracy of what is nothing more than a label, the sole purpose of which is roughly to identify the nature of the invention and to allow it to be properly classified. Its origin points to the existence of atavism in law, because in Great Britain in the very early history of patents the grant was made on the title only, and consequently a person who had in fact invented a spade was very properly pulled up from obtaining a patent for an agricultural instrument. We are, however, now long past that time, and the Examiner of Patents must,‡ and does, see that the title is correct before he passes on an application for publication.

Item (*e*) is also thought to be objectionable. Surely what is meant is that the invention does not come within the statutory definition of a patentable invention.

It is only the barest justice that an opponent should be compelled to particularize his objections so that an applicant can know exactly what he has to meet and so make up his mind whether to proceed or not. The Bill, however, contains no such provision; nor does it provide for proper security being found by the opposing parties before entering into litigation.

Claims.—As is probably fairly generally known, a patentee is required to put on record a claim or claims precisely defining the monopoly which he seeks. Hitherto the law in this country on this point appears to have been that one invalid claim vitiated the whole patent. This is now altered by Section 53, which makes the claims independent as regards their validity. Had this been the law in 1897 the inventors of the cyanide process would perhaps have received an adequate return for what they have done for the industry. The same care and skill in drafting claims will still be necessary as hitherto since invalid claims may be severely penalised.*

It is proposed that opposition proceedings shall be before the Commissioner of Patents who, as mentioned above, is to be both a scientist and lawyer. The Commissioner is to have practically the same status as a judge, and appeals go direct from him to the Appellate Division of the Supreme Court.†

Specifications.—One of the conditions of obtaining a patent is the filing of a full specification. Instead, however, of doing this at once an applicant may lodge a provisional specification with the object of obtaining provisional protection for any period not exceeding 9 months before he proceeds to the full patent. The differences which in practice occur between these two documents relating ostensibly to the same invention are dealt with somewhat fully in the Bill.

Thus, if the complete specification goes substantially beyond the provisional, the Commissioner may refuse to accept the former, or with the applicant's consent, may cancel the provisional specification and regard the complete as constituting a fresh application. This of course involves possible loss of priority. Again the application may be allowed to proceed as far as the invention originally specified is concerned, whilst the additional matter is treated as forming a fresh application.‡

It is further provided that disconformity between the specifications is not a ground for

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*Sect. 34.
†Sect 49.
‡Sect. 37.

*Section 83.
†Sections 54, 18, 19.
‡Section 42.

invalidating a patent provided the additional matter was novel at the date of the complete specification and was invented by the applicant or one for whom he acts.* This provision is difficult to reconcile with clause (d) of the grounds of opposition.

Conversely, a single complete specification may cover the subject matter of two or more separate provisional specifications so long as the invention described in the latter are cognate or modifications of one another. The patent granted in this case bears the date of the earliest provisional application, but in considering its validity and for the purposes of opposition (*e.g.*, under clause (d)), the different claims of the complete specification may have to be correlated in date with the different provisional specifications—often a sufficiently unenviable task one would think.†

Patents of Addition.—Patents of addition may be obtained for improvements or modifications of an already patented invention. The only difference between them and main patents appears to be that no renewal fees are payable on them and that they expire with the main patent, which is generally a drawback from the patentees point of view.‡

Renewal Fees.—Renewal fees will be required to keep the patent in force, but their value is not stated.** The present Transvaal fees are very moderate, viz., £2 increasing annually to a maximum of £7. These have been found ample and it is to be hoped that under Union they will not be exceeded. We all know that the tendency of governments is to seize upon all possible sources of revenue but it is wrong to look upon patents and patentees as direct sources. The chief utility of renewal fees is to weed out at an early stage patents which are of no value, but they should not be regarded as taxes on invention. The short memoried and unmethodical are looked after since overdue renewal fees may be paid with a fine; and even if a patent has actually lapsed by nonpayment of said fees it may be restored, subject to proper safe guards to protect any person who may have begun to use the invention.††

Extension.—Those to whom Dame Fortune has been tardy may be glad to know that in the case of a particularly important invention, for which the patentee has been inadequately remunerated, the Court may order the patent to be extended for a further period not exceeding 7, or in exceptional cases, 14 years. ‡‡

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*Section 59.
†Section 60.
‡Section 71.
**Section 58.
††Section 72.
‡‡Section 70.

Surrender.—Although a patent is generally something of an asset, it may, under some circumstances, prove to be an inconvenient possession which one is glad to be rid of. Hitherto there has been no means of putting an end to its existence before the renewal fees fall due, but now a patentee may apply to surrender his patent at any time provided he is not engaged in litigation, and subject to the rights of others.*

Patent Agents.—The portion of the Bill relating to Patent Agents† is perhaps of more interest to the writers than to other members; but probably everyone is liable to invent, just as they are to fall ill or make a will, in all of which contingencies qualified assistance is at least desirable. Even when we have a tooth out we insist, through the legislature, on seeing the dentist's diploma. If we take this care in ridding ourselves of an encumbrance, should not the channels by which we seek to secure the results of industry and thought be equally well guarded? To resume the analogy for a moment, we demand the whole tooth and nothing but the tooth, and equally is it necessary for the inventor to cover all that he has invented, but nothing else, if he is to be fully and safely protected.

Now, and for the past 11 years, patent agents practising in the Transvaal have required to be enrolled and (except those here before the war) have been admitted only after passing a somewhat stringent examination covering South African and foreign law and practice and technical subjects.

The Bill proposes to apply similar conditions to the whole of the Union but first to admit any person who has been practising as a patent agent in any part of the Union for 6 months prior to the commencement of the Act, and also any one who has been an "officer" in any Provincial Office. It is not very clear what "officer" means, but it does not appear to exclude an office boy who might have been employed 20 years ago and might now be a farm labourer on the other side of the world. Comment is surely needless. There can be no objection to solicitors acting as patent agents but it is submitted to be wrong to allow them, as is proposed, to call themselves patent agents without specially qualifying and without registration.

Compulsory Working.—A feature of the Bill, on which there will probably be widely divergent opinions is that dealing with compulsory working of inventions. The relevant sections are:—

"77. (1) Any person interested may, after the expiration of two years from the granting of the patent present a petition to the Commissioner alleging that the reasonable requirements of the

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*Section 76.
†Section 89.

public with respect to a patented invention have not been satisfied and praying for the grant of a compulsory licence or in the alternative for the revocation of the patent. . . .

"Provided that no order of revocation shall be made before the expiration of three years from the date of the patent or if the patentee gives satisfactory reasons for his default.

"78. For the purposes of the last preceding section, the reasonable requirements of the public shall not be deemed to have been satisfied

"(*a*) If by reason of the default of the patentee to manufacture to an adequate extent, and supply on reasonable terms, the patented article, or any parts thereof which are necessary for its efficient working, or to carry on the patented process to an adequate extent, or to grant licences on reasonable terms, any existing trade or industry in the Union is unfairly prejudiced, or the demand for the patented article, or the article, or the article produced by the patented process is not reasonably met ; or,

"(*b*) if any trade or industry in the Union is unfairly prejudiced by the conditions attached by the patentee, before or after the passing of this Act, to the purchase, hire or use of the patented article, or to the using or working of the patented process."

The compulsory licence is old and not very unreasonable, but has never to our knowledge been made use of. The novel feature is the alternative of revocation and the detailed explanation of what constitutes failure to satisfy the reasonable requirements of the public.

It is necessary to recall that a validly patentable invention is something quite new to the public, something which they have previously never had either the right or the opportunity to use, and which, had it never been invented, they would be no worse off without. One might think, therefore, that the public can have no reasonable requirements in regard to the invention, that no existing trade or industry can be prejudiced, unfairly or otherwise, by not getting the use of the invention, and that as the result these sections are largely nugatory. However, proceedings which have taken place under the present British Patent Act (from which these sections are taken) show that this is not the case and that the law requires a patentee to exploit his invention or take the chance of losing his patent. Why this should be is not very clear. Surely an inventor does enough in return for his limited monopoly when he fully and publicly discloses the invention so that it may be used by anyone after his patent expires. Now he is to be required not only to disclose the invention, but also apparently to exploit it.

Whilst we are fully alive to the desirability of fostering new industries in the country, we believe that this section as a means to that end is too drastic. There are many inventions which, of course, cannot be worked in this country, and the fewness of industrial establishments and lack of competition for new things introduces the danger of an inventor being frozen out of his invention. The result to be feared from this section is the discouragement of inventors and loss of inventions to the country.

Contracts.—A rather curious new feature is one restricting the patentee's right to make contracts. He cannot licence the use of his patented invention conditionally upon the licensor binding himself not to use material supplied by third parties or to buy unpatented goods only from the licensor. Is not this class legislation against patentees? Licences are not to extend beyond the term of the patent. The existence of a contract such as the above, which is illegal according to the Act, is available as defence in an action for infringement of the patent and apparently whether the infringer is a party to the contract or not.*

Damages—Marking.—A patentee's remedy for the unauthorized use of his invention is an action for infringement, in which he may claim an interdict and also damages for the loss the infringement has caused him. In the future the recovery of damages will be contingent upon the infringer having had reasonable notice that a patent exists. Patented articles will accordingly require to be marked with the number and date of the patent.† On the other hand, false representation that an article is patented will meet with severe punishment.‡

Infringement, Revocation.—In defending an action for infringement the alleged infringer can attack the validity of the patent on practically every possible ground. He can in future have the patent revoked,** although formerly a patent declared invalid in an action for infringement could remain in force unless separate proceedings for revocation were taken.

Threats.—The provision regarding unfounded threats of legal proceedings remains the same, that is, if any one unjustifiably threatens another with liability for infringement without bringing the matter to a head, or as has been said, "holds the sword of Damocles over him," the one threatened can obtain an interdict against the continuation of the threats and recover any damage he has suffered.††

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*Section 80.
†Section 81.
‡Section 101.
**Section 81.
††Section 85.

Government.—The Union Government can acquire a patented invention upon compensating the patentee. Patents for instruments or munitions of war may be assigned to the Minister of Defence, and if the latter so directs are made secret patents and not published as are others.*

Joint Patentees.—As regards joint patentees, the draft Act permits each to use the invention for his own profit without accounting to the other, but not to grant licences independently. When one of them dies his interest passes into his estate, a point which has not been clear hitherto.†

Ships.—The fact of our country being now a maritime one is brought home by the exemption from liability from infringement of patented apparatus used in the working of ships in South African waters. It has apparently been overlooked that the country is not insular and that vehicles from other countries should logically be on the same footing as ships.‡

Designs.—An entirely new departure as far as the Transvaal is concerned is the provision for registration of designs.**

A design differs from a patentable invention in that function is an essential feature of the latter whilst the former depends for its distinctiveness entirely upon its appearance. Thus, novel forms of tools, implements, fittings, etc., may be protected by designs registration although they are unpatentable. In some cases double protection is obtained by simultaneous patent grant and design registration.

Among miscellaneous matters the Commissioners recommend a speedy classification and indexing of all patented inventions and the issue of a Patent Journal, both of which will, it is thought, meet with general approval. The establishment of an adequate library is probably a no less urgent matter.

The Bill on the whole seems to be a good one. With the object of making it as perfect as possible it may be suggested that this Society should use its influence to eliminate or modify some of the defects which have been referred to. In our view the ideal is to ensure that as far as possible every patent which is issued shall be reasonably valid and to grant it without conditions or restrictions except perhaps the payment of nominal renewal fees.

Mr. Andrew Crosse *(Past-President)*: I beg to propose a vote of thanks for this paper. Dr. Caldecott ought to have been here. I think he has had more to do with patents than most of us. There are numbers of legal points raised which I fear I am not competent to question, but I do hope that before long we shall be able to take out a patent for the whole Union and not be obliged to take one for each Province separately. It is high time this matter was arranged.

Mr. Thos. Donaldson (*Member of Council*): I beg to second the vote of thanks to the authors. When we consider the number of trades and professions necessary to the mining industry and the consequent scope which the industry, in our midst, provides for the development of new ideas, it can hardly be disputed that the authors have performed a public service in bringing this paper on the "Union Patents Bill" forward.

Before saying anything about the paper itself, I would like to express concurrence with the authors' suggestion that we, as a Society, make representations in the proper quarter with a view to having any blemishes in the Draft Act modified or removed.

The authors express doubt as to the utility of Provincial Offices which it is proposed to establish. It occurs to me that these offices might be made useful to the public by providing in each one a proper index of old patents and copies of all documents necessary for the making of a complete search. It is to be remembered that the Commission responsible for this Act recommend that no examination as to novelty be undertaken by the Patent Office, so that the onus of making such a search falls on the intending patentee, and if no facilities for this are available in the different provinces, it is conceivable that the process might prove very expensive to any one living a considerable distance from the Central Office. True, no such facilities are provided in certain other big countries, for example, America, but then the American Patent Office does undertake examination as to novelty.

I agree with the authors that the definition of what constitutes an invention is far too exacting and ought to be modified to bring it more into line with the existing Patent Laws of older countries. I like the author's definition much better and would like to see it adopted. It appears to me also that the authors are right in their objection to the provision that inaccuracy of title may form grounds of opposition, especially when in terms of Section No. 37 of the Draft Act the title will have received the sanction of the Commissioner of Patents before the contents of the patent have become public knowledge. After all the wording of a title in such a way as to

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*Sections 86-7-8.
†Section 94.
‡Section 100.
**Chapter III.

convey properly the real nature of the subject matter, is not always easy, and often leaves room for different opinions.

I find myself again in agreement with the authors in their objections to the regulations dealing with the qualifications of persons entitled to act as Patent Agents. Many of us have had experience of the "man with an idea." In most instances he is not really responsible for his actions, and if he gets into the hands of a Patent Agent whose keenness for business is more marked than his knowledge of the law, it is likely that he will find his "idea" to be an unprofitable one. In the matter of taking out a patent, honest and competent advice is very necessary.

On the question of working of patents and granting of licences, the authors state that there is room for much difference of opinion, and that is undoubtedly the case. Personally I think the provisions of the Act, in this respect, are in the main sound. It is all very well to argue that had the patentee never made his invention known the public would have been none the wiser and no worse off, but having once made his ideas known and obtained protection for it in the form of a patent, the individual concerned takes upon himself a responsibility, the importance of which varies directly according to the value of his patent.

Had he kept his idea entirely to himself he would have made it possible for some other individual who may also have conceived the same idea to exploit in his own and the public interest.

Let us imagine that an invention of sufficient importance to revolutionise a certain industry has been protected, say, in Great Britain and also in South Africa, and that no provisions for working the patent or for granting compulsory licences have been made in the Patent Laws of the latter country. The Patentee might, for reasons of his own, exploit the invention in Great Britain while depriving South Africa of the benefits of it altogether. It may be argued that if an idea can be profitably worked in a certain country the Patentee would not be fool enough to neglect doing so, but, on the other hand, if such is the case he does not suffer much from a certain amount of compulsion, and under Section No. 77 of the Draft Act he is afforded protection from gross injustice.

The President: The authors have presented for our consideration a clear and comprehensive review of the new Patents Bill; and inventors would do well to study carefully the many important points raised.

THE SAND-FILLING OF MINES.

(Read at September Meeting, 1913.)

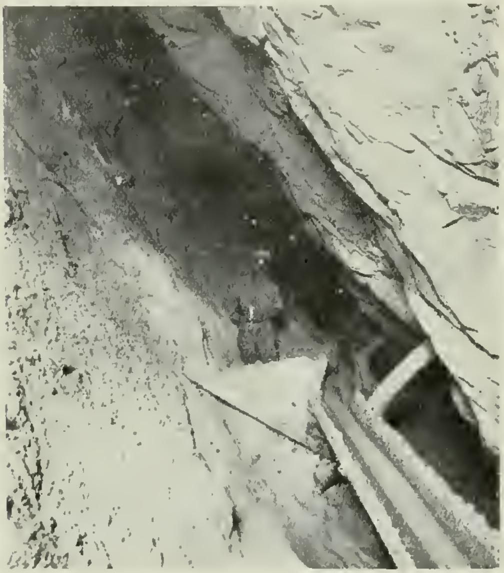
By W. A. Caldecott (Past-President) and
O. P. Powell (Member).

DISCUSSION.

Mr. A. McArthur Johnston *(Past-President)*: I would like to take this opportunity of thanking Mr. White for his remarks about the methods employed in testing solutions draining from the sand filled stopes and published in this paper. I would, however, draw his attention to two points, first, that the method is not mine, and second, that the test is an exceedingly delicate one, and can be used only for very weak cyanide solutions. If a relatively strong cyanide solution had to be tested, this test would not generally be adopted, or if so, then only a small amount of the original solution would be taken.

Again, Mr. White would, in many instances, find that there is a considerable amount of ferrous sulphate present. This, however, does not to my knowledge interfere with the test. The sodium hydrate and the ammonium sulphide are, practically speaking, added at the same time, and the selective action of the latter is such that the possibility of the formation of ferrocyanide is remote. Our tests have so far not shown the presence of ferrocyanide, but should Mr. White at any time detect such we will be pleased if he could communicate that knowledge to us.

Mr. R. E. Sawyer *(Visitor)*: The sand-filling on the Angelo Section E.R.P.M. appears to be a somewhat different proposition from that described, in so much as the work, while primarily designed to prevent caving on a large scale, is at present mainly useful for reclaiming. The sand is lowered from two plants, in one case down an old shaft, in the other through a 300 ft. borehole 8 in. dia. into the upper levels. From here it is run through the old and partially caved-in workings, through launders and on the foot-wall down to the 18th level, a vertical depth of 2,630 ft. The distributing launders, where possible, are laid with an average dip of 15°, 12° being the minimum. They are made of two 9 in. × $1\frac{1}{2}$ in. planks nailed together underground into a V-shape. A single plank can be handled in places underground where it would be difficult to deal with a complete launder. The V-shape is preferable to the flat-bottomed launder for, being nearer the theoretically correct shape for a launder (hyperbola), the sand will flow in it on a flatter grade than in a rectangular launder and it is not so liable to choke as the smallest stream of water will entirely clear it. The wear on a launder dipping 15° is found to be inappreciable except where there is a curve or a sudden drop. At these points old belting is used as a liner. One thousand tons per day of 10 hr. is easily carried by these launders. As regards barriers, jute matting, at half the price of cocoanut matting is used throughout, except in important places where a large quantity of water has to be drained off rapidly through a small area. The jute is laid on lagging poles 3 in.—4 in. dia., old 1 in. pipes, etc., which are nailed on to light upright poles. These poles need not be heavy, 6 in.—8 in. dia. being quite sufficient for an 8 ft. stick. In places where the dip is fairly flat an ordinary strong 12 ft. lagging has been used as an upright successfully, thus showing that the pressure on barriers is practically nil. See Fig. 1. With the high-class sand produced by the Caldecott process, these barriers are efficient and drain perfectly without any other contrivance, such as drainage launders, being provided. A rise can be put through this sand by keeping to



Sand running in V-shaped launders through stope.

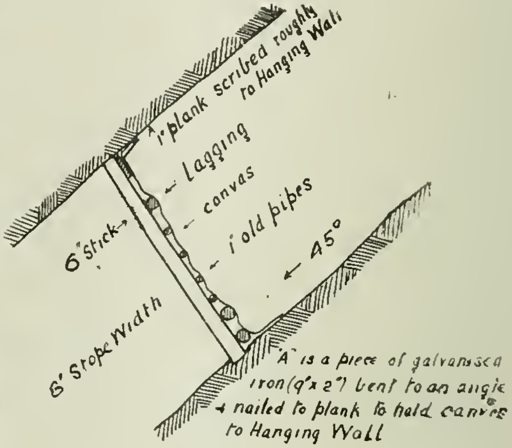

1" plank scribed roughly to Hanging Wall
Lagging
canvas
1 old pipes
45°
6" Stick
6' Stope Width
"A" is a piece of galvanised iron (9"x 2") bent to an angle + nailed to plank to hold canvas to Hanging Wall

Fig. 1.

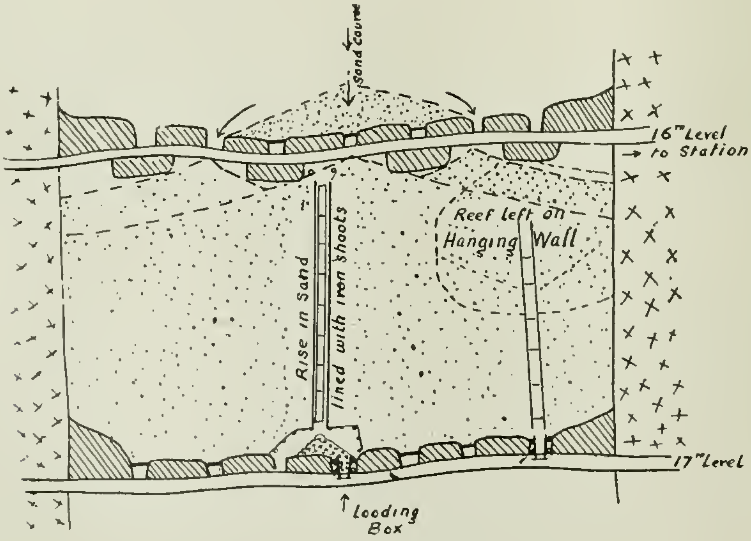


Fig. II.

the hanging wall, within two days of filling. A good boy will rise 12 ft. per hr. in the sand.

Cost.—The cost per ton of sand lowered when the plant is running full strength and when there is no excessive capital expenditure should not exceed 4d.

Below is given a table for an average month. October 1913:—

White wages ...	£169
Coloured wages ...	46
Col. labour sundries	28
Timber and stores ...	112
Workshops ...	13
Transport	1
Water	42
Power	12
Total	£423 for 19,490 tons. = 5·2d. per ton.

When the sand is being run into a new barrier, bales of bedding, or coarse veldt grass, are kept ready and when a leak is noticed a handful of grass is put in. Very large holes can be cheaply and quickly stopped in this manner.

An illustration of some of the methods of reclaiming may be of interest.

1. Take the case of a row of pillars above and below a drive, and a patch of reef left on the hanging wall in a very wide stope. See Fig. No. 2. The bottom boxholes of the stope are closed and sand run in until a cone is formed with apex just below the level. The boxhole immediately above is then closed and the sand run down through the boxholes East and West. This process is continued until there is a series of cones below boxholes and all the boxholes above the level are closed and the stope above filled with sand. The rock reclaimed may be either trammed out on a track laid on the sand or shot down a rise to the level below. The floor of the rise is lined with ordinary iron chutes hung from the top. The hanging wall is got by rising to it and undercutting in the sand sufficiently to enable the reef to be blasted down as required. No timber is used in the rises. Light timber is

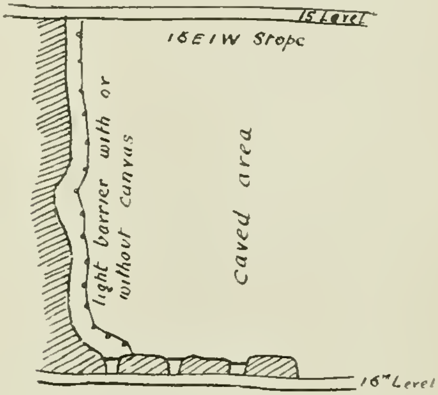
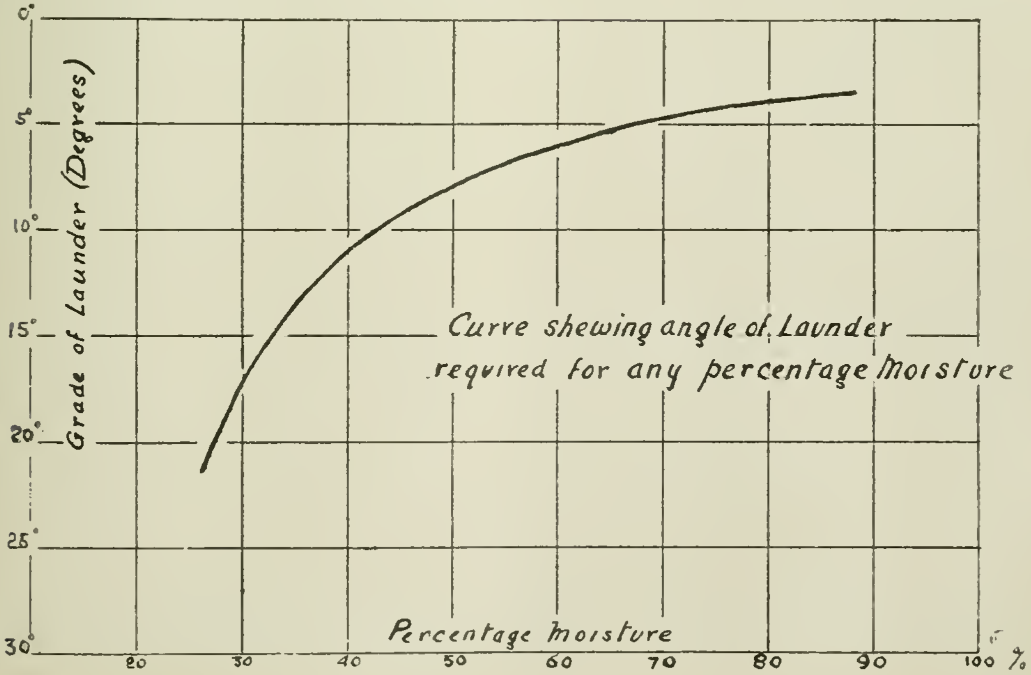


Fig. III.



used to keep the sand off the track as the pillars are taken out.

2. The next case is that of a stope which shows signs of caving and has become too dangerous to work. See Fig. No. 3. The stope is swept down and a light line of sticks and lagging is run within 4 ft. of the face. The boxholes are then closed and sand run in. When the stope is full, the sand between the face and the timber is then dug out from below and stoping resumed. When possible, of course, jute matting is laid on the timber and the stope face kept free of sand, but sometimes owing to bad hanging and foot this is not possible.

A curve showing percentage moisture of sand mixture and the grade of launder in which it will flow safely, is attached.

MINING COPPER ORES AT MESSINA.

(Read at August Meeting, 1913.)

By J. Allan Woodburn, M.Inst.M.M. (Associate).

DISCUSSION.

Mr. J. H. Veasey *(Member)*: This paper being up this evening for final discussion, I would state my deep regret that such a valuable paper has not received more attention.

It is sometimes said that the value of a paper can be gauged from the amount of discussion it produces. This is *not* true in the case before us. The apparent lack of appreciation may be due to the fact of the paper dealing with a phase of mining foreign to Rand methods or that many members by lack of experience of the particular kind are shy to come forward. With twelve years experience of prospecting, exploring ancient workings and opening up new mines, I have not found one point for contention, and am writing this to join my thanks with the Society's for a most instructive paper, which can be highly recommended for study to our younger mining members. We know not what the future has in store nor when our labours may be transferred from the famous Rand whose reef seems "never to be lost for ever" to a new prospect where one's constant thought after each blast is: "Will the reef *still* be there."

While the paper teems with sound practical knowledge, the section on development, August *Journal* p. 57, and the method of stoping the soft lode when over 20 ft. in width, merit particular attention.

VENTILATION OF THE MINES OF THE RAND: THE PROBLEM OF OBTAINING HEALTHIER CONDITIONS.

(Read at September Meeting, 1913.)

G. H. Blenkinsop, M.Inst.M.M. (Associate).

DISCUSSION.

Mr. E. J. Moynihan *(Associate)*: In the solution of all problems the first necessity is to state the facts correctly. The author's main object is to lower the temperature in the working places of the mines of the Rand. In order to achieve this object, it is first of all necessary to know the rate at which the rock temperature increases with depth. It is generally assumed here that this is known, and that the rate of temperature increase is an exceptionally low one. It is an easy matter to prove that it is neither known nor exceptionally low, in spite of the general assumption to the contrary, which has even found its way into the Encyclopædia Britannica Year Book.

The rate of increase is generally taken to be approximately 1° for every 254 ft., on the basis of a paper read before this Society by Messrs. Whitehouse and Wotherspoon in May, 1911. The Government Mining Engineers' report for the current year contains a large number of actual mine temperatures, which prove that the accepted figure is very far from the truth.

The temperature of the undisturbed rock within a few feet of the surface at any particular place, is necessarily and actually, within a degree or so, the mean temperature of that place. The mean temperature of Johannesburg is, closely, 60° F. For purposes of calculation, that may therefore be taken as the temperature of the rock at the surface. The temperature of the mine air of the Rand at any great depth, is in every case less than the actual original temperature of the rock at that depth. If that were not so, air entering the mine at a normal temperature of 60° would come out of the mine colder than it went in—which is not the case. The temperature gradient calculated on the basis of the mine air temperature, is therefore always less than the true original temperature gradient of the rock. How much less depends on the conditions of the mines.

The records published by the Mines Department show that the rate of increase of temperature of mine workings is greater than 1° for every 100 ft. in about a dozen of the mines of the Rand. In two cases the rate is greater than 1° in 70, and on the New Modderfontein the rate is 1° in 61 ft. It is therefore clear that the original rate of increase of rock temperature in many of the mines of the Rand must have been very much more than 1° in 100 ft., and it is probable that in some of them it was nearly 1° for every 50 ft.

The following are some of the details referred to:—

Mine.	Depth for Observation.	Temperature at that depth.	Rate of Increase.	
	Ft.	Deg.	Deg.	Ft.
Cons. Langlaagte	1,960	80	1 for	98
Glencairn	996	70	1 ,,	99
Langl. Block "B"	1,750	80	1 ,,	88
Luipaardsvlei Est	1,675	80	1 ,,	84
New Kleinfontein	1,800	80	1 ,,	90
New Modderfont'	1,820	90	1 ,,	61
New Primrose ...	980	71	1 ,,	90
New Rietfontein	1,160	73	1 ,,	89
New Unified ...	1,260	75	1 ,,	84
Robinson	1,545	76	1 ,,	97
Rose Deep ...	1,540	83	1 ,,	68
Village Main ...	100	71	—	—
,, ,,	2,000	75	1 ,,	134
Village Deep ...	1,924	60	—	—
,, ,,	4,245	70	1 ,,	424

In only one case, apparently, is the rate as low as that given in Messrs. Whitehouse and Wotherspoon's paper; and that is in Mr. Whitehouse's own mine—the Village Deep. The rate there is phenomenally low, according to the figures furnished in the Mines Department Report. It is 1° for 424 ft. There must be something wrong with the figures. They give the temperature of the highest and lowest workings as 60° and 70° respectively. In the first place the lowest temperature in the Village Main—the upcast for the Village Deep—is given as 1° higher than the highest temperature in the deeper mine below it. In the second place Messrs. Whitehouse and Wotherspoon in their original paper give plenty of workings temperatures much higher than 70°. In the third place the air temperature at 1,924 ft. in depth is given as equal to the mean temperature at the surface. In the fourth place, if Messrs. Whitehouse and Wotherspoon's temperature gradient, and the Mines Department figures are both correct, then the rock in that mine has the somewhat remarkable and exceptional power of withdrawing heat from air that is colder than the rock itself.

The whole question of the determination of the temperature gradient in rock—and especially in mines—is in a very unsatisfactory state. I wish to say distinctly that I do not question the good faith of any of the observations, but I think that certain most important considerations have been overlooked. I hope to deal with them in a short paper for this Society at an early date.

The fact that experiments have shown the uselessness of the suggested means of ventilation and cooling by Roots blowers with high pressure air has already been pointed out. The large consumption of power necessary has been also shown by former speakers. But one of the reasons for the failure has yet to be made clear. In compressing air in a Roots Blower or any other similar machine, you virtually have adiabatic compression. That raises the temperature of the air several degrees before it starts going down the mine. Once in the air pipe it can never become absolutely as cool as the mine air by radiation or conduction. Also, and especially as it is not free to take up moisture in the pipe, it tends to become still hotter as it goes down the mine, and its absolute pressure due to depth increases. Since it only does frictional work in the pipe its temperature is not sensibly lowered by expansion or "wire-drawing" in overcoming the friction of the pipe, nor on escaping at the end of its journey. There is nothing like the refrigerating effect of the rock drill, which as it does an appreciable amount of work, releases its air at a temperature much lower than that at which it receives it. A few such machines will reduce temperature several degrees over a fairly large area in an hour or so, as every miner knows.

The author is aiming at the right target but he cannot hit the mark with this particular gun. The gun shoots backwards.

EXPLOSIVES IN HAND AND MACHINE LABOUR STOPES.

(Read at November Meeting, 1913.)

By T. D. Delprat (Member).

DISCUSSION.

Mr. J. H. Veasey *(Member)*: While agreeing that theoretically the better place for the detonator is the bottom of the hole, I would like to ask the author for more information on this novel practice, and if it has the approval of the Inspector of Mines.

To obtain the highest practical effect from a given charge of explosive, it is generally considered important that there should be no air spaces between the explosive and the sides of the hole. One of the advantages of plastic explosives in the safety with which they are "squeezed" into the bottom of a hole and take its shape, which squeezing is done quite safely before inserting the primer. With the detonator at the bottom of the hole two risks present themselves, viz.:—

1. A nervous man will not sufficiently press the cartridges home.

2. A careless man might tamp too hard and cause an explosion.

I submit that more information on this new method would be welcomed.

ELECTRIC BLASTING.

(Read at October Meeting, 1913.)

By Wm. Cullen, (Past-President), Thos. Donaldson (Member of Council), and W. Waters (Associate).

DISCUSSION.

Mr. E. M. Weston *(Member)*: I have listened to Mr. Moynihan's criticism on my contribution to the discussion on this paper with some amusement. I am afraid the feelings of the authors must have been somewhat mixed and that they often felt inclined to pray to be delivered from such friends. If my effort was so inane as he imagines, he had no right to rob them of the glory of annihilating me in their reply. Mr. Moynihan does not credit my statement that the amount of CO and CO_2 given off by fuse is negligible compared with that produced from other sources. What are the facts? Let us omit CO_2 from breathing, lights, etc. Let us suppose that in one round I case of gelatine, is exploded and 85 ft. of fuse is used to fire the holes. According to Dr. Moir

CO_2 from 50 lb. gelatine = 262·5 cu. ft.
CO_2 from 85 ft. fuse = 1·06 cu. ft.
CO from 50 lb. gelatine = 50 cu. ft.
CO from 85 ft. fuse = 0·78 cu. ft.

Comment is needless—CO_2 fuse to gelatine 0·4% ! ! CO fuse to gelatine 1·6% ! !

Mr. Moynihan waxes indignant when I humbly ask for information as to how and why smoke keeps dust suspended. He talks about condensation of vapours round dust, electrical phenomena, and other things which have nothing to do with the matter except to make one expect that vapour might condense upon smoke particles and drag down dust in settling.

I wrote "I do not endorse the author's suggestion that electric blasting should be made compulsory for shaft and winze sinking. The authors should have obtained the opinion of a few practical men on this subject and their opinions would carry more weight."

Mr. Moynihan writes: According to Mr. Weston the authors and the Meyer & Charlton people should have taken the advice of the "practical" Rand underground men and then they would never have done that wicked thing (*i.e.*, start the system at the Meyer and Charlton). So much for his fairness in misquoting, and hence the pointless diatribe on "practical" men and their failings. I venture to say that the progress of the world has been retarded far more by the men who put theories before facts than by the class to which Mr. Moynihan can be easily proved not to belong.

Mr. Moynihan next finds "Mr. Weston contradicted himself and was apparently unable to see that his reasoning was self-contradictory." Speaking of development work, I pointed out that as long as men had to fire their cut before their round and go back through smoke the claim that electric blasting was a perfect system was absurd; but I fully recognised with regard to firing rounds and stope holes when the men had to ascend upcast shafts that electric blasting was of value. Hence I wrote "There would doubtless always be found isolated areas where electric blasting would render any long travel through smoke areas unnecessary."

He gives another example showing this curious inability to understand a simple argument. I argued that if electrical blasting saved any extra time for drilling it was not because electricity exploded the holes; but because a special blasting gang was employed, and that if a blasting gang were employed with fuse firing an equal amount of time might be saved. This reminds Mr. Moynihan "of the Irishman who said that the day was not as fine as he expected, and added that he never thought it would be." ! ! !

Regarding the danger of premature explosion by stray currents, as the authors know, this is quite a possibility; but it is useless to discuss the matter with Mr. Moynihan. I know there is a wastage of electric cable through damage by blasting, because I use them below under similar circumstances. If the employment of this system does not involve the employment of more skilled labourers on the Meyer and Charlton, it is because they had not enough work to do before, not that there is not more work to do now.

Mr. Moynihan is evidently so ignorant of the principles of accountancy that he seems to think that because the cost of installation was charged to working costs in this instance it cost nothing at all, and that I am wrong when I state that in comparing the cost with that of any other system it can be dealt with only in the way I indicated.

Mr. Moynihan writes "The most illogical part of this criticism was the claim that *all* the advantages of the electric system could be obtained by a reorganization of the present time fuse basis." I wrote *most* of the advantages claimed for this method are the advantages of single shift against double shift. . . .

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Volumetric Determination of Gold.—"It has been shown that sulphurous acid is capable of reducing auric chloride to aurous in the presence of certain other salts (*Journ. Am. Chem. Soc.*, 1913, xxxv., 546). Through the agency of this reagent it is possibly to estimate gold volumetrically easily and accurately.

In the following work a sulphurous acid solution was prepared by evolving sulphur dioxide from sodium acid sulphite by means of hydrochloric acid, and, after washing the gas, conducting it into repeatedly distilled water. The strength of the sulphurous acid necessarily varied from time to time, due to its gradual oxidation. Its strength varied from 0·005 gm. Au to 0·002 gm. Au per cc. Sulphurous acid solution requires frequent standardisation, owing to its ability to slowly oxidise by absorbing oxygen from the air in the container and pass over into sulphuric acid. After a large amount of experimentation under widely differing conditions the idea of using a permanganate solution as a convenient standardising solution was abandoned. Working under the most favourable conditions, it has not been found possible to obtain results having an error lower than 1% between permanganate and sulphurous acid.

The most reliable method of standardising sulphurous acid is through the agency of iodine. It is entirely immaterial for the methods to be described whether the sulphurous acid solution is standardised on a known weight of iodine in potassium iodide, or whether solid potassium iodide is dissolved in water acidulated with sulphuric or hydrochloric acid and a definite amount of standard permanganate introduced. As a precaution both of the methods indicated were used continuously, and in addition a gold solution whose content had been determined gravimetrically was also used as a standardisation check on the sulphurous acid solution.

1. Auric chloride when treated with potassium iodide liberates iodine according to the equation—

$$AuCl_3 + 2KI = AuCl + 2KCl + I_2.$$

The iodine thus liberated can be titrated in presence of excess of potassium iodide by means of standard sulphurous acid. Starch can be used as the indicator.

2. Auric chloride can be treated with excess of a strong solution of magnesium chloride, then with potassium iodide, and the liberated iodine titrated with sulphurous acid.

3. Auric chloride when treated with potassium bromide yields bromine thus:—$AuCl_3 + 2KBr = AuCl + 2KCl + Br_2$. The liberated bromine can be titrated with a solution of sulphurous acid.

4. Auric chloride treated with magnesium chloride and potassium bromide similarly liberates bromine, which can be titrated with sulphurous acid.

5. Auric chloride when treated with magnesium chloride gives a yellow colour, which can be decolourised by titration with sulphurous acid.

6. Auric chloride and sodium chloride give a yellow colour which by sulphurous acid can be titrated to the colourless or aurous state.

In order to test the efficiency of these principles auric chloride solutions were prepared by solution of weighed amounts of metallic gold in hydrochloric acid and potassium chlorate, and the excess of chlorine removed by the addition of ammonia until the formation of a permanent precipitate, which was then dissolved in hydrochloric acid. Unless precautions of this character are taken the gold solution will either contain some free chlorine, on the one hand, or some metallic gold will separate on the other, when an attempt to remove the free chlorine by evaporation to dryness, or nearly to dryness, is made. Evaporation of a pure auric chloride solution, even on the water-bath, causes the production of a small amount of aurous chloride, which on subsequent dilution gradually changes to metallic gold and auric chloride. By the treatment of metallic gold with hydrochloric acid and potassium chlorate followed by the ammonia and hydrochloric acid treatment, an auric solution can be satisfactorily obtained.

Series 1.

Definite amounts of gold chloride, prepared in the manner indicated, and acid with hydrochloric acid were treated in a dilution of 100 cc. with about 3 gm. of potassium iodide, and the iodine liberated was titrated with a standard solution of sulphurous acid with the following results:—

No.	Au taken (gm.).	Au found (gm.).	Error.
1.	0 0395	0·0395	—
2.	0·0439	0·0438	−0·0001
3.	0·0878	0·0876	−0·0002
4.	0·1098	0·1092	−0 0006
5.	0·1107	0·1108	+0·0001
6.	0·2289	0·2289	—
7.	0 0395	0·4397	+0·0002
8.	0·7301	0·7302	+0·0001

Series 2.

Gold chloride solutions were treated with an excess of a saturated solution of magnesium chloride, potassium iodide was added, and the titration by means of sulphurous acid effected in a dilution of 100 cc.

No.	Au taken (gm.).	Au found (gm.).	Error.
9.	0·0439	0·0440	+0·0001
10.	0·0878	0·0879	+0·0001
11.	0·1061	0·1063	+0 0002
12.	0·2653	0·2652	−0·0001
13.	0·5305	0·5298	−0·0007

Series 3.

Known amounts of gold chloride solution were treated with excess of a saturated solution of magnesium chloride. The yellow colour of the auric chloride is considerably deepened by the addition of magnesium chloride. When sulphurous acid is added from the burette, the solution is bleached progressively until colourless. No other indicator is necessary.

No.	Au taken (gm.).	Au found (gm.).	Error.
14.	0·0439	0·0438	−0·0001
15.	0·0439	0·0440	+0 0001
16.	0·2289	0·2289	—
17.	0·2289	0·2289	—

In numbers 14, 15, and 16, 25 cc. portions of auric chloride solution containing a small amount of hydrochloric acid were treated with sufficient excess of concentrated magnesium chloride to bring the volume to 100 cc., and titrated. In No. 17 conditions were the same, except that 20 cc. of concentrated hydrochloric acid were present, showing that a considerable amount of free hydrochloric acid does not materially affect this titration of auric chloride in presence of magnesium chloride.

Series 4.

Fifty cc. of the slightly acid solution of auric chloride were brought to a volume of 100 cc. with a saturated solution of sodium chloride, and titrated to a colourless state by means of sulphurous acid.

No.	Au taken (gm.).	Au found (gm.).	Error.
18.	0·0439	0·0438	−0·0001
19.	0·2289	0·2289	—

In all of the above titrations a decided excess of alkaline salts was present, and at this point the reaction of sulphurous acid were studied on auric chloride solutions containing free hydrochloric acid, but no salts. In such a gold solution sulphurous acid not only does not show a distinct bleaching effect but metallic gold is thrown out of the solution with ease.

Series 5.

Gold chloride containing a small quantity of hydrochloric acid was brought to a volume of 100 cc., 3 gm. to 5 gm. of potassium bromide were added, and the red colour of the free bromine was titrated to colourless by means of sulphurous acid.

No.	Au taken (gm.).	Au found (gm.).	Error.
20.	0·0439	0·0439	—
21.	0·0439	0·0440	+0·0001
22.	0·0439	0·0438	−0·0001
23.	0·0878	0·0876	−0·0002
24.	0·2289	0·2289	—

Series 6.

Auric chloride solutions were treated with 3 gm. to 5 gm. of potassium bromide, and made up to 100 cc. with a strong solution of magnesium chloride. The red colour of the free bromine was titrated to colourless by sulphurous acid.

No.	Au taken (gm.).	Au found (gm.).	Error.
25.	0·0439	0·0440	+0·0001
26.	0·1061	0·1063	+0·0001
27.	0·2653	0·2655	+0·0002

In the application of the above methods, the gold is originally usually in the metallic form. In order to dissolve the metal either chlorine water or hydrochloric acid and potassium chlorate are convenient solvents. The excess of the oxidising agent cannot be removed by evaporation on account of the production of some aurous chloride. The free chlorine can be advantageously removed by addition of ammonia until a permanent precipitate forms, acidulating with hydrochloric acid, and heating to re-dissolve the precipitate formed by the ammonia. On cooling, 3 gm. of potassium iodide are added, and titration is effected by a standard solution of sulphurous acid, using, if desired, starch as the indicator; or, instead of using potassium iodide, the solution can be treated with potassium bromide or with excess of sodium chloride and titrated by sulphurous acid."—VICTOR LENHER (from *Journal of the American Chemical Society*, xxxv., No. 6), *Chemical News*, August 29, 1913, p. 100. (C. T.)

DETERMINATION OF PLATINUM IN BULLION.—"Since the rise in the price of platinum the assay of coins and other alloys of the precious metals has needed considerable modification. The platinum alloys required a very high temperature in cupellation, causing a great loss of the precious metals. The button contained lead in varying amounts; it was brittle and could not be rolled in the usual way.

The platinum when alloyed with silver dissolved in nitric acid with a brown colour, though not completely, while the residue fell to a fine powder which was hard to gather and did not agglomerate when heated, thus making direct weighing impossible.

The French Mint in Paris published in its annual report of 1901 its experience. The account showed, however, that the difficulties had not been surmounted.

According to the instructions of the Direction Générale des Contributions Indirectes of June 8, 1910, platinum in gold can only be determined with an accuracy of 10/1000 (or 1%) and the error permitted by law for platinum was therefore, fixed at 10/1000 (10 milliemes).

In the following article a method will be described which allows of the determination of platinum and gold with an accuracy of 0·5/1000, or 0·05%; with more extended experience a still greater accuracy may be reached.

With alloys containing more than 250/1000 gold + platinum 500 mgm. are weighed out; with alloys containing less gold and platinum 1000 mgm. = 1000 A.T. are weighed out.

For a preliminary assay the material is disintegrated by hammering and rolling. It is better to use granulations than cuttings obtained with the chisel, since platinum bars show considerable variation; it is also advisable to add silver to facilitate melting and to obtain a homogenous alloy. 250 mgm. of this material are cupelled at a temperature of 200° to 950° C., with an addition of 3 to 15 gm. of lead, according to the amount of base metal present. The button always retains some base metal, mainly consisting of lead. With large amounts of platinum present the button weighs more than the weighed out portion, due to the lead retained.

From the appearance of the button its content of platinum and gold may be estimated with some experience. If this fails, the button is treated in the manner described later, but without the caution which is required later on by putting the button into concentrated sulphuric acid, thereby dissolving the silver. The remaining gold and platinum is weighed.

In the regular assay 250 mgm. (generally in duplicate) are weighed out and chemically pure silver is added, making the proportion of silver to platinum + gold = to 10 to 1. This is cupelled with the necessary amount of lead and after deducting the amount of silver added the total amount of silver + gold + platinum is obtained by weighing the button.

The loss in cupellation may amount to 2 to 20 mgm.; it depends on the temperature, the amount of lead used, the time of cupellation, the kind of cupel used, the amount of silver present, and the composition of the alloy. This loss must be determined by experiments with alloys of varying composition. More accurate results are obtained by cupelling simultaneously a check assay of approximately the same composition as the one to be assayed.

The alloying with silver has the following advantages: The button is free from lead (see Sharwood, Cupellation of Platinum Alloys Containing Gold and Silver, *Jour. Soc. Chem. Ind.*, Vol. 23, p. 412, 1904), it is easily removed from the cupel and no appreciable amount of gold and platinum is lost by this cupellation, because this loss diminishes with the increase of the silver.

The button is put into a parting flask containing 25 cc. concentrated sulphuric acid of 1·84 specific gravity; the button is neither hammered nor rolled.

The acid is heated with a round burner without boiling, avoiding uneven heating or overheating. The silver dissolves in from 25 to 30 min. The residue of platinum, gold and small amounts of silver always retains its coherence, which prevents mechanical losses on decanting. When the acid is heated to boiling considerable bumping takes place; this, however, does not cause the complete dissolution of the silver even when the acid is renewed. Furthermore, platinum is soluble in boiling concentrated sulphuric acid, and boiling causes, therefore, a loss of platinum. After cooling somewhat, the acid is decanted carefully and the residue washed three times with boiling water: the wash water is conveniently passed through a filter. The residue in the flask is put into a porcelain crucible in the usual manner, by filling with water and tilting, and is then annealed together with the filter in front of the muffle.

In this way an addition of gold is avoided, also rolling and the making of cornets. The residue always retains its coherence. Cornets under similar circumstances retain their coherence only within very narrow limits so that the amount of platinum + gold must be fairly well known beforehand.

The residue is dissolved in the smallest quantity possible of aqua regia, the free acid is evaporated as much as possible, and diluted with distilled water. The chloride of silver is filtered. We have now a solution of about 150 cc. containing platinum and gold.

The best and most convenient reduction reagent for the gold is 'hydrazine hydrochloride.' It sometime carries down some platinum (compare Jannasch and Mayer, *Ber. d.d. chem. Ges.*, Vol. 38, p. 2129, 1905); but it is still the best method if certain conditions are observed. Its power of reduction is decreased by an excess of hydrochloric acid and when working at low temperatures.

The gold is precipitated in the solution after adding 15 cc. hydrochloric acid (1·19 specific gravity) and about 1 gm. hydrazine hydrochloride. The solution is left standing at 18° to 20° C. for 1 hr., with occasional stirring, which causes the gold to agglomerate.

After filtering, incinerating, and annealing, the gold is weighed directly (without the crucible). The gold which frequently contains some platinum is melted with a little lead and five to eight times its amount of silver in a cupel in the muffle and then parted with nitric acid, thus dissolving the silver and any platinum that might be present. The remainder is gold.

In the filtrate from the precipitated gold the platinum is precipitated by boiling with an excess of ammonia in about 10 min. (compare Jannasch, *Ber. d.d. chem. Ges.*, Vol. 37, p. 1980, 1904). If the platinum contents are low it is better to evaporate most of the hydrochloric acid present by gentle boiling, as ammonium chloride in great excess prevents the precipitation of platinum through the formation of complex ions. The platinum may also be precipitated by caustic potash or caustic potash and ammonia, in which case, however, the filter has to be washed very carefully with water containing hydrochloric acid. The platinum may then be weighed directly (without crucible), after incineration and annealing.

In order to show that the gold is precipitated quantitatively under the conditions given the following experiments were made:—

(1) 500 parts of gold are dissolved in aqua regia and most of the acid evaporated, diluted to 150 cc., 10 cc. hydrochloric acid (1·19) and about 1 gm. hydrazine hydrochloride added, left standing at 18° 20° C. for 1 hr., gave exactly 500 parts of gold.

(2) The same conditions, but with additions of 15 cc. hydrochloric acid, 499·7 Au,

(3) 20 cc. hydrochloric acid, 499·8 Au.

(4) 25 cc. hydrochloric acid, 499·5 Au.

(5) 20 cc. hydrochloric acid, 12° to 13° C., 498·8 Au.

(6) 1·5 cc. hydrochloric acid, 12° to 13° C., 499·8 Au.

In experiments (4) and (5)—0·5 and 1·2 parts were not precipitated respectively, while in the other experiments the results are within the limits of error.

(7) 500 parts Pt, 1 part Au, 15 cc. HCl, 1 hr. at 18° C., gave 3 parts of gold containing platinum, which after treatment with nitric acid gave 1 part Au and 500 parts Pt.

(8) 120 parts Au, 5 parts Pt, 15 cc. HCl, 1 hr. at 18° C., gave 120·2 parts Au and 5·0 parts Pt.

(9) 300 parts Au, 200 parts Pt, 15 cc. HCl, 1 hr. at 23° C., gave 303·1 parts. Au containing platinum, which after melting with silver and parting gave 299·6 Au and 200·4 Pt.

(10) 300 parts Au, 200 parts Pt, 15 cc. HCl, 1 hr. at 18° C., gave 302·2 parts Au containing Pt, after alloying with silver and parting 300·1 Au and 200·2 Pt.

Check assays:—

(1) Weighed 120 A.T. (1000 A.T. = 1 gm.) Au; 5 parts Pt; 370 parts Ag: 5 parts Cu. Cupelled with 1000 parts Ag and 3 gm. lead. The button weighed 1489 A.T., loss in silver 6 A.T. Precipitated with hydrazine hydrochloride gave 120·2 Au and 5 Pt.

(2) 5 A.T. Au (1000 A.T. = 1 gm.); 5 A.T. Pt; 345 A.T. Ag; 245 A.T. Cu. Cupelled with 8 gm. lead, gave 250 Ag + Au + Pt; loss in silver therefore 5 A.T. At 18° C. 15 cc. HCl, etc., gave 4·8 parts Au and 5·3 parts Pt.

(3) 100 A.T. Au (1000 A.T. = $\frac{1}{2}$ gm.); 100 A.T. Pt; 50 Ag; 250 Cu. Cupelled with 8 gm. lead, after adding 2000 Ag gave 2218 Ag + Pt + Au. Loss in silver, 32 parts. Precipitated at 18° C., etc., gave 103·2 Au containing Pt, which after alloying and parting gave 99·6 Au and 100·2 parts Pt.

(4) 50 A.T. Au (1000 A.T. = 1 gm.); 50 Pt; 20 Ag; 380 Cu. Cupelled with 1000 Ag and 10 gm. lead gave 1104, showing a loss in silver of 16 A.T. Precipitated at 18° C., gave 52·4 Au containing Pt, which after treatment with nitric acid gave 50 Au and 50·2 Pt.

The assay of ores and by-products is carried out by scorification or by the crucible method and subsequent treatment of the regulus by the method described."—TRENKER, *Metallurgical and Chemical Engineering*, Oct. 1913, p. 567. (H. A. W.)

THE CHEMICAL INDUSTRIES AND THE UNIVERSITIES.—"Two of the most important factors in the advancement of chemistry are the universities and the chemical industries, and it is consequently a matter of concern to all of us whether they are working together at present to the best advantage or not. My own observation leads me to believe that they are not, and this address is intended primarily to indicate some of the lines along which it seems to me there might be much closer co-operation." . . . . . . "As indicative of the extent to which chemistry enters into our modern university curricula, I may be permitted to refer to my own Alma Mater, Columbia University, where this subject is taught under seven of the eleven constituent faculties; and, as proof of the demand for higher education in this same field, I would call attention to the doctorates conferred in 1911 (the latest compilation I have seen; "Science" N.S., 36, 129; Aug. 2, 1912) by the leading universities of the United States, when more than twice as many were awarded in chemistry as in any other subject (78 out of a total of 492), the nearest competitor being English (with 34)." . . . . "Perhaps the greatest opportunity of all for co-operation between the universities and the chemical industries, and the one where in many countries such co-operation is conspicuously lacking, is in extending the boundaries of human knowledge by original investigation and research. Here, too, the universities and technical schools are doing what they can, with insufficient funds and equipment, and with an overburdened staff, to serve the community and the chemical industries, both by conducting original research themselves, and by throwing open their great libraries and occasionally certain of their laboratories for the solution of problems of financial value to the manufacturer. The latter, in return, can assist by supplying raw material, additional funds, and knowledge gained in the laboratories of the plant.

When a chemical industry has problems to be solved, these problems can be attacked either inside or outside of the plant. If the policy of the management is that all chemical problems are to be studied only within the establishment, a research laboratory or at least a research chemist must be provided for the plant or for the company. At present, in the United States, probably not more than 75 or 100 manufacturing establishments have research laboratories or employ research chemists. In Germany, and perhaps also here in England, such research laboratories in association with chemical industries are much more common. The great laboratories of the Badische Anilin und Soda Fabrik and of the Elberfeld Company are good examples of the importance attached to such research work in Germany, and it would be difficult to adduce any stronger argument in support of its value than the marvellous achievements of these huge concerns." . . . "There are but four great manufacturing nations in the world—England, Germany, France and the United States. Outside these four, and looking to them for their manufactured articles, stand the one and a half billion human beings in the rest of the world. The prizes to be won in this international trade are beyond compare. They are to be measured not only in money, but also in intellectual advancement, in closer bonds of international friendship, mutual respect and esteem, in national spirit, and in heightened civilisation. . . I will but touch upon one other opportunity for co-operation, and that is in the organisation of Chemists' Clubs, where the members of the profession may regularly foregather and the university man rub shoulders with his manufacturing brother. A Chemists' Club is a most excellent reagent for blending together in one homogeneous, harmonious whole all brands of chemists, and it is quite true that a happy union of theoretical and practical chemists can often be accomplished most pleasantly by clubbing them into it. Our Chemists' Club in New York has brought together, in one big contented family, all the chemists of that metropolitan district, and its influence has extended over the length and breadth of the United States. After all, friendship is the main thing between brother chemists, and I had rather have a colleague's intimate and warm-hearted friendship for me personally than merely his respect and admiration for my scientific achievements.

To recapitulate briefly, the university exists for the community, and its supreme desire is to render ever better and more efficient service. It is the servant of our chemical industries and their agent in the training of the men needed by these industries. It seeks closer co-operation from them in disseminating knowledge and in extending its boundaries, in endowing chemical research and chemical publications, in aiding our chemical libraries, in the matter of patents and legislation, in the creation of great museums of peaceful arts, in the organisation of chemists' clubs and the provision of suitable buildings therefor, and in countless other ways, and if this short and very imperfect presentation of the subject serves in any way to arouse increased interest in these matters, it will have been worth while. Efficiency is the only sure path to supremacy in any field of human endeavour, and closer co-operation between our universities and our industries will go far to assure increased efficiency to both."—Prof. Marston T. Bogert, LL.D., Presidential Address to the Society of Chemical Industry, July, 1913. *Jl. Soc. Chem. Ind.*, 31st July, 1913, pp. 720-724. (J.W.)

Retarding Effects on the Dissolving of Gold and Silver in Aqueous Cyanide Solutions.—"In a 0·25% solution of potassium cyanide the rate of solution of gold is increased by very dilute ozone (electrolytic), but is markedly decreased by more concentrated (3 to 4%) ozone. The cause of the decrease is the formation of a visible layer of oxide. A similar phenomenon is exhibited by hydrogen peroxide, although a large excess of peroxide or a mixture of the two oxidising agents increases the rate of solution. This increased rate is accompanied however by a decomposition of the oxidising agent with evolution of oxygen. In more dilute cyanide solutions (0·01 to 0·05%) even atmospheric oxygen will produce the film of oxide and cause the rate of solution to be much lower than would otherwise be the case. In the case of silver there is no reduction in the rate of solution, but a steady rise to a maximum as more hydrogen peroxide is added. Ozone alone has a very slight effect, but in combination with hydrogen peroxide it increases the effect of the latter."—J. J. Andreejew (from *Z. Elektrochem*, 1913, *19*, 667-672), *Journal of the Society of Chemical Industry*, October 15, 1913, p. 947. (J. W.)

Chemical Analysis of Aluminium.—"Lead, arsenic, tin, manganese and magnesium are rarely present in metallic aluminium and then only in traces; copper is often present, iron and silicon always. For the analysis of aluminium or an alloy rich in aluminium, 2 gm. of the metal borings are dissolved in a mixture of 40 cc. of water, 40 cc. of concentrated sulphuric acid and 2 cc. of nitric acid (sp. gr. 1·48), 300 cc. of water are added, the solution is digested at 80° C. for some time, and the silica is filtered off, incinerated, weighed, then tested with hydrofluoric and sulphuric acids. If arsenic and bismuth are present, the metals of the hydrogen sulphide group must be separated in the usual way, otherwise, the copper and traces of lead are determined by electrolysis. To the filtrate from this group, 20 gm. of ammonium tartrate, dissolved in 100 cc. of water, are added and the liquid is made alkaline with ammonia, and colourless ammonium sulphide added. The filtrate from this is concentrated, made strongly ammoniacal, and the magnesium is precipitated by sodium phosphate and ammonium chloride. The ammonium sulphide precipitate is ignited, dissolved in strong hydrochloric acid, the solution oxidised by bromine and the iron, nickel and manganese precipitated by sodium hydroxide. The filtrate is acidified with hydrochloric acid, boiled and made alkaline with sodium hydroxide; 1—2 gm. of potassium cyanide are added and the zinc precipitated electrolytically in a copper-plated platinum dish. The precipitate from the sodium hydroxide is dissolved on the filter with hot dilute sulphuric acid, oxidised by bromine and the iron and manganese precipitated by ammonia; the filtrate from this is acidified, boiled, treated with ammonia and ammonium sulphate and the nickel deposited electrolytically. For the determination of silicon, iron and zinc only, the metal may be slowly dissolved in sulphuric acid without nitric acid, the iron titrated by permanganate, the silica filtered off and the zinc precipitated as sulphide from the filtrate nearly neutralised by ammonia."—J. CZOCHRALSKI (from *Z. angew. Chem.*, 1913, *26*, 501-503), *Journal of the Society of Chemical Industry*, October 15, 1913, p. 947. (J. A. W.)

THE ROLE OF PRESSURE IN CHEMICAL REACTIONS. —"High pressure has been adduced by geologists as an 'explanation' of many phenomena otherwise not readily accounted for; just as many chemists have attributed actions which they did not understand to colloids or to some catalytic element. There is considerable experimental evidence which, when rightly interpreted, shows that high pressure alone—uniform pressure, especially—is incompetent to produce many of the effects ascribed to it; that its effects have been much over-rated, especially in comparison with those produced by change of temperature. For instance, the influences of temperature and of pressure on solubility of solid substances are such that a pressure measured in thousands of atmospheres is required to equal the effect consequent upon a few degrees change of temperature. In many cases—as, for instance, the system water-silicate at 400° to 450°—the main rôle of pressure is to ensure an appreciable concentration of the volatile component; whether its influence is in such cases subordinate to that of temperature or not depends, of course, upon the stability relations of the various substances involved. Existing knowledge of the domain of high pressures is confined almost entirely to qualitative generalities deducible from theoretical principles; its scope can be increased only by careful quantitative investigation, which, it may be added, is certain to yield valuable results, technical and economic as well as scientific. A recent volume, 'Die Anwendung hoher Drucke bei Chemischen Vorgängen und eine Nachbildung des Entstehungsprozesses der Steinkohle,' by Friedrich Bergius, is therefore to be welcomed for the extension of knowledge which it brings as well as for its suggestiveness.

It records work on the equilibrium relations of the system $2CO_2 \rightleftarrows 2CO + O_2$ at temperatures ranging up to 420° C. and gas pressures up to about 140 atmospheres, and on the effects of water at 300° to 350° and the corresponding pressure on carbon, peat, and cellulose. The main results of these investigations have already been published elsewhere, but the following points are noteworthy: Water at temperatures of 300° to 350° and the corresponding pressure reacts with carbon according to the equation $C + 2H_2O \rightleftarrows CO_2 + 2H_2$, no appreciable amount of carbon monoxide being formed. Under the same conditions it gives, with peat or cellulose, products which closely resemble natural coals, the type of coal produced depending mainly upon the duration of heating. This process goes on to a certain limiting composition only (corresponding to a soft coal); but it was found that the percentage of carbon can be further increased by exposing the product to stress, a fact which tends to show that natural anthracite was formed from a soft coal by the action of the stresses to which any deep-lying bed must have been subjected. A rise of temperature of 10° doubles (as usual) the rate of formation of this artificial coal; whence it is calculated that the age of the natural coal beds is of the order of eight million years, an estimate which is no more uncertain than any of the existing estimates of the age of the earth. In conclusion, it may be pointed out that investigations similar in character to those described by Bergius may be counted on to contribute something toward the solution of the still moot question of the origin of petroleum."—JOHN JOHNSTON (*Jour. Am. Chem. Society*), *Mining and Scientific Press*, September 27, 1913, p. 501. (H. A. W.)

SELF IGNITION OF COAL.—"Coals which when heated to 150°, in a stream of O, show no tendency to increase the temperature, are absolutely safe in storage. Those which show a slight tendency to heat up locally but fail to ignite inside of an hour, are safe enough for storing and transporting on board ship. Those which ignite within 1 hr. and below 150° are relatively dangerous coals. The temperature of self-ignition and the time of heating are necessary to determine relative degree of danger. In general, the heat value of the coal is not an index as to its tendency to self-ignition. Brown coals are more dangerous in this respect than soft coals. The coals tested showed that the greater the O content the greater the liability to self-ignition."—*Gas World*, *58*, 697; *Chemical Abstracts*, Vol. 7, No. 17, Sept. 10, 1913, p. 3012. (J. G.)

ACCURACY AND LIMITATIONS OF COAL ANALYSIS. —"F. claims that it is almost impossible to state definite limits of accuracy for coal analysis. The factors are many and variable that influence the results. The quality and kind of coal, the method of sampling and analysis, and the skill and experience of the analyst affect the final result. Excepting the volatile matter, fixed C and O determinations, the best analytical methods are more accurate than the usual methods of sampling. With properly standardized calorimeter and thermometer heat values are reliable and comparable. A statement of method of sampling should always be made. Illustrated by curves, tables and other data."—A. C. FIELDNER, *Chem. Eng.*, *17*, 50-65; *J. Ind. Eng. Chem.*, *5*, 270-82; *Chemical Abstracts*, Vol. 7, No. 17, Sept. 10, 1913, p. 3013. (J. G.)

RARE METALS IN COPPER.—"Some copper ores contain considerable quantities of rare metals, and many copper refiners take the precaution of saving them by the electrolytic refining process. In a paper in the *Transactions of the American Institute of Mining Engineers*, A. Eilers gives the proportions of rare metals obtained from several varieties of copper. The copper from Garfield, Utah, contains in 5,000 tons:—

Gold	14,400 oz.
Silver	174,000 oz.
Platinum	17 oz.
Palladium	59 oz.
Selenium	2,800 lb.
Tellurium	277 lb.
Bismuth	305 lb.
Nickel	2,000 lb.

Selenium was present in all the varieties of copper examined, and nearly always in greater quantity than the tellurium. Some of the copper contained no tellurium."—ANON, *Journal of the Franklin Institute* (*Brass World*), Oct. 1913, p. 470. (J. W.)

METALLURGY.

ORE-DRESSING IMPROVEMENTS. — "This paper deals only with the concentrating or ore-dressing problem, and gives some thoughts which point towards greater profit. In a paper on 'The Development of Hindered-Settling Apparatus,' the writer showed that there were two main lines of work that would lead to greater saving, lower tailing assay, and, therefore, greater profit. The first is the recrushing of middling or tailing products for the saving of the contained values. This is well understood and largely carried out in existing mills. The second is the saving of fine free mineral at the first opportunity. This is a matter less well understood in the mills, and to dwell upon this the present paper is written.

The writer is himself convinced and hopes to convince the reader that, for obtaining the above result, not only is better classification needed, but also more classification; that is, there should be more spigots or pockets to the classifier. His attention was recently called to a small mill which had a three-pocket cone classifier fed with from 2 to 0 mm. stuff, supplying feed to four Wilfley tables. The first spigot had about all the sand and a great amount of slime, the second and third spigots and the overflow were about all slime. As a result, all four Wilfley tables had wide bands of slime on them and all their tailing products were charged full of fine free mineral. In fact, the cone classifier was not classifying the ore at all, and the mill process was greatly harmed from the lack of classification. It is a proved fact that when the feed to a Wilfley table carries slime, then the Wilfley tailing will carry fine free mineral. When this ore was tested in the laboratory it easily yielded twelve spigots, all of which when fed to Wilfley tables gave no slime bands on the table, and the fine free mineral was absent from the tailing of all the coarser Wilfley tables, and present only in small quantity in a few of the finer.

Better Classification and More Classification.—All classifiers which classify by a rising current of water have a feature which we call the sorting column, in which the sand is trying to settle, and is allowed to settle or is prevented from settling according to the size and weight of the grains, by the velocity of the rising current of water, a stronger current lifting larger and heavier grains, a weaker current lifting smaller and lighter grains. This sorting column, whether square or cylindrical in shape, will have adverse downward currents of water carrying slime down into the spigot, thus defeating the whole object of classification unless some special provision is made to stop it.

To obtain better classification and prevent the adverse down currents, two methods have been employed:—

1. Causing a vortex or horizontal rotation in the sorting column.

2. Using a pulsating rising current.

Both of these methods give better classification by destroying the hurtful downward current, avoiding the carrying of slime into the spigot, and preventing fine free mineral from going into the tailing of all the coarser Wilfley tables.

To obtain more classification we simply lengthen the classifier, using more pockets. As the benefit of this may not be apparent to the concentrator man, figures are here given to prove the point:—

Three-Spigot Classifier.

Spigot.	Quartz, Dia., Mm.	1-3-5 of the Quartz is Galena, Dia., Mm.
1	6·3 to 2·15	6·3 to 0·614
2	2·15 to 0·73	0·614 to 0·208
3	0·73 to 0·25	0·208 to 0·0715
Overflow ...	0·25 to 0·00	0·0715 to 0·0000

Twelve-Spigot Classifier.

Spigot.	Quartz, Dia., Mm.	1-3-5 of the Quartz is Galena, Dia., Mm.
1	6·30 to 4·79	6·30 to 1·37
2	4·79 to 3·66	1·37 to 1·05
3	3·66 to 2·80	1·05 to 0 80
4	2·80 to 2·14	0·80 to 0·61
5	2·14 to 1·63	0·61 to 0·47
6	1·63 to 1·26	0·47 to 0·36
7	1·26 to 0·96	0·36 to 0·27
8	0·96 to 0·73	0·27 to 0·21
9	0·73 to 0·56	0·21 to 0·16
10	0·56 to 0·43	0·16 to 0·12
11	0·43 to 0·33	0·12 to 0·094
12	0·33 to 0·25	0·094 to 0 071
Overflow ..	0·25 to 0·00	0·071 to 0·000

We will take two instances: 1, the classifier which has only three pockets (see table) and is classifying stuff from 0 25 in. (6·3 mm.) to 0: 2, the classifier with twelve pockets, treating the same product. We will suppose that the classification is being done upon quartz (sp. gr. 2·64) and galena (sp. gr. 7·5) ore, and we will consider it to be a free-settling classifier, which will demonstrate the point better than if hindered settling was used. It will be borne in mind that with free settling the classified products after the first spigot and before the final overflow will contain grains where the quartz is approximately 3·5 times the galena in diameter.

If we consider any pair of products except the first, which in both instances has the added increment in it (see paper on 'The Development of Hindered-Settling Apparatus'), and the overflow, both of which require special treatment, different from that of all the intermediate spigots, we believe it will be clear that a Wilfley table treating the second spigot of the three-spigot classifier, which has to separate 0·614 to 0·208 mm. galena from 2·15 to 0 73 mm. quartz, will not have so easy a task as the Wilfley table which treats the second spigot of the twelve-spigot classifier, and has to separate 1·34 to 1·001 mm. galena from 4·70 to 3·51 mm. quartz. The same opinion will hold if the third spigot of the three-spigot classifier is compared with the twelfth spigot of the twelve-spigot classifier, and is true of all the intermediate spigots of the twelve-spigot classifier. The comparison would be much stronger if quartz 2·65 sp. gr., and pyrite, 5 sp. gr., were used, and still more so if quartz, 2·65 sp. gr., and blende, 4 sp. gr., were used.

To put the case still more strongly, the writer adds the following facts:—

The late George Woodbury, of California, found that the Wilfley table could not compete with the vanner in treating unclassified feed of the gold mills of the mother lode of California, which were treating 100 tons a day, more or less. He did not locate the cause of this difference between the machines, or, if he did, he did not make his views public. At the Massachusetts Institute of Technology we have located the cause. The chief loss in the tailing of the Wilfley or vanner, when treating unclassified feed, is the presence of fine free mineral. The maximum size of the grains of fine free mineral in the tailing of the Wilfley table is larger than that of the vanner. This amply explains Woodbury's observation. We have many measurements on these sizes, but want better check results before we publish them.

Next, let us see what a long classifier with many spigots will do which a short classifier cannot do. We shall find, as we follow on down from the coarser spigots toward the finer, that Wilfley tables will yield tailing products free from fine free mineral, and therefore at the minimum assay that can be obtained without further re-grinding. After a time, however, we shall arrive at a spigot where the heavy mineral in the spigot is the same size as the normal size of Wilfley fine free mineral. Then the Wilfley table breaks down, can no longer yield clean tailing, and the vanner should take its place, because the fine free mineral in the vanner tailing is smaller than that in the Wilfley table. If we follow still further down the spigots we shall find a spigot or the final overflow where the size of the heavy mineral is the same as the size of the fine free mineral in the vanner tailing. Then the vanner breaks down, can no longer yield clean tailing, and we go to the new great Anaconda round table, which takes all the rest down to the size of the heavy mineral that floats away in a slow-moving stream of water. With this the methods of water-gravity separation of fine material, that are now in sight, close.

The critic will say: 'But you are diluting your slime beyond reasonable limits by using so many pockets.' The writer answers that that depends upon whether the classifier is evenly fed. If it is not, the water may be unreasonably large in quantity, but if devices can be installed that give even, constant feed, as, for instance, those in the Utah porphyry mills, then there need be no fear of excess water. The following argument will show this: Suppose we have to classify 400 tons in 24 hr., and we wish to classify into fifteen spigots. It takes 3·5 tons of water to 1 ton of sand for feeding the classifier; again 3·5 tons of water rising in a sorting column to allow 1 ton of sand to go down to the spigot, whether coarse or fine; again 3·5 tons of water to 1 ton of sand to discharge it through the spigot. Next, suppose we need 350 tons per 24 hr. from the spigots and 50 from the overflow. What difference does it make whether we settle the 350 tons of sand in one spigot against (multiplying by 3·5) 1,225 tons of rising water, or settle the 350 tons of sand divided out between fifteen sorting columns, requiring 1,225 tons of rising water also divided out between the fifteen sorting columns? The fifteen-spigot classifier, then, if it is steadily and evenly fed, requires no more water and delivers overflow no more dilute than a one-spigot classifier.

The writer believes he has clearly proved that both better classification and more classification are needed.

Let us see what difficulties, if any, have to be overcome in order to put good classifiers into a mill: 1. No one has yet succeeded in designing a good classifier that does not require some attention. The concentrator man who swears by fool-proof machinery and declares that nothing shall go into a mill that is not absolutely fool-proof can never get good classification or make the highest saving. 2. There is a certain lack of breadth about concentrator men who, naturally enough, perhaps, like to have their own machines, and refuse to allow the machines of outsiders to succeed in their mills. One instance of this may be mentioned. A classifier which had a screen in it was tested in a mill. In some way unknown to the designer the screen had holes developed in it as large as a crowbar would make. The manager asked the concentrator man to have samples taken and assays made to see what results the classifier would give. Of course the result was as bad as could be, and the classifier was condemned and thrown on the junk pile. If the holes came by accident, the classifier was not in fit condition to test. This paragraph really belongs to the discussion of professional ethics, which has been so much under discussion lately. For example, has a concentrator man a right to turn down good machinery by unfair tests and deprive the stockholders of his company, whose money pays his salary, of the extra dividend that might be developed?

If, then, it be admitted that fool-proof machinery of certain classes is bad machinery and that more perfect performance can only be obtained where the machines requires greater care, how can this care be secured? The answer is, get a higher-priced man to inspect and see that the machines are run properly and are always in good order.

Good classification can save large sums of money—$25,000, $50,000, $100,000 or more a year. If to run such classifiers we require $1,500 man and three of him for the three shifts, it would pay well to spend £4,500 a year to make any of the above figures of saving: $4,500 is only 18% of $25,000, and 9% of £50,000. What successful business man hesitates to spend 25% or even 50% of the net income in advertising in order to make the business? It is common, every-day custom. Then why should we not spend 18% or 9% of the increased net income in order to bring ore dressing to the highest pitch of profit?

The conclusion, then, is that we need better classification, more classification, and higher-priced, broader-minded men to see that the greatest benefit is derived from the improved machinery."—ROBERT H. RICHARDS, *American Institute of Mining Engineers*; *London Mining Journal*, Oct. 11, 1913, p. 977. (A. R.)

GOLD RECOVERY BY VOLATILIZATION.—"In the *Journal* of June 7, 1913, I notice an abstract of an article by Ben Howe on 'Gold Recovery by Volatilization,' which has been going the rounds of the technical press as something new.

Since the *Journal* has also quoted Mr. Parmalee's discussion of the article, and stated that this method was worked out by Edwin C. Pohlé and me in 1897 and 1898, there is not much more to say on the subject. However, the volatilization of gold by this method was discovered by me as early as 1893, while experimenting on Cripple Creek ores at Aspen, Colo. After going to the Globe works in Denver in 1896, I found that Mr. Pohlé of that plant had made a similar discovery with silver ores, so we began to work out the method as stated.

Our work was published in full in the *Journal*, Aug. 29 and Sept. 19, 1903, and is quoted in Rose's 'Metallurgy of Gold.' Our patents were taken out in U. S. and in all foreign countries where mining operations exist, and are noted in the *Journal* of Oct. 31, 1903.

The process for gold ores, as described by Mr. Howe, is identical with our experiments. The size to which the ore is crushed, the type of furnace used, the temperatures maintained, the percentage of salt required, the method of condensing the fumes and collecting the gold and the character of the gold slime obtained, might easily have been taken from our note books.

We had no difficulty in collecting gold fumes in the manner described. Those of the base metals were more difficult to collect, but we succeeded in making a commercial recovery of all valuable metals or fumes in our experimental plant. The advance in metallurgy and in mechanical appliances since our work was done, together with a better knowledge of the character of the base metal fumes would no doubt enable me to simplify and improve the process considerably at the present time."—S. CROASDALE, *Engineering and Mining Journal*, June 28, 1913, p. 1303. (H. A. W.)

RARE METALS IN COAL ASH.—"The occurrence of vanadium in the ash of coal and other carbonaceous deposits in Peru has been known for some time. A. Jorrissen now announces (*Bull. de la Soc. Chim. Belge*, No. 21, 1913), that molybdenum is found in the cinders of certain Belgian coals, and that it exists in water-soluble form. The moral seems to be that ashes from coal from diversely mineralized districts should be examined for rare and precious metals."—A. JORISSEN, *Engineering and Mining Journal*, June 28, 1913, p. 1309. (H. A. W.)

ALUMINIUM PRECIPITATION AT DELORO, ONTARIO.—"In the *Journal* of May 10, 1913, E. M. Hamilton discussed, in an interesting paper, the use of aluminium as a precipitant of silver at the Nipissing mill in Cobalt. I have read the paper with much pleasure, as for several years I have advocated the use of this precipitant for these silver ores, if for no others.

It may not be generally known that up to the present, excluding the Nipissing bullion, over 14,000,000 oz. of silver have been precipitated by aluminium from cyanide solutions. Most of this has been recovered by the Deloro Mining and Reduction Co. through the cyanidation of Cobalt high-grade ores or speiss produced from these ores, and a minor portion by the O'Brien mill treating low-grade Cobalt ores. A few notes in connection with the commercial development of this process may be of interest.

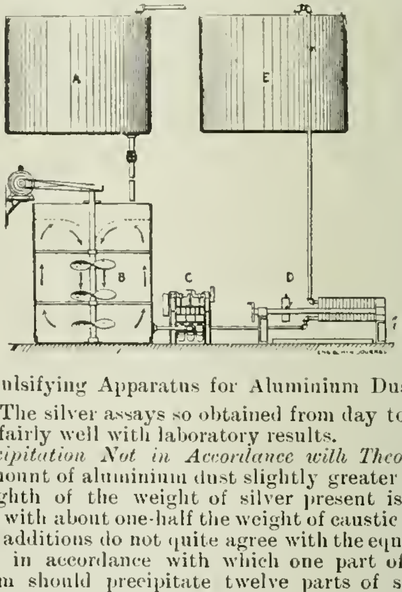
System Devised Through Requirements of Cobalt Ores.—In 1906, I undertook some experiments, with the assistance of the Ontario Bureau of Mines, on the ores of the Timiskaming district in order to develop a commercial process of treating them and saving the by-products. Experiments showed that the ores were fairly amenable to cyaniding even when they contained 2,000 to 4,000 oz. silver per ton, but the cyanide consumption was heavy and zinc was not an ideal precipitant, tending to foul the solutions and give a bullion below market requirements. Considering these questions I was struck with the claims advanced by Moldenhauer, for aluminium as a precipitant. I substantiated these claims and found that the difficulties experienced by the earlier experimenters could be overcome by the use of the metal in the form of dust. The commercial application of the process was introduced by the Deloro Mining and Reduction Co. in 1908, and has been in use ever since.

Assay for Silver by Cyanide Regeneration.—The solution to be precipitated contains from 25 to 250 oz. silver per ton, varying from day to day. The operation of precipitation is illustrated by the accompanying drawing. From the storage vat *A*, 20 to 30 tons of silver solution are drawn off to the precipitation vat *B*, which is fitted with a set of propellers for agitating the solution and creating a vortex for drawing down the light aluminium powder. To determine the amount of silver in solution, a sample of the solution is taken from the precipitation vat, and titrated for free cyanide, then 100 or 200 cc. are placed in a bottle and a slight excess of aluminium powder and caustic soda added. The bottle is corked and shaken for one or two minutes, then decanted through a filter and the filtrate titrated for cyanide.

The gain in cyanide (the difference between titration before and after precipitation) is directly proportional to the silver present as there is a theoretical regeneration of sodium cyanide according to the equation of Roessler,

$$6NaAgCy_2 + 6NaOH + Al_2 = 6Ag + 12NaCy + 2Al(OH)_3$$

After estimating this regeneration of cyanide, the operator may read from a prepared table the silver equivalent. For instance, a regeneration of 10 lb. of cyanide per ton of solution equals 125 oz. silver per ton. The silver assays so obtained from day to day agree fairly well with laboratory results.



Emulsifying Apparatus for Aluminium Dust.

Precipitation Not in Accordance with Theory.—An amount of aluminium dust slightly greater than one-eighth of the weight of silver present is then added with about one-half the weight of caustic soda. These additions do not quite agree with the equation given, in accordance with which one part of aluminium should precipitate twelve parts of silver. The discrepancy might be accounted for by a partial precipitation according to the reactions suggested by Mr. Hamilton for low-grade solution, or by loss due to the action of soda and water on the aluminium, or by incomplete solution of the aluminium due to fine particles being coated with silver. The amount of soda required varies with the acidity of the ore or speiss treated.

One precipitation usually gives from 2,000 to 7,000 oz. of silver, and after agitating for 10 or 15 minutes, it is transferred by a three-throw plunger pump *C* to one of two presses *D*, the barren solution passing to storage vat *E*. The silver in the press is water-washed, air-dried and discharged into a hopper. It is then melted in a Schwartz furnace after fluxing with approximately 16% borax, 8% sodium nitrate, 4% fluorspar, and 5% sand, the bullion being generally about 999 fine. A copy of a page, taken from the daily precipitaion record, at random, is shown in Table I.

Table I.—Precipitation Data at Deloro using Aluminium Dust.

Date, 1913, April.	Tons Solution.	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.	X.
22	33	5·2	156·6	18·0	18·0	Tr.	12·8	160	5280	620	46
23	32	7·8	145·2	20·1	20·0	1.2	12 3	153·7	4920	300	43
24	33	6·3	123·9	16 4	16·6	Tr.	10·1	126·2	4166·2	280	36
25	32	4·6	175·8	19·8	19 8	0·6	15·2	190·0	6080	420	53
25	32	5·7	114·0	15·7	15·8	0·6	10·0	125·0	4000	380	35
26	32	7·1	160·8	20·2	20·2	Tr.	12·8	160·0	5120	600	45
26	32	6·1	110·4	15·7	15·6	Tr.	9·6	120·0	3840	240	33
28	33	5·5	167·7	18·8	19·9	Nil	13·3	166·2	5486	300	49
28	32	6·1	151·5	19·5	19·5	Nil	13·4	167·5	5360	360	48
29	32	3·6	149 1	16·1	16 3	Nil	12·5	156·2	5000	360	44

Col. I. Free cyanide in pounds per ton of solution before precipitation. Col. II. Ounces silver in solution before precipitation. This is laboratory result recorded day after precipitation. Col. III. Titration of cyanide after regenerating with aluminium in bottle. Lb. KCN per ton. Col. IV. Titration of cyanide after precipitation in vat. Col. V. Ounces silver in solution after precipitation in vat. This is laboratory results received nex day. Trace means ¼oz. or less. Col. VI. Gain in cyanide by regeneration in bottle test. Difference between col. I. and III. Col. VII. Ounces silver in solution before precipitation, deduced from gain in cyanide. N.B. Gain of 1 lb. cyanide = 12·5 oz. silver per ton. Col. VIII. Total silver contents of vat in ounces found by multiplying tons by ounces silver per ton. Col. IX. Pounds soda added. Col. X. Addition of aluminium, approximately one-eighth the weight of silver. Amount required read from prepared table.

Cost of Aluminium Precipitation Low.—The production of silver and the consumption of aluminium and caustic soda for the first four months of this year were shown in Table II.

Table II.—Material used in Aluminium Precipitation.

1913	Tons Speiss Treated.	Oz. Silver Produced.	Lb. Soda Used.	Lb. Soda per 1000 oz.	Lb. Aluminium Used.	Lb. Aluminium per 1000 oz.
Jan.	130·0	123,602	4,380	35·4	1289	10·4
Feb.	151 4	165,701	6,570	39·6	1463	8·8
Mar.	160·0	190,045	6,686	35·2	1654	8·7
Apr.	169·1	184,359	6,008	32·6	1590	8·6
Total	610·5	663,708	23,644	35·6	6996	9·1

The figures given for the silver are from the daily precipitation record, and are slightly higher than the bullion shipped for the same period. The bullion shipped from the cyanide department for these months was 643,565 oz. This latter figure does not include slag, absorption of lining of the Schwartz furnace and other material returned to the blast furnace from the melting operation. Estimating from these results the cost of chemicals for precipitating 1000 oz. of silver includes: 35·6 lb. soda at 2c. per lb., $0·712, and 9·1 lb. aluminium at 32c. per lb., $2·91; total, $3·622. The advantages of aluminium precipitation were admirably set forth by Mr. Hamilton, whose extensive experience with such problems lends weight to his opinions.

Use of Patents Offered Gratis.—In 1910, I took out patents in Canada, No. 126,468; United States, No. 959,757, and Mexico, No. 10,899, on the apparatus required for precipitating with aluminium dust, viz., a vat with some means of producing a vortex to draw down and drown the aluminium powder. As this powder is light and subject to dusting if violently agitated at the surface of the solution and is not easily drowned, this vortex would seem to me an important item in either an intermittent or continuous precipitation process. When taking out these patents, the company I was associated with had some idea of developing the process commercially, but I may now say that anyone adopting aluminium precipitation is welcome to the use of the vortex system and I will be pleased to see the idea developed.

In conclusion I wish to thank the officials of the Deloro Mining and Reduction Co. for the help given in developing the process, referring particularly to the large share of the work done by S. B. Wright, the present manager."—S. F. KIRKPATRICK, *Engineering and Mining Journal*, June 28, 1913, p. 1277. (H. A. W.)

TREATMENT OF REFRACTORY GOLD AND SILVER ORES.—"A dilute solution (0·3%) of sodium hydroxide is mixed with the fluid pulp of ore, and the mixture agitated in contact with metallic aluminium or an alloy of aluminium. The ore thus acts as the cathode in an electrolytic action which results in the gold and silver being liberated. When the action is complete, the sodium hydroxide solution is separated, and the residue is treated with a suitable solvent to extract the precious metals."—(Patent) C. BUTTERS, *Journal of the Society of Chemical Industry*, Sept. 30, 1913, p. 915. (H. A. W.)

ZINC-DUST PRECIPITATION OF GOLD AND SILVER.—"The introduction of zinc dust as a precipitator of gold and silver from cyanide solutions is due to H. L. Sulman, who proposed its use in 1894. As originally applied zinc dust apparently offered no advantages over the old shaving process. There were many reported failures due probably to the disinclination of the ultra conservative mill man to work out the details. However, the great success of C. W. Merrill's zinc dust plant at the Homestake again called attention to the undoubted merits of zinc dust, or fume, as a precipitant. To-day zinc dust is recognized as the standard precipitant and in most of the up-to-date large plants on this continent it has displaced the old zinc box. My notes show that in the United States, Canada and Mexico about 50 plants, with a total daily precipitating capacity of 60,000 tons of solution, are now in operation. It is safe to say that 75% of the gold and silver recovered by cyanide in those countries is precipitated by dust.

Probably no single event gave a greater impetus to the use of zinc dust than the fire at the Goldfield Con. mill, which used that method of precipitation, and which was entirely destroyed in 1910. During the fire the precipitate presses became so hot that the excess zinc distilled and the solution spigots were discharging a long zinc flame. The contents of the presses, valued at over $200,000, were found intact and the recovery was complete. What the losses would have been if the gold precipitate had been in zinc boxes can be left to the imagination.

The use of zinc dust offers many advantages, the most important of which may be enumerated as follows: (1) Safety; (2) minimum of attention; (3) completeness of clean-up; (4) neatness and compactness of arrangement; (5) efficiency, high grade of precipitate: (6) low consumption of zinc; (7) cheap grade of zinc used. The first three points in favour of zinc dust may be dismissed by saying that they are obvious. The precipitate is practically in a cast-iron safe, secure against theft or fire. There is no necessity of touching the precipitate between the clean-ups as compared with the daily dressing of zinc boxes. The entire amount of precipitate is available at the clean-up in contrast to the large amount of precipitate left on the long zinc in the boxes.

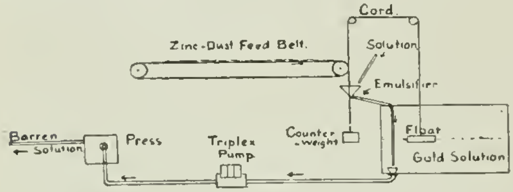


Diagram of Zinc Dust Precipitating Plant.

The arrangement of zinc dust precipitating plants to-day is exceedingly neat and compact. The original process of adding the zinc dust to a large vat of gold solution and agitating is obsolete. The clean-up in this manner of precipitation is not perfect and the consumption of zinc dust is high.

The modern arrangement is to mix a small stream of zinc dust emulsion with a flowing column of solution and forcing the mixture through a press to remove the precipitate. The necessary amount of zinc dust fed to the flowing solution is regulated, usually, by the rate of flow. The most frequently seen type of zinc feeder is an endless belt which is given motion by floats in the gold vat. As the solution in the vat lowers the floats fall, and by means of a cord running over the end drum carring the endless belt the latter is slowly moved. The belt is loaded with zinc dust and as it moves forward the dust is discharged into an emulsifier, which may be a simple cone or, as frequently seen, a small tube mill made of a 3 ft. length of 12 in. pipe. Of late, screw feeders with hoppers holding a day's supply have been introduced. However, in most of the arrangements the rate of dust feed is determined by the speed of an independent motor and thus the necessary ratio between the flow of gold solution and the rate of dust feed is not invariable as when the older belt and float arrangement is used. By attaching the screw feeders to the pumps filling the presses the feeder and press would work together, a condition which is necessary for regular and uniform work.

The mixture of zinc dust emulsion and solution is forced through the presses by means of triplex pumps. The pump, being placed between the presses and the intake of the zinc dust emulsion, plays an important part in thoroughly mixing emulsion and solution. The velocity of reaction between the zinc dust and the double cyanide of gold and potassium is high. The reaction is complete in less than a minute provided that the diffusion of the dust through the pregnant solution is complete. The necessary time for the reaction is furnished by the period the precipitated solution is in the press, it being estimated that the duration of the solution in the press is, on the average, about 1½ minutes for a gold solution and about 3 minutes for a silver solution. However, it is noteworthy that the plants which give the most complete and uniform precipitation are those in which there is the longest length of pipe between the gold solution vats and the presses. The zinc emulsion being introduced close to, or at, the gold vat discharge a long time will elapse before the solution is filtered. In some cases 3 or 4 minutes. This lends itself to the common arrangement of placing the presses in the melting and refining building which is usually placed a considerable distance from the mill proper.

The Bosqui vat, or an equivalent, will increase the time of contact and cause more complete mixing if the presses and gold vat are necessarily placed close together.

The efficiency of precipitation by zinc dust is very high. A Mexican plant treating several thousand tons of solution daily gave the following results: Assay of gold solution, 175 gm. silver per metric ton; assay of barren solution, 1·8 gm. silver per metric ton, an extraction of 99%. With gold solutions assaying from $1 to $2 per ton the barren solutions generally run from 1 to 2 cts. per ton, an extraction of from 98% to 99%.

The speed and efficiency of precipitation by zinc dust is due to the enormous area exposed, as compared with even finely-cut shavings. Good commercial zinc fume (dust) exposes about 300 sq. ft. of surface per pound of zinc. Shavings 1/1500 in. thick expose about 85 sq. ft. of surface per pound of zinc. In a well designed and operated plant the distribution of the emulsion through the pregnant solution, undergoing precipitation, is very uniform. In such a case, using 0·15 lb. dust per ton of solution, each cubic inch of solution going to the press will contain 20,000 to 30,000 particles of dust. Thus a much more intimate and uniform contact between the molecules of gold-silver-potassium cyanide and the molecules of zinc is possible than in the ordinary zinc box.

One of the most troublesome problems of the cyanider is the precipitation of gold and silver from solutions containing copper. Shavings soon become plated with copper and the precipitation of gold and silver presently stops. Dust being added continuously, there are constantly presented fresh surfaces of zinc to do the work. With a precipitate containing as high as 40% copper the precipitation of gold was excellent when using dust; such a result would be impossible if shavings were used.

The consumption of zinc dust primarily depends upon the strength of solution in cyanide as well as the number of ounces of metal to be precipitated per ton of solution. The strength in alkali has also something to do with the consumption. The range of zinc consumption is from 0·128 lb. per ton of solution, in the case of the Homestake, up to 0·404 lb. in the case of a Mexican silver mill using a strong cyanide solution with a high degree of alkalinity. In the first case the amount of bullion to be precipitated is only about 0·017 oz. per ton of solution, in the second case 5·5 oz. per ton. The most common run of solutions will consume from 0·16 to 0·19 lb. per ton of solution. The consumption when using shavings will usually run 25% to 50% higher than when employing dust.

Zinc dust delivered at the mill will ordinarily cost 25% less than the sheet usually used for making shavings. It costs from 1ct. to 1½ct. per pound to cut the sheets into shavings. Hence when ready for use shavings will cost about 30% more than zinc dust. Thus in a Mexican mill shavings cost, ready for use, 9 cents per pound; zinc dust, best German, cost 6·6 cents per pound.

On the whole the grade of precipitate from zinc dust is higher than from shavings. The strong and necessarily impure silver solutions of Mexico give precipitates containing from 75% to 85% bullion, when using dust. When using shavings the precipitates rarely go above 70%. However, it is impossible to judge from loose general comparisons. It is necessary to compare results from solutions of the same type and from the same or, at least, very similar ores. Wherever such comparisons have been fairly made the results are invariably in favour of zinc dust. The importance of handling high-grade precipitate needs no emphasis. The reduction of shipping and refining charges are clearly apparent.

The following table shows some typical results of zinc dust precipitation :—

Zinc Dust Precipitation Data.

Mill No.	KCN pounds per ton.	Assay of solution.	Zinc dust used per ton of solution.	Value of precipitate.	Cost of precipitation per ton solution.	Cost of melting and precipitation per ton solution.
1	0·5	$0·32 Au.	0·128	$4·00 per lb.	$0·0138	—
2	2	1·50 Au.	0·162	18·00 per lb.	0·0182	—
3	—	1·09 Au.	0·167	—	0·0243	$0·0565
4	7	5·5 oz. Ag.	0·404	75% Ag.	—	0·0375
5	5·0	0·19 oz. Au. 0·95 oz. Ag.	0·166	10% Au. 55% Ag.	—	0·0263

The area of filtering surface and cubical contents of press will largely determine the effectiveness of the work. For ordinary gold solutions ($1·50 to $3 per ton) it is usual to allow 0·5 sq. ft. of filtering surface per ton of solution filtered per 24 hours; for cubical contents of press allow 1 cub. ft. for each 24 tons filtered per 24 hours. For silver ores both the area and cubical contents of press is increased. Each ton of solution filtered per 24 hours will require 0·66 sq. ft. of surface and an allowance is made of 1 cub. ft. of press capacity for each 12 tons of silver solution filtered per 24 hours. Any good press may be used, either square or rectangular. Much of the high efficiency of the press is due to the intimate contact between the solution and unacted-upon zinc. By forcing the solution through the minute pores of the cake on the filter-cloth the contact between zinc dust and solution is almost molecular and the thicker the cake the more complete the action.

The amount of attention required by a dust precipitation plant is almost nominal. It is, of course, essential that the right amount of zinc dust be spread upon the feed belt, but beyond that it is only necessary to give a general looking over such as is required even by zinc boxes. The clean-up is simpler than the sloppy process usually seen around zinc boxes. The cakes come out clean and soft and there is practically no loss from spills. The cake will weigh from 115 to 135 lb. per cubic foot, the former being for a gold precipitate containing 10% gold and the latter for a silver precipitate containing 80% silver. The percentage of moisture will vary according to the degree of air blowing the cakes have received in the presses. The average moisture will be 35%, varying from 30% to 45%.

Possibly the most important precaution that should be taken in running a zinc dust precipitation plant is the testing of all zinc dust consignments. Zinc dust varies greatly in composition, and it should be purchased strictly on the basis of a specification. Before any consignment of dust is put into the mill storehouse it should be thoroughly sampled and tested. The dust should be stored in a dry place and it should always be remembered that it deteriorates rapidly when exposed to the mill air. Due cognizance should be taken of the fact that zinc dust is inflammable, and that it might be classed almost as an explosive. Feed belts have been destroyed by the zinc charge becoming moistened by a rain drip and spontaneous combustion, due to rapid oxidation setting fire to the belt.

In operating any kind of feeder it should be noted that zinc dust is extraordinarily responsive to atmospheric conditions. During rainy seasons in Mexico when the air is surcharged with moisture the dust tends to cohere. Then in belt feeders the dust will not discharge in a thin steady stream, but will cling to the belt and slide off in minature avalanches. In screw feeders the dust will sometimes 'arch' under such conditions, and, of course, shut off all feed."—A. M. MERTON, *Mining and Engineering World*, September 6, 1913, p. 429. (A. R.)

THE CONTROL OF NOXIOUS FUMES.—"Recent experiments at the Heroult smelter in California have been so eminently pleasing that hopes are cherished for the control of the fumes which have caused copper producers so much annoyance and brought the California copper industry to the verge of ruin.

The furnace used at Heroult is an old one of the cupellating type. It is 5 ft. in dia., with an approximate capacity of 200 tons per 24 hr. The ore used in the experiments contained an average sulphur content of 50%. It was particularly selected, as a product especially high was desired. For firing California crude oil was employed. Four oil-burners were used and the oil introduced as a spray. The intense heat thus developed facilitated tapping of the furnace an hour after firing. As the gases develop, hydrogen gas is introduced, the idea being the separation of oxygen from the sulphur and release of the latter mineral from the fumes. From the furnace the gases pass into a bustle-pipe which discharges into the flue. This is provided with down-takes which carry the fumes into a carbon chamber. Here the SO_2 is further dissociated, and any SO_3 prevailing brought under complete control, by bringing the gases in direct contact with vaporized carbon generated from red-hot coke. Sulphur dioxide is the gas that has caused so much trouble in the generation of smelter fumes, and it is asserted that this element is completely controlled by the hydrogen and carbon treatments in this process.

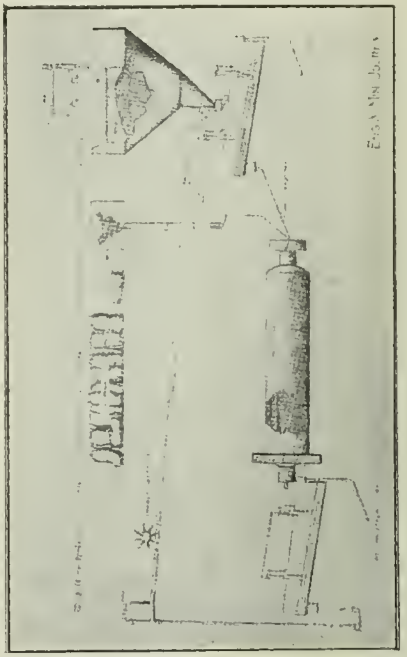
From the carbon chamber the fumes continue downward to the condensing chamber, through which the stream of water is driven by the pump. From this compartment the water and fumes pass into an agitating chamber, where the solution is kept in violent circulation by water admitted under a pressure of 60 lb. to the square inch. This effectively breaks up the bubbles of gas that would otherwise float on the surface, and prevents any marked escape of fumes from this point. The solution continues from the agitator into an open reservoir, divided into two compartments. The sulphur and other metallic elements are deposited in one section, and after the water has constantly delivered the product to one compartment for a specified period, the flow is altered to deposit the material into the second division ; while the first can be cleaned up and the sulphur recovered in a elementary state. The harmless smoke rises from the surface of the reservoir and escapes into the atmosphere.

To guard against a possible escape of fumes from the agitating chamber it is proposed to place an air-tight hood over the compartment, and guide the smoke into a baghouse arrangement. The baghouse has proved highly efficient in recovering zinc oxides and sulphates, as well as other minerals from smoke, and by using the device in connection with the main plant it is claimed all values carried off by the fumes will be effectively recovered, and every portion of the smoke brought under perfect control. It has been shown by experiments that the water used in the condensing and agitating chambers may be employed over for an indefinite period, provided dust chambers are used for the settling of flue dust. Otherwise the deposition of dust would seriously interfere with work. By employing the hydraulic suction draft in preference to the usual forced draft by means of stacks, it is claimed by the inventor that not only is the volume of gases cut down fully four-fifths, but that the capacity of furnace is augmented one-third. This in itself, if borne out in actual practice, commands decided consideration.

The process was developed for the control of noxious fumes, but it was early realised that the method must be economical, otherwise its use would be largely curtailed. Consequently, every endeavour has been exercised to keep operating costs at a low point. In addition to reducing volumes of fumes, and rendering the gases more susceptible to treatment, the inventor, Mr. J. A. Heslewood, claims his furnace will handle zinc or other refractory ores without danger of 'freezing,' basing his assertion on several tests with ores carrying a high zinc content. The intense heat generated by burning crude oil in place of coal and coke, and the powerful suction developed by the hydraulic vacuum, speedily cleared the furnace of obstructions.

The inventor is planning a further series of comprehensive tests, as there remain several problems yet to be solved. A number of mechanical alterations are contemplated, and the furnace walls are being reinforced to guard against air-leakage. The process has not been demonstrated as a complete success, but sufficient in the way of actual results have been shown to command the interest of engineers familiar with the tests thus far made. The control of sulphur dioxide has become the question of the hour in California copper fields, as most of the smelters have been closed because of damage caused by the fume, while the operation of the few plants still active is menaced."—A. H. MARTIN, *Engineering Magazine*, Sept. 1913, p. 901. (J. A. W.)

TUBE-MILL CIRCUIT AT THE ALASKA TREADWELL MILL.—"Concentrate is cyanided at the mill of the Alaska Treadwell Gold Mining Co., at Douglas Island, Alaska. The material is brought to the plant in cars running by gravity into revolving tipples which invert them and drop the contents into round, conical-bottom steel bins. The bins are 15 ft. in dia. and the bottoms have a 55° angle. The concentrate is sampled while in the car, using a long ship-auger. Unslaked lime is added to each of the cars after leaving the tipple, forming a bed for the concentrate and also aiding in discharge of the material. The concentrate in the bin is kept covered with water which prevents its oxidation.



Outline of Tube-Mill Circuit at Alaska Treadwell Mill.

Tightly fitting gates control the bottom of the bin, the contents of which are sluiced directly into a Dorr classifier, the sluicing medium being the oversize return. The classifier makes 24 strokes per minute. Slime from the classifier is led to the Pachuca vats for treatment and the coarse material is delivered into a 5×22 ft. Abbé-type tube mill which is equipped with a spiral feeder. Corrugated sectional lining is used in the tube, which makes 27 r.p.m. The finely ground product from the tube-mill is delivered into another Dorr classifier from which the slime is taken to treatment in the Pachuca vats and the sand delivered into the foot of an air lift. The sand is lifted to a point high enough to enable it to flow back to the first classifier by gravity, and is delivered into an open launder.

Accumulated iron is removed from the coarse pulp passing through the launder by means of a magnetic device which has already been described. The return of this coarse oversize is used to sluice the concentrate from the bin into the initial classifier.

Pebbles are automatically fed into the spiral feeder through a pipe leading from the pebble bin. The pebbles are fed, a few at a time, into the feeder and are controlled by the movement of the first Dorr classifier. The whole circuit is illustrated above."—*Engineering and Mining Journal*, Sept. 13, 1913, p. 502. (A. R.)

LAUNDER DATA OF THE WASHOE CONCENTRATOR.—"In the accompanying table is given information concerning the launder system installed in the concentrator of the Anaconda Copper Mining Co., at Anaconda, Mont. The grades shown are not necessarily the minimum at which the given pulps will flow, but are all safe slopes. Where extra headroom was available, it was utilized in steeper launders. There are a great many factors which effect the flow of pulp in launders which must be taken into consideration in laying out a launder system. The more important of these factors are: The sizes and specific gravities of the grains of solid matter to be transported, the shape of the grains, the sharpness of the fractures, the density of the pulp, the depth of the stream, the presence or absence of slime and the character of the launder material in contact with the pulp. All of the launders in the Anaconda mill are constructed of wood and are rectangular in cross-section.

Material Handled.	Size of Launder, In.		Grade. In. per Ft.	Lining.	Ratio Solids to Water.	Dia. Largest Particle mm.	Depth of Stream in Launder, In.	% of Solids on 60 Mesh.	Remarks.
	Depth	Width							
Undersize ⅞ in. round-hole trommels	15	10	2·0	Cast iron	1 : 3·5	21·0	1·25	82·0	Contains slime
,, 5 mm. round-hole trommels	7	8½	3·0	Cast iron	1 : 5·2	5·0	1·25	75·0	Contains slime
,, 4 mm. round-hole trommels	9½	10½	1·75	Cast iron	1 : 5·0	4·0	1·50	50·0	Contains slime
,, 2½ mm. round-hole trommels	7	8½	2·5	Cast iron	1 : 8·2	2·5	0·50	62·0	Contains slime
Hutch product of harz jigs	7	6½	1·0	Cast iron	1 : 22·7	6·0	1 00	95 0	No slime
Feed to Huntington mill	7	6	1·9	Cast iron	1 : 3·6	2·0	1·25	81·0	No slime
⅜ in. concentrate	10	9	1·3	Cast iron	1 : 12·8	7·0	1·00	100·0	No slime
Evans jig concentrate	9½	9½	1·2	Cast iron	1 : 27·8	2·0	1·50	72·5	No slime
Coarse table feed	7½	7	1·2	No lining	1 : 14·2	0·3	0·50	7·0	Contains slime
Fine table feed	5½	4½	1·2	No lining	1 : 13·0	0·2	0·75	0·0	Contains slime
Table concentrate	11	8¼	0·75	No lining	1 : 15·6	0·2	0·75	0·0	No slime
Table middling	4½	5	0·75	No lining	1 : 15·6	0 2	0·75	0·0	No slime
Table tailing	7	7½	0·75	No lining	1 : 10 0	0·3	0·50	10·0	A little slime
Secondary table feed, remodelled section	7	7½	1·20	No lining	1 : 7·4	0·9	0·50	44·0	No slime
Fine primary table feed, remodelled section	7	7½	1·20	No lining	1 : 14·0	0·35	0·50	2·0	No slime
Coarse primary table feed, remodelled section...	7	7½	1·20	No lining	1 : 4·4	0·9	0·50	46·5	No slime

The gangue material in the ore treated at Anaconda is quartz and highly altered granite. The principal minerals are chalcocite, enargite and pyrite."—*Engineering and Mining Journal*, Sept. 13, 1913, p. 501. (A. R.)

THE USE OF LIME IN CYANIDE WORK.—"Text books on cyanide practice usually treat the question of lime consumption and the addition of lime to the ore in a very casual manner. It appears to be accepted generally that, if the solutions in the percolating and agitating vats are alkaline, the lime has effectively done its work. That is probably the case when the ore is crushed in water, but the now almost universally used process of crushing in cyanide solution and reducing the ore to a mill slime has introduced new metallurgical problems, not the least of which is the mode of adding the lime to the ore. The fact seems to have escaped the authors of text books that lime may be added to an ore in exactly the amount shown by tests and the solutions going to percolators or agitators may have just the right degree of alkalinity, yet, at some point during the process of crushing a great percentage of the pulp may be acid, with a resulting loss of cyanide.

Few mill designers make provision in their plans for the convenient handling of lime in the mill nor adequate arrangements for the addition of the necessary amount of lime to the ore. A shovel or an old box is the sum total of the lime handling machinery at many a pretentious mill.

The lime item shows up in a cyanide mill cost account as one of the heavy charges. There are cases in which the expense for lime is even greater than the combined cost of cyanide and zinc. The consumption of lime is especially high in cyaniding silver ores. The following figures show the consumption in some of the principal Mexican districts and plants: Dos Estrellas mill, 28 lb. per ton of ore: Pachuca district, 20 lb. to 24 lb. per ton; Guanajuato district, 18 lb. to 22 lb. per ton.

Lime performs many functions. Thus: (1) Neutralises acids formed by the oxidation of sulphides, etc.; (2) breaks up metallic salts forming innocuous hydrates; (3) protects the solution of cyanide against carbonic acid; (4) furnishes the alkali necessary for the breaking up of silver minerals; (5) protects the solution against the latent cyanicides in the ore.

The effect of actions 1 and 2, and to a certain extent 5, becomes most apparent during the crushing and grinding of the ore. The amount of lime to be added for these purposes can be determined in the laboratory. The amount of 'protective' alkalinity to be carried in the solution depends upon conditions in the mill and must be found experimentally. So far as the degree of alkalinity required by silver ores is concerned, it may be stated that it is usual to have practically at all times a concentrated lime solution. At Pachuca, for example, the pulp in the agitators carries a slight excess of solid lime, so that, as the lime in solution is used up, it is restored by dissolving some of the calcium hydrate in the pulp. The lime losses are therefore made up as follows: (1) Lime consuming constituents in the ore; lime destroyed by CO_2 in the air used in agitating; (3) mechanical loss by solution passing out of the mill system with the tailing.

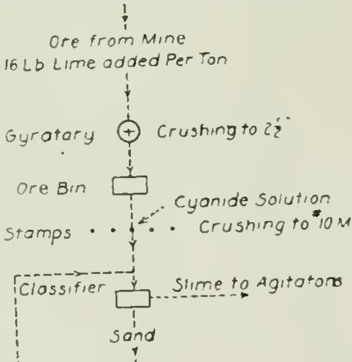
It would unduly extend the scope of this article to consider in detail all the functions of lime and the losses due to causes (2) and (3) just given. In passing it may be stated that the tremendous volumes of compressed air now used in cyanide plants contain very considerable percentages of CO_2, due to the decomposition of lubricants used in the cylinders of the compressors. If compressed air is bubbled through a clear alkaline mill solution a cloudy precipitate of calcium carbonate will soon make its appearance and shortly its effect on the cyanide will become apparent. Care in the choice and use of compressor lubricants and perhaps a purification of the compressed air will be well repaid.

The main object of this article is to call attention to a point overlooked in the general literature of cyaniding, although well understood by many cyanide operators. The point is that the mere addition of lime to the ore, in quantities called for by test, will not suffice to protect the cyanide solutions, even in spite of the fact that the solutions and pulp running to the agitators from the crushers may be quite alkaline.

Not so very long ago practically all ores were crushed in water, amalgamated, the sand and slime separated, the sand treated in percolators and the slime either thrown away into dams or given a perfunctory treatment. Then, the sand having been washed with water, contained almost no acid and a little lime added to the sand or even just sprinkled over the surface of the vat charge fully sufficed.

Now, however, the all-sliming process has become almost the standard process. The ore is crushed in cyanide solution so that almost all passes a 200 mesh screen. During the crushing the solution must at all times be protected against the action of cyanides in the ore and thus the old hit-or-miss style of adding lime must be abandoned or unduly large losses of cyanide will be made.

Take, for example, the conditions illustrated in the accompanying flow sheet. The diagram shows a typical 'all-sliming' crushing and grinding plant working in cyanide solution. The ore consumes 16 lb. of lime per ton and the pulp going to the agitators is continuously alkaline. Yet there is a heavy consumption of cyanide, hastily, and erroneously, attributed to the action of soluble minerals in the ore. The fact was that, roughly speaking, from one-third to one-half the ore was being ground in an acid solution. By adding lime to the tube mill and cutting down the amount of lime fed to the ore before the gyratory crusher, the cyanide loss in the crushing circuit was reduced to one-half.



Flow of Ore in Crushing Circuit.

In such a circuit as that illustrated the conditions calling for the addition of lime will be difficult at each crushing stage. For example, at the gyratory crusher only a small portion of the lime can be used owing to the coarseness of the ore. Yet tests of the fine disclosed the fact that it required much more lime (per ton of fine) than the general average of the ore. This was due to the presence of vugs, many of which contained soft, partly decomposed intensely acid sulphides. The ore may lie in the bin 24 hours before crushing in the stamps. The natural moisture in the ore will generally slack the lime. Under the action of the stamps most of the lime will be converted into milk of lime, washing and neutralising the acid on the outside of each ore particle. A few chips of lime will escape with the 10 mesh ore. On passing to the classifier the portion of the ore already fine enough will pass away together with most of the lime in solution. The sand, containing a higher percentage of sulphides than the original ore, will go to the tube mill with a deficiency of solid lime. The pulp fed to the tube mill will contain only about 40% of solution and this will not be sufficiently alkaline to protect itself against the acid in the great excess of ore. Thus, it is inevitable, in many cases, that the tube mill will discharge an acid product, which on being returned to the classifier will be neutralised by the excess of lime in the stamp mill discharge. Under such conditions I found as high a loss as 50% of the cyanide in the solution fed to the tube mill.

In considering the question of acidity it would seem, at first sight, that the action of the acid would be to form only hydrocyanic acid and that by the addition of lime the titratable cyanide would be entirely restored to the solution. Experience is against that opinion. Cyanide is a chemical compound that tends to break up into a multiplicity of chemical bodies. Even the simplest reactions have an efficiency of only 80%, or so, the remaining 20% going into cyanogen, ammonium compounds, complex organic compounds, etc. In mill solutions the conditions are worse owing to their complex nature. These considerations demand that, in order to economise cyanide in mill solutions, the entire mill circuit must at all times and at all points be kept neutral or at least slightly alkaline.

In a silver mill the lime conditions are slightly better than in a gold mill, owing to the strongly alkaline solutions used in the former mill. Gold mills usually carry the solutions near the neutral point or at best only very slightly alkaline.

The lime conditions in the all-sliming circuit illustrated are worthy of further study. It was found by investigation that the lime was consumed unequally at different points in the circuit. The requirements at different points were as follows:—

At breaker—pounds lime per ton of ore milled ...	3
At stamps—pounds lime per ton of ore milled ...	9
At tube mill—pounds lime per ton of ore milled...	4
Total	16

Clearly, in order to get correct lime conditions in each part of the circuit it is necessary to have those amounts (as shown above) present at every stage. From considerations already discussed it is obviously impossible to obtain correct conditions by adding lime at the commencement of the circuit. Thus, in the case used as an example, the tube mill discharge required three pounds of lime to neutralize the acid developed by grinding in the mill. In other words, 16 lb. of lime were used and only about 14 lb. were really useful.

To add lime correctly, that is in such a manner as to keep the solutions at all times protected against cyanicides, it is necessary to add those amounts of lime such as are consumed by the ore at each stage of the circuit. Thus, in the case in question, by adding 12 to 14 lb. of lime to the ton of ore as it comes into the mill and from 2 to 4 lb. to the tube mill feed, at no stage of the crushing will the pulp run acid.

Characteristics of Lime.—Not only is the ore a very irregular material but the lime used will vary greatly in strength. Different consignments of lime may have a very varying value as a neutralizing agent, although fortunately each lot is usually very uniform. Especially in Mexico, where small kilns are used, the lime is likely to be very irregularly burned. Lime is valuable only for the calcium oxide (CaO) it contains. The percentage will depend on the quality of the lime rock used and the temperature and time of burning. Lime burners are not above running in a lot of inferior lime and every new lot purchased should be sampled and tested.

On the receipt of a consignment of lime at a mill, it should be inspected, sampled and tested. Different lots at the same mill during a year may show the following amounts of lime required to neutralise the acid in a standard ore sample: 6, 13, 9, 8, 7, 12 lb. per ton of the same ore. That is, lot No. 2 was worth less than one-half per pound that of lot No. 1. Lime, for cyanide work, should be purchased strictly on the basis of the percentage of available lime.

Lime in large quantities which has to be stored for long periods should be free from an undue quantity of smalls. It goes without saying that the lime must be free from charcoal, coke, charred wood, etc., as these tend to reprecipitate dissolved gold and silver.

Determination of Available Lime.—The test used for many years differs slightly from that recommended by the Mine Trials Committee. A stronger sugar solution gives quicker results and with a correctness well within that of the method of sampling.

Weigh out 2 gm. of a lime sample crushed to a 60 mesh screen. Pour into a litre flask. Add 50 gm. of crystallized sugar and about 800 cc. water. Shake thoroughly a few minutes and allow to stand for a couple of hours with occasional shaking. Make up to 1,000 cc. and shake. Take out 50 or 100 cc. with a pipette and titrate with standard oxalic acid using phenolphthalein as an indicator.

Standard Oxalic Acid.—Oxalic acid is now generally used in cyanide plants for the determination of alkalinity owing to the ease with which the standard solution can be prepared. By dissolving 22·5 gm. of the crystallized acid in 1,000 cc. of water, 1 cc. = 0·01 gm. CaO. The crystals of oxalic acid contain two molecules of water and they are constant enough in composition to allow a standard solution to be prepared directly by weighing. Where very great accuracy is required the solution can be standardized against chemically pure lime.

The following data on lime may be valuable to mill men and chemists:—

Solubility of Lime in Water.

Temp. °C.	CaO in Saturated Solution.
10	0·128%
15	0·124%
20	0·120%
25	0·115%

Percentage of CaO in Milk of Lime.

Sp. Gr.	Per cent. CaO.
1·01	1·15
1·05	6·10
1·10	11·45
1·15	16·70
1·20	21·25
1·25	25·80

Solubility of CaO in Sugar.

Sugar in 100 cc. Water.	Sp. Gr. on Saturation with CaO.	CaO in 100 cc. of Syrup.
40 gm. ...	1·179 ...	10·6 gm.
30 gm. ..	1·148 ..	7·5 gm.
20 gm. ...	1·104 ...	4·7 gm.
10 gm. ..	1·053 ...	2·2 gm.
5 gm. ...	1·026 ...	0·9 gm.
0 gm. ...	1·000 ...	0·12 gm.

Lime (CaO) is also very soluble in solutions of glycerine.

Storage of Lime.—Many mills have been destroyed by fires starting from the lime house. The shed used for lime storage should be built of fireproof material and thoroughly proof against moisture. Lime should not lie on an earth floor. A satisfactory floor may be built by laying down corrugated iron sheets on a rough board floor raised eight or nine inches above the ground. The iron sheets should be well lapped to prevent any lime working through to the wood. In case a frame building is used both sides and ceiling should be covered with corrugated iron well lapped and closely nailed. Treat lime as a dangerous body."—A. M. MERTON, *Chemical Engineer*, Sept. 1913, p. 113. (H. A. W.)

DESULPHURIZING SILVER ORES AT COBALT.—"Owing to difficulties, until the advent of the Nipissing low-grade mill in the latter part of 1912, cyaniding had played a minor part in the silver production of the camp, being employed in three mills, the O'Brien, Dominion Reduction Co., and the Buffalo, and in each case merely as an adjunct to mechanical concentration. In the fall of 1911, therefore, I undertook a series of experiments in connection with the projected Nipissing low-grade mill to test the possibilities of several proposed methods of treatment and to discover, if possible, an all-cyanide process that would improve on the general practice.

Variations in Results of Experimental Work.—My experiments showed that while some veins gave excellent results by ordinary cyaniding methods, the tailing from the treatment of other veins was persistently high, owing to the presence of varying amounts of the complex minerals, pyrargyrite, tetrahedrite, proustite, dyscrasite, and argentite. Some of the veins on the Nipissing property contain considerable amounts of these minerals, and my experiments were therefore directed to discover some inexpensive chemical or electrical process for breaking up these refractory compounds and so rendering them amenable to cyanide. Working along these lines, I finally discovered that all of these minerals excepting dyscrasite were readily decomposed into their respective elements when brought into direct contact with aluminium in an alkaline solution. This preliminary reducing treatment left the silver in a spongy metallic state, and when followed by the usual cyaniding process the results were found to be very satisfactory.

In order to study their behaviour in detail, specimens of exceptional purity of each of these minerals were secured, and samples for further experiments were prepared by grinding the minerals with quartz, in a porcelain mortar, to pass a 100-mesh screen. The attempt was made in each case to dilute the silver-bearing mineral so as to give as nearly as possible a sample that would assay 30 oz. silver per ton. The results of three sets of experiments are given below.

In experiment A, ordinary cyanide is used; experiment B is straight cyanide with the addition of enough lead acetate to combine with the calculated amount of sulphur in the sample. In experiment C the sample was given a preliminary reducing treatment for ten hours in a 0·25% solution of caustic soda, dilution 2 to 1 in an open-mouth bottle in which were placed three 6 × ½ × 1/32 in. strips of sheet aluminium The strips were then removed, the sample filtered and given the ordinary cyanide treatment without washing or drying.

In each case 100 gm. of sample was taken. The ordinary cyanide treatment in all the experiments consisted in treating the ore with a 0·25% cyanide solution, dilution 3 to 1 and agitating for 48 hours, lime being added to the extent of 5 lb. per ton. Agitation was effected by rolling in large open-mouth bottles.

Table No. 1.

		A.			B.		C.	
		Plain cyanide.			Plain cyanide and lead acetate.		Desulphurizing and plain cyanide.	
No.	Mineral.	Ag, oz. per ton.	Residue, oz. per ton.	Extraction. per cent.	Residue, oz. per ton.	Extraction, per cent.	Residue, oz. per ton.	Extraction, per cent.
1.	Pyrargyrite ...	27·40	19·60	28·46	22·40	18·24	3·29	88·70
2.	Proustite ...	31·4	15·15	51·90	21·20	32·70	0·59	98·00
3.	Tetrahedrite ...	29·97	24 21	19·20	28·23	5·80	6·44	78·50
4.	Argentite ...	26·70	8·20	69·28	0·54	97·97	5·18	80·59
5.	Average ...	28·89	16·63	42·21	17·87	38·70	3·87	86·45

Discussion of Results.—Experiment B shows that the addition of lead acetate in every case is a detriment as compared with ordinary cyaniding except with argentite where the extraction is raised to a marked extent. It is interesting to note that similar results were obtained by Theo. P. Holt at the Utah School of Mines in 1909.

Experiment C shows a remarkable increase in extraction as a result of the preliminary reducing treatment in every case except with argentite, where the results are lower than in experiment B. This was owing to the poor contact with aluminium sheets during the preliminary reducing treatment as a result of the tendency of the mineral to float as a black scum. Further experiments where granulated aluminium was substituted for the sheet metal gave 90% extraction as a result of the better contact obtained.

In practice, the system of agitation employed gives a better contact and therefore a better reduction and higher subsequent extraction than the rolling agitation employed in these experiments.

The Reducing Treatment and Probable Reactions Involved.—By the preliminary treatment the silver, and in part at least, the antimony and arsenic, are reduced to the metallic state, and are so found. The reduction is accomplished by the nascent hydrogen resulting from the action of caustic soda on the aluminium according to the following equation :

(1) $2Al + 2NaOH + 2H_2O = Na_2Al_2O_4 + 6H$.

The probable reactions involved in complete reduction are indicated by the following equations :

(2) Argentite.

$$6H + 3Ag_2S + 6NaOH = 3Na_2S + 6H_2O + 6Ag$$

(3) Pyrargyrite.

$$6H + Ag_3SbS_3 + 6NaOH = 3Na_2S + 6H_2O + 3Ag + Sb$$

(4) Proustite.

$$6H + Ag_3AsS_3 + 6NaOH = 3Na_2S + 6H_2O + 3Ag + As.$$

The reactions being reversible, probably the arsenic and antimony are not completely reduced to the metallic state in practice, and the investigation of the subject is rendered difficult by reason of secondary reactions by which the arsenic and antimony are possibly redissolved to form arsenates and antimonates by the excess caustic of the reducing solution, and the protective alkali of the cyaniding solution. The working solution shows the presence of these compounds, but in practice they are found to have no detrimental effect either in the reducing or the cyaniding treatments. The solution assays, antimony 0·0084% and arsenic 0·026%.

Necessity of Fine Grinding.—Further experiments along these lines established the necessity of fine grinding to raise the extraction to the desired point. The results of one series of experiments are given in Table No. 2. As the samples used in the previous experiments were used up, new samples had to be prepared. These were ground to pass a 200 mesh screen, 12·4% of this product being a fine −200-mesh sand, and were calculated to have as nearly as possible the same silver content as the former samples. The experiments with the addition of lead acetate were discontinued and the plain cyanide treatment was run against cyaniding after the desulphurizing treatment. The same conditions were maintained as described in experiments given in Table No. 1.

Table No. 2.

		Plain cyanide.			Cyanide after desulphurizing.	
No.	Mineral.	Head assay, oz. Ag.	Residue, oz. per ton.	Extraction.	Residue, oz. per ton.	Extraction. per cent.
1.	Pyrargyrite ...	24·40	10·75	60·00	1·48	94·60
2.	Proustite ...	30·90	9·80	68·20	0·22	99·20
3.	Tetrahedrite ...	28·40	12·47	56·10	2·95	89·60
4.	Argentite ...	27·04	4·74	82·40	2·14	92·10
	Average ...	27·68	9·44	66·67	1·69	93·87

My experiments also established the fact that the degree of fineness required varies with the amount of antimony present. Some results from plain cyaniding that show this clearly are given in Table No. 3. The method of preparing the samples and the conditions of the experiments were the same as before.

Table No. 3.

		Crushed through a 100-mesh screen.			Crushed through a 200-mesh screen.		
No.	Mineral.	Head assay, oz. per ton	Residue, oz. per ton.	Extraction, per cent.	Sb, per cent.	Residue, oz. per ton.	Extraction, per cent.
1.	Dyscrasite ...	28·40	14·20	50·00	17·64	5·40	80·98
2.	Impure silver	27·60	4·90	82·24	9·64	1·49	91·00
3.	Impure silver	31·49	1·40	95·50	5·15	0·76	97·50
4.	Clean silver	31·49	0·28	99·10	0·63	—	—

Commercial Tests.—I then undertook a large number of tests on a commercial scale, using ordinary run-of-mine ore to determine the following points: (*a*) the effect of the reducing treatment; (*b*) the time of treatment; (*c*) the degree of comminution required to give the best economic results.

In Table No. 4 are given the results of two series of experiments, showing the effect of the reducing treatment as against plain cyaniding, and giving the results of different times of treatment. In these tests 50 lb. charges were used, and desulphurizing was effected by agitating for ten hours by mechanical means in a small vat. 24 × 18 in., with paddles to which were tacked 4 × 8 in. sheets of aluminium. When not otherwise stated, the conditions maintained were the same as in previous experiments. The samples used in these experiments were ground in a pebble-mill to pass a 200-mesh screen.

Table No. 4.

H—Plain cyanide treatment. Cyanide strength, 0·25%; dilution, 3 : 1. Lime, 5 lb. per ton of ore.

Head assay, oz. per ton.	Residue after 24 hours.	Extraction per cent.	Residue after 48 hours.	Extraction per cent.	Residue after 72 hours.	Extraction per cent.	Residue after 96 hours.	Extraction per cent.	Residue after 120 hours.	Extraction per cent.
26·13	5·8	77·80	3·76	85·61	2·76	89·43	2·6	90·04	2·3	91·19
25·45	6·2	75·65	4·90	80·74	3·80	85·96	3·6	85·85	3·1	87·81
I—Desulphurized before cyanide treatment. Cyanide strength, 0·25%: dilution, 3:1. Lime, 5lb. per ton of ore.										
26·13	2·8	89·28	1·64	93·70	1·58	93·95				
25·45	3·0	88·21	2·22	91·22	1·70	93·3				

These results show clearly that on the run-of-mine ore the desulphurizing treatment raises the extraction appreciably and greatly reduces time. These and similar experiments establish the efficacy of the desulphurizing process and point to 48 hours as the most economic time of treatment.

Degree of Comminution.—It was early apparent in this preliminary investigation that extremely fine grinding was absolutely essential to give a high extraction in the desired time (48 hours), and this fact has been substantiated by many later experiments.

Table No. 5. giving the results from plain cyaniding on sized material, shows this clearly. For these experiments, two 50 lb. samples of mine ore were crushed, one to pass through a 150 and the other a 200-mesh screen, after which they were treated for 48 hours in a 0·25% cyanide solution (dilution 3 : 1) with the addition of 15 lb. of lime per ton of ore.

Table No. 5.

J Through 150-mesh. Sand + 200-mesh, 0·84% Sand − 200-mesh, 42·4%			K Through 200 mesh. Sand − 200-mesh, 16·36%		
Head assay, oz. per ton	Residue, oz. per ton.	Extraction, per cent.	Head assay, oz. per ton	Residue, oz. per ton.	Extraction, per cent.
28·6	3·6	87·4	28·6	2·48	91·3

The same fact was established by means of detailed screen analysis of mine ore, the results of which are not included here.

Aluminium Precipitation.—The circumstances that led to the adoption of aluminium precipitation in connection with this process together with the results of some of my experiments, have already been published by E. M. Hamilton, but a few further remarks at this point may be of interest.

In my preliminary experiments, zinc was used for precipitation and for a time all was well, but at the conclusion of a series of experiments on ore that contained considerable amounts of the complex minerals, I was surprised to find my working solution badly fouled and proceeded at once to investigate the cause. The deterioration of the working solution has usually been attributed to the complex silver minerals, and one writer explains the action as due to the formation of soluble sulphides, sulphocyanides, and ferrocyanides. My investigations showed that not only were these compounds formed to some extent, but, which is more important, proved the presence in small amounts of soluble arsenites, thio-arsenites, antimonites, and thio-antimonites. These compounds are strongly reducing and are oxidized to arsenates, antimonates by the dissolved oxygen of the solution, the action being accompanied by the decomposition of cyanide. The complex minerals were clearly not to blame, as those had already been reduced to their respective elements by the preliminary desulphurizing treatment, and, while the formation of the above-named compounds accounts, in part at least, for the heavy consumption of cyanide, they are not present in amounts large enough to seriously affect the efficiency of the working solution. In any case, the oxidation of these compounds by aeration and the use of oxidizers would restore the solution to normal efficiency if they were the cause of its deterioration, but these were found to be without effect. The cause of the trouble then was evidently to be found elsewhere.

I then started a new series of experiments with a fresh solution and followed the action closely, testing the solution after each step in the process. The results proved favourable in every case until the solution passed over the zinc-box, after which it was found that its dissolving efficiency was greatly impaired. The cause of the trouble was therefore shown to be the use of zinc as a precipitant, and further experiments, given in the article already referred to, proved this conclusively.

Fatigue of Solution.—In what form the zinc exists in the solution, and why, in conjunction with arsenic and antimony, its deteriorating effect on the dissolving power of the solution should be so marked is not apparent.

It was found that while the addition of oxidizers to the solution had no effect, after allowing it to stand for eight days the solution recovered its original dissolving power. No explanation of this peculiar phenomenon has suggested itself, and as by this time it had been proved that the use of aluminium dust instead of zinc as a precipitant would solve the whole difficulty, investigation of this very interesting subject was dropped for lack of time to continue it further.

The method of treatment as outlined above was immediately put into practice at the Nipissing low-grade mill on the recommendation of Charles Butters, the consulting engineer. As this is the first and as yet the only commercial application of the desulphurizing process, a brief description of the practice followed here will be of interest.

Nipissing Low-Grade Mill.—The object of the new low-grade mill was to treat the low-grade wall rock by cyaniding and likewise recover the silver in the form of fine bullion. The mill was designed and constructed by James Johnston of the Butters Engineering Co., and owes much of its success and smoothness of operation to his experience and foresight. Operations on a small scale were commenced on November 16, 1912, and after a short period of minor adjustments started at full capacity. Up to the present it has continued to run with gratifying success.

Nature and Analysis of Ore.—The rock is mainly the Cobalt series of conglomerate and is very hard and tough. The following is an analysis of the average run of mine ore :—

	Per cent.		Per cent.
Ag	0·106	Pb	0·064
Cu	0·270	CaO	9·020
As	1·880	MgO	4·330
Fe	1·920	Al_2O_3	10·030
S	0·640	CO_2	11·060
Bi	0·010	Insolubles ...	59·840
Ni, Co	0·730	Hg	Trace

Washing Plant.—The washing plant, where the ore is crushed in breakers and given a preliminary jigging treatment, is not part of the mill proper, as the concentrate from this section is credited to the mine. The ore from the mine averages about 60 oz. per ton, and the tailing from the washing plant as sent to the mill proper averages about 28 oz. per ton.

Stamping and Fine Grinding.—In the mill proper, the ore is crushed by stamps in a 0·25% solution of caustic soda, lime being added to the extent of 5 lb. per ton of ore. The lime is used merely for settling, to facilitate subsequent decantation of clear solution back to the battery storage vat, the alkalinity for cyanide treatment being furnished by the caustic solution. The fine grinding is effected by a closed system of tube mills and classifiers, two of the tube mills being used to regrind the battery discharge, and the other two for the ultimate fine grinding. Of the final product, only 0·5% remains on a 200-mesh screen, 16% to 20% is a fine −200-mesh sand, and the remainder an impalpable slime. After settling and decanting the solution, the thickened slime passes on to the reducing treatment.

Desulphurizing Process.—The pulp is given a preliminary desulphurizing treatment by being passed through a tube mill which is charged with aluminium ingots. The final treatment is given by lining the filter stock vat with aluminium plates and agitating for about ten hours by mechanical means. From the stock vat the pulp is drawn off by the filters as required. After filtering, the cake carries 26% alkali solution as moisture, and is thence discharged without washing to the cyanide vats.

Cyanide Treatment.—The cyanide treatment consists of agitating the pulp for 48 hr. in a 0·25% cyanide solution, dilution 2·5 : 1. The pulp is then settled, the excess solution decanted, and after being again agitated the pulp is pumped to the stock vat for filtering.

Precipitation and Treatment of Precipitate.—As already mentioned, the pregnant solution is precipitated with aluminium dust. The details of the mill practice, together with a statement of costs, are included in the article by E. M. Hamilton. The precipitate is then sent to the refinery of the high-grade mill, where it is melted in a reverberatory and refined, the bullion averaging 999 fine.

Early Difficulties.—When the process was first put into operation, the mechanically agitated stock vats were depended on for the desulphurizing treatment. Here the aluminium plates were soon found to form a coating which was thought to be a calcium aluminate, and the reducing action was seriously retarded. To overcome this difficulty, the tube mill charged with aluminium ingots was added. This kept the aluminium clean and bright, but the aluminium consumption was increased as a result of the wear on the ingots. Lately, however, the trouble has been found to be due to impure aluminium containing iron and silica ; sheets of pure aluminium are found to remain clean and to have no tendency to form a coating.

Comments on Practice.—Crushing a neutral ore in an alkaline solution is unusual, though not unknown, in cyaniding, and the 26% of the alkali solution passing over with the cake to the cyanide vats will be criticised as being contrary to general practice. However, instead of being detrimental as is ordinarily maintained, in this particular case, namely, with Cobalt ores where the native silver is associated with antimony, the addition of alkali to the cyaniding solution has proved to be a decided benefit. Daily tests, running the working mill solution against fresh cyanide solution, show in every case an increased solvent power of from 0·2 oz. to 0·5 oz. of silver per ton of ore in favour of the mill solution. In this connection, the behaviour of the mineral dyscrasite is interesting. As mentioned at the beginning of this article, the reducing treatment has no effect on this mineral, probably due to the fact that it is a complex of variable composition of the metals, silver and antimony, and does not contain sulphur. However, the results from treating this mineral by plain cyaniding, compared with the results of similar treatment, after the preliminary reducing treatment, show a decided advantage in favour of the latter, owing to the beneficial action of the caustic soda solution during cyanide treatment. A further advantage of the caustic in the cyanide solution is the fact that it is necessary to precipitation with aluminium dust and saves the addition of caustic at that point.

Application to Gold Cyaniding.—A promising field for investigation is the possible application of this desulphurizing process to the cyaniding of refractory gold ores. A sample of sylvanite was found to reduce readily to metallic gold and tellurium, but lack of time and the difficulty of obtaining a pure sample of the gold telluride for experiments, have prevented further investigation on these lines. However, I have every confidence that the desulphurizing process will prove to be a valuable adjunct to the treatment of these ores by cyanide."—J. J. DENNY, *Mining and Scientific Press*, September 27, 1913, p. 484. (H. A. W.)

ORES AMENABLE TO CYANIDING.—*Physical Characteristics of an Ore Influence its Treatment.*—It must not be forgotten that the suitability of an ore for cyanide treatment does not depend altogether upon its chemical character, but also, in large part, upon its physical properties. It is easy to see that some chemical combinations will be difficult to cyanide, but it has not always been recognised that the physical characteristics also have a great influence upon ore treatment. They may not go so far as to exclude it altogether from the possibility of successful cyaniding, but may certainly make necessary a different kind of plant from that required for ores more easily handled. It used to be considered that an ore which made a great quantity of slime was an embarrassing one to cyanide, but at present it may be said that such an ore presents facilities which make its treatment simple and cheap. At this stage of the metallurgy of gold and silver by cyanide, the greatest physical difficulties are presented by ores which cannot be readily reduced to extreme fineness, and in which that extreme fineness is necessary to liberate the contained gold and silver, so that cyanide solutions may have maximum opportunity to act. It is easy to see that the additional grinding required to subdivide finally an ore of this kind would cost a great deal more, and even require a plant of different character from that required by an ore which could be easily and cheaply ground to an extremely fine state. It is by no means impossible that an ore should require practically no grinding at all to enable it to be successfully cyanided. Examples are on record of ores which have merely been put through a breaker, principally for the reason of making a product which would be rather simple to handle mechanically, dumping this product into a cyanide tank and subjecting it to a leaching process. The explanation of this situation is that the ore was an oxidized one and was largely porous, and that the gold, which was its principal valuable constituent, could be reached by the solution without difficulty. A comparatively high extraction might result in such cases. One example of a somewhat similar case in operation at the present time is that of the Wasp No. 2 mill in the Black Hills of South Dakota, where the ore is only passed through breakers and rolls as a preliminary preparation for cyaniding. The rock is reduced to ¼ in. in size, delivered on belt conveyors to leaching tanks, and subjected to leaching for a protracted period. The explanation in this case is that the gold is almost entirely in the cleavage planes, which are naturally the weakest points in the rock. By this coarse breaking most of the contained gold is exposed to the action of cyanide solution.

Ores which contain large proportions of clay or colloid material used to be considered objectionable because they interfered with easy leaching. Where it was necessary to grind ores down to pass a 30-mesh screen, or even a great deal finer, it was found extremely unsatisfactory to try to leach both sand and slime together. The clay, or colloid material, formed impermeable, or difficultly permeable beds, which retarded or stopped altogether flow of solution through the tank. Great relief was felt when it became known that sand and slime could be separated. At first the sand portion only was treated and the slime either thrown away or impounded in dams for future treatment, should a process become available. Such a treatment has not only become possible, but sometimes is easier and cheaper to manage than leaching, so that now not only is the slime treated, but effort is made to get the whole of the material in the form of slime and treat it by agitation. Here the tendency has been to go to the other extreme, and many ores are being agitated which are really not sufficiently ground to justify that treatment. It takes a large amount of power and results in great wear on machinery and plant to try to agitate sand which is even comparatively coarse, and it is probable that if the situation were regarded philosophically, an effort would be made to separate sand, no matter how fine it is, and leach it. There is certainly no great difficulty about leaching a clean, granular sand, irrespective of the size of the separate particles. An additional physical difficulty with some ores is the reluctance with which the slime, or finely divided colloid material, settles when in solution. This difficulty may be so marked as to require specially expensive treatment, different plant design, and even a different treatment system. During the days when the decantation system of slime treatment was in vogue, this objection to settling was a most serious difficulty. At the present time, however, when slime filters, together with continuous agitation, are so much in use, it is not necesssry that the slime be easily settling, as it is not required to settle it at all. It is agitated continuously throughout the extraction treatment, and when that is finished, it is passed through the filter, which dewaters it successfully, without the difficulty of settling.

Chemical Nature of Ore of Greatest Influence.—The chemical nature of an ore is usually referred to when statements are made as to its suitability for treatment by the cyanide process. Interference, such as this, results from elements which occur in the complex ores, and which obstruct successful cyaniding either by destroying the cyanide solution, consuming all its available oxygen, or excessively consuming the cyanide itself by its dissolving the baser elements themselves to an excessive extent. An element like copper, for instance, which really has no detrimental effect upon cyanide solutions, is just as prejudicial to successful cyaniding as an element which might act in an entirely different way, because if it is in suitable form, it is entirely soluble itself and will use up excessive quantities of cyanide, producing no equivalent return. Purely copper ores are, of course, not treatable by cyaniding, because the amount of cyanide necessary to dissolve the metal is worth more than the metal recovered. Statements of this sort, of course, are subject to reservation, as methods may be discovered, and are entirely possible, whereby copper or any other element dissolved in excess by cyanide solution, may be recovered with a return of the entire amount of cyanide required to dissolve the metal, in which case these elements could no longer be considered as interfering with straight cyanide practice. The recent development of aluminum as a precipitant points to the entire probability of such methods coming into practice in the near future. At present they have not received a great deal of study, so far as the general public is aware of, but it is easy to see that if the cyanide used up in dissolving can be returned to the solution with the extraction of the copper itself from the solution, there is nothing to prevent ores containing large quantities of copper being readily treated by the cyanide process. Other methods have been suggested for doing the same thing, among which may be mentioned the ammonia-cyanide process, which really presents probabilities for successful treatment.

Iron Not Generally Harmful in Cyaniding.—Metallic iron was formerly considered a great objection to cyaniding, as it was believed that it robbed the solution of oxygen, thus materially reducing the soluble effect of the solution on the gold. Now, however, it is well recognised that iron pipe, tanks, valves and other fittings have no effect whatever upon the cyanide solution. It is undoubtedly true, nevertheless, that finely divided iron, such as filings, abrasions from stamps, pan dies, and other grinding machines, does have a reducing effect upon the solution. Some iron ores are readily acted upon by cyanide solutions, and are injurious to treatment. The yellow oxide of iron, limonite, is not so much acted upon by solutions, but has a strong tendency toward making slime, and is somewhat objectionable for that reason. This slime is particularly pasty and is hard both to settle successfully and to filter.

Pyrite, the well known iron sulphide, and the one most generally found in both gold and silver ores, is acted upon by cyanide slowly, although it is not easily decomposed White-iron pyrite, or marcasite, is not as soluble as pyrite, but decomposes more readily. The decomposition products of sulphides of iron are the products which are most harmful. Among these may be mentioned ferrous sulphate and oxide, and sulphuric acid, all of which consume cyanide. In this case iron enters solution as potassium ferrocyanide, and the alkaline cyanide is neutralized by the acid.

Sulphur Forms well known Objectionable Compounds.—As is well known, alkaline sulphides are found in solutions where any appreciable amount of sulphur is contained in the ore. Sulphur acts to form alkaline sulphides, and does not directly influence the cyanide. The alkaline sulphides react with cyanide and oxygen and form sulphocyanides, or thiocyanates, or they may unite with oxygen to form sulphates, the result being abstraction of oxygen from the solution. Probably the most important and detrimental action of alkaline sulphide is to precipitate silver from cyanide solutions in which it is dissolved. The silver is precipitated as silver sulphide, and thus a cycle of operations is set up which prevents the successful recovery of the silver. When thus precipitated from cyanide solutions, silver is extremely difficult to redissolve, and in some cases it may be said to be impossible. There are remedies for the ill effects of sulphur, of course, the most popular of which is the use of lead salts, acetate or oxide, which unite with the alkaline sulphide and form insoluble lead sulphide which is precipitated, thus removing the sulphur from solution.

Copper Dissolves Too Readily in Cyanide Solution. Copper has already been mentioned as one of the elements which are objectionable in the cyanide treatment of ores. The objection, as has already been mentioned, is not that the element has any particular detrimental action upon cyanide, but is due to the extreme solubility of the metal itself in cyanide solution. Copper in many forms, such as sulphides, silicates and other well known copper minerals, is not readily soluble, and these minerals do not cause any great difficulty in cyanide treatment. When, however, one encounters soluble copper, such carbonates, oxides, hydrates, etc., they seriously disturb treatment by dissolving readily and using up too much cyanide to allow economical results.

Not only is the cyanide itself used in excess, but the action upon precipitation is also prejudicial. When precipitation is upon zinc shavings, copper covers and fills the shavings and prevents further precipitation of either gold or silver, thus causing a huge waste of zinc. This difficulty may be partly obviated by using a smaller quantity of zinc for precipitation, thoroughly covering it with lead acetate, or with a lead precipitate, before precipitating the copper-bearing solution, and by shaking the zinc so as to dislodge the copper already precipitated, every day. This process must be attended to with great regularity, otherwise the copper will form a hard, compact coating upon the zinc, and render it useless for further precipitation. If it were possible to precipitate the dissolved copper and return to the solution all the cyanogen used in dissolving the copper, these great objections to its presence in ores would not be found.

Mercury is Destructive of Zinc.—Another element which presents serious difficulties in cyaniding is mercury. It has generally been stated that when this element occurs in a native state, it is not easily dissolved by cyanide, but this statement will have to be modified somewhat. It may be true that pure mercury is not readily soluble in pure cyanide solutions, but in working cyanide solutions which, as is well known, contain many other chemicals beside cyanide, even metallic mercury is rather readily dissolved. In old tailing where mercury exists as chloride, oxide, etc., it is dissolved quickly, even more quickly than gold or silver, and introduces complications. Its effect upon zinc is even worse than that of copper. It makes zinc shavings brittle, decomposes them until they fall to pieces, and induces an enormous waste of zinc. A little mercury in cyanide solution is beneficial to silver and gold extraction, as it precipitates the soluble sulphides in the same way that lead salts do, throwing down the insoluble sulphide of mercury. A great amount of mercury, however, has no particular advantage and has great disadvantages. It may be said with truth that both mercury and copper are much less objectionable where zinc-dust precipitation is used, because these elements are removed from solution immediately by being precipitated on the extremely fine particles of zinc as they occur in the dust. The waste of zinc is not apparently large, and is considerably less than when shavings are used. It is altogether probable that the same conditions would obtain, were aluminum dust used for precipitation in place of the zinc. There are, however, no definite facts which are well enough known to be stated as such. Many gold or silver ores do contain small amounts of mercury, either in the native form or as oxides in cinnabar ores. A notable example of the occurrence of mercury-silver amalgam is in the rich silver ores of the Cobalt district, of Ontario, Canada.

Lead and Zinc in Ores may Prevent Dissolution of Precious Metals.—Lead and zinc are objectionable, to some extent, to cyaniding processes. Lead sulphide, galena, is very slightly attacked by cyanide and presents no particular obstacles to cyaniding except through the contained sulphur. Should the galena contain silver or gold, the combination is, of course, prejudicial to good extraction, as it is not attacked by cyanide solution. Lead in the pure state, especially when finely divided, is rather readily dissolved by cyanide, and also when in the form of carbonate or oxide, but to a somewhat less extent, Zinc acts very much as lead does in cyanide solution the sulphide being only slightly attacked and the pure metal dissolved by cyanide solutions, in this case, much more readily than in the case of lead. As is the case with lead-sulphide ores, zinc sulphides are not prohibitive of successful cyaniding except where the silver value is held by the sulphide combination, in which case, of course, there is small chance for the cyanide to dissolve it.

Zinc-lead combination ores, in which the mineral occurs as sulphide, or, in fact, in almost any other chemical combination, present one of the complexities which has so far resisted successful cyaniding. There are in existence a great number of mines containing lead-zinc sulphide ores, which carry small amounts of silver, which, if each of these metals could be recovered separately, would be of considerable value. Of course, in cases where the amount of silver is large, or where either lead or zinc is largely predominant, these ores can be concentrated and the product smelted with success. When all the metals are low in quantity, it is not a profitable operation, however, as it is difficult, if not impossible, to recover lead and zinc separately from such ores by smelting methods. It should not be at all impossible to make such ores amenable to a modified cyanide process by some preparatory treatment which would make the lead and zinc soluble in cyanide solutions, and then to devise some process by which exhorbitant amounts of cyanide should not be lost. That such processes are, at least theoretically, not impossible is evidenced by the ammonia-cyanide treatment, whereby copper-zinc-sulphur ores may be treated with high recovery of each of the metals.

Arsenic and Antimony Objectionable but not Prohibitive.—Ores containing arsenic and antimony are generally supposed to be difficult to cyanide, but in many cases it is being successfully done. Neither arsenic nor antimony appear to react very strongly with cyanide, most objections to their presence coming from the decomposition products of the sulphide of either element. Arsenical ores usually give a good extraction when treated by the usual method, but in some cases may require extremely fine grinding to give up the maximum percentage of precious metals. Antimony acts very much the same, both elements exerting apparently, a reducing effect upon solutions, robbing them of their oxygen and forming decomposition products which are destructive to cyanide. Strongly alkaline solutions are believed to be more effective in treating ores containing these elements. In some cases where cyaniding cannot be practised directly on ores which contain arsenic and antimony, particularly in the sulphide form, some preparatory treatment may be resorted to, which will do away with the elements themselves or put them in some form which will not be particularly objectionable. Roasting or acid treatment, followed by alkaline wash, may be effective at times.

Tellurides now Successfully Treated.—Great difficulty was originally experienced with the attempt to treat telluride ores by the cyanide process. The ores of the Cripple Creek district, in Colorado, are an example of this class. It seems that the telluride, while itself not particularly destructive to cyanide solutions, forms a combination with gold which is almost indissoluble in straight cyanide solutions. Tellurium does not react with cyanide usually and perhaps only slightly in extremely alkaline solutions. Fine grinding of the ore seems to be beneficial in enabling an economically satisfactory extraction, but, for thoroughly satisfactory commercial results, bromo-cyaniding is usually resorted to. In fact, with Cripple Creek ores, this seems to have been the only means by which the gold telluride can be attacked and dissolved. As a matter of fact, in the present treatment of Cripple Creek ores, the process is mainly straight cyaniding, after which a light bromo-cyanide treatment is given to the material which has resisted the action of ordinary cyanide. The additional extraction gained by the bromo-cyanide treatment, while extremely small in proportion to that gained with straight cyaniding, is important on account of the low grade of ores being treated, which require the recovery of all possible values, in order that commercial success may be obtained.

Carbon or carbonaceous material is extremely objectionable in cyanide treatment, but, of course, this cannot be said to be an essential combination in any ore. As a matter of fact, it does occur in large numbers of accumulated stocks of ore and in tailing piles that have been standing a long time. In order that treatment of such material may be economically successful, the carbonaceous matter must be removed, or oxidising agents must be used to minimize its reducing effects.

Rare Forms of Sulphur May Be Objectionable.—It is well known that both sulphate of magnesia and sulphate of lime decompose cyanide solutions. The mineral kieserite, having the composition $MgSO_4 + H_2O$, is not rare and is but slowly dissolved in cold water. Hydrated sulphate of calcium, or gypsum, is also common, but it interests us more because sulphate of calcium is produced when the acidity in tailings and slime is neutralized with lime. Alumina also is taken up in cyanide solutions to some extent, so that probably some aluminum sulphate or double sulphate exists in some ores. An ordinary analysis of a sample of the ore to be treated by the cyanide process is nearly useless. Analysis showing the content of iron, sulphur, magnesium, etc., is no help whatever to the cyanide manager, as his knowledge must tell him what will take place when the finely crushed ore comes in contact with an ordinary cyanide solution. The analysis of an ore may not show magnesium sulphate, for instance, although small quantities of magnesia may be found, and sulphur may be believed to be in form of sulphide, when it really exists in some form much more objectionable.

Ferric arsenate is not a rare mineral. It occurs often in conjunction with limonite, a hydrated ferric oxide. Possibly these minerals when formed together are the result of the oxidation of arsenical pyrites. Scorodite has the formula $FeAsO_4 + 2H_2O$. It is insoluble in water, but soluble in weak hydrochloric acid. SO_2 in solution also acts upon it. Finely ground mineral containing scorodite, left in contact with a weak caustic-soda solution, will form arsenate of sodium and cyanide of sodium or potassium acts in the same way, accounting for the slow and continuous decomposition of cyanide solutions when treating ores containing arsenates. The arsenates formed are reduced in the zinc box and remain in the precipitated gold. They may be dangerous in acid treatment.

Some Manganese Ores Altogether Untreatable.—Ores containing manganese have been the subjects of experiments in connection with cyaniding for a number of years. It is a strange thing, but, nevertheless, a fact, that many ores containing manganese in various combinations can be, and are, cyanided with greatest ease, while other ores of apparently the same composition resist treatment. Many of these ores do not contain any rebellious elements except the manganese, but many of them have absolutely defied successful treatment. They can all be so treated as to give good results in the laboratory by roasting, acid treatment, or by some other method whereby the chemical combination of silver and manganese, which is the principal occurrence, can be altogether changed, but these methods are by no means practical and have never been applied. Those ores which have resisted practical treatment are few in number, but at present there seems to be no hope of overcoming the obstacle. No one knows just exactly what the trouble is, as other manganese-bearing ores are rather easily cyanided, or at least, do not present insuperable obstacles.

Nickel and Cobalt Ores Now Successfully Cyanided.—Nickel and cobalt combinations have generally been considered difficult to treat. While there is no information at the present time to change that opinion, it is certain that such ores, however difficult to beneficiate, can be and are being treated successfully at the present time. Only a brief survey of the work accomplished at the Cobalt district of Ontario in cyaniding silver-bearing ores is necessary to demonstrate practically that these extremely complex ores carrying nickel, cobalt, manganese, and many other elements are being cyanided satisfactorily. As has been noted[*], many of these compounds are soluble and may be separated from the solution after precipitation in zinc boxes. Undoubtedly they are, to a certain degree, detrimental to the solution, as they must consume cyanide, but the consumption has not been so great as to prohibit successful work.

Cyanide Field Has Widened Rapidly.—In summing up the discussion of ores which may be amenable to cyaniding, the first thing which attracts the attention of those conversant with the history of cyaniding since its inception, is the enormous width of the field at present compared to what was originally expected. At first only gold was considered susceptible to cyaniding, and at that only free gold in clean, siliceous ores. Then more refractory gold ores were undertaken and successfully treated, the process passing gradually to silver ores. The halogen compounds of silver, such as chloride, bromide, etc., were always easy to dissolve in cyanide solution and their treatment has never offered any difficulties. The sulphide, however, did offer an obstruction to successful treatment for a long time, and it may be said that it is only within the past eight or ten years that it has been successfully done. As a matter of fact, most of the low-grade silver ores in Mexico, which is being treated by the cyanide process, is in the form of sulphide, and certainly the production from that source is extremely great at this time. Even the concentrated silver sulphide, which almost always contains arsenic and antimony compounds, is sometimes treated by itself with cyanide solution, maximum extraction being often obtained at less cost than if the material were shipped to smelters and treated in the blast furnace. With the remarkable progress of the cyanide process in sight, it is difficult to say where the end will be, and certainly we may not legitimately place any limit upon the class of ores which will probably be successfully handled by it within the not very distant future.

Probably the most promising line of improvement is in the direction which will naturally suggest itself to every operator after having read E. M. Hamilton's article upon the use of aluminum dust in precipitation at the Nipissing mill, in Cobalt, Ontario. What naturally occurs to the mind is that even the baser metals contained in many ores may be commercially extracted if it can be assured that after removing the metal from the cyanide combination, the cyanide will be returned to the solution for further use. This field seems extremely promising and I expect to see much development in it within a short time. Another development, which may be more or less combined with aluminum precipitation, is the ammonia-cyanide treatment of copper, lead and zinc-bearing ores, of which large quantities exist, now altogether useless to its owners. There is no reason why successful treatment of these ores should not be accomplished, and I fully expect to see it done at some time."—HERBERT A. MEGRAW, *Engineering and Mining Journal*, Oct. 4, 1913, p. 629. (A. R.)

MINING.

WITWATERSRAND MINING CONDITIONS. — "The surface death-rate on the Witwatersrand mines remains fairly constant, the coloured rate being approximately double that of the white rate. In the annual report of the Mines Department of the Union of South Africa, Mr. R. N. Kotze, the Government Mining Engineer, says this indicates to some extent the relative capacity of the two races in looking after their personal safety, for on the surface works of mines the safety of the individual is very largely dependent on his own care, knowledge, and judgment.

On the surface and below the accident rate per 1,000 in 1912 was 3·9 as against 4·10 in 1911, and remarking in passing that there still remains a great deal of room for improvement, Mr. Kotze discusses the general question as follows:—

'Speaking generally, the following three factors appear to be the chief reasons for the continued high accident rate: (*a*) Bad health conditions; (*b*) 'Speeding up' to secure large outputs; (*c*) Anxiety to reduce working costs.

'(*a*) *Bad Health Conditions.* — Throughout the whole Witwatersrand the bad effects of miners' phthisis has now reached its climax; many of the best miners are dead, others have left or are leaving before reaching the worst stages of the disease, and a general panic has set in, preventing fresh miners from coming to the Rand from oversea. The mines now employ largely second rate types of miners of mixed nationality who are short of the standard of experience and efficiency necessary for dealing with huge outputs and for superintending the work of ignorant natives. A careless and inefficient miner discharged from one property easily obtains work at another mine, and wanders up and down the Reef putting in bad work on a number of mines and assisting to increase the accident rate by his carelessness in dealing with explosives, weakening the strata with injudicious use of heavy machine drills and heavy charges of explosives, and taking little heed of the dangers of badly supported hanging wall. In some of the deep-level mines a want of systematic artificial ventilation must also affect considerably the health of both white and coloured miners, causing them to be listless and non-observant, and reducing their efficiency and usefulness to a considerable degree. It does not seem possible that men who at the end of a day's hard work have to sit for an hour at a waiting station thick with dust and nitrous fumes, can day after day put in a good day's work, nor can a native who walks a thousand feet up a stiff incline every day be really physically fit for his work on the following morning. If health conditions were improved, better men would compete for the comparatively high mining wages on the Rand; both white and coloured would put in a better day's work, and it is reasonable to suppose that more intelligent white supervision would tend to reduce the accident rate. If these arguments were considered insufficient from a humanitarian point of view, it might be further urged that improved health conditions, by bringing better and more efficient miners, might tend to actually reduce working costs and cause larger profits to be made. Working under unhealthy conditions the efficiency of the unskilled miner must remain low, and a larger number of persons must be employed to do a certain amount of work than would be under normal conditions. As regards white miners, the skilled miner with all his faculties clear and in working order breaks more ground and uses less explosives than a mere labourer, and the difference between placing machine holes rightly or wrongly is very great indeed, and has a far-reaching effect not only on the immediate output from the mine, but on the future cost of timbering and supporting weakened excavations.

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[*] Nipissing high-grade Mill, *Eng. and Min. Journal*, Dec. 14, 1912.

'(*b*) *'Speeding up' to Secure Large Outputs.*—With increased stamps and mills, and increased depths to wind from, 'speeding up' has to be resorted to to secure a large output. From the consulting engineer down to the last joined gang of boys, every one is rushed, and so safety measures are apt to get scamped, stopes are not properly inspected or trimmed down, packs are neglected, and the contractor, urged to his utmost by those over him, takes risks which were never intended to be included among 'dangers inherent to the industry of mining.' We have cases where the white man gets killed or injured with his boys, thus sharing with them the result of his careless work, or where he strenuously endeavours to protect the natives under his charge, but in many instances these consequences fall only on the natives, the white miner being away at the critical moment or working elsewhere out of the actual danger zone. If speeding up and rushing mining work to extremes is responsible for accidents, the responsibility rests principally with the controllers of the mine who determine the output on a scale which is frequently in excess of the labour complement available and of the capacity of the mine and its shafts, and who have now to live up to the promises made to shareholders. The manager only increases his anxiety and trouble by forcing work to extravagant limits, the mine officials are averse to driving men and boys, and are incessantly worried with accidents to men and plant.

'(*c*) *Reduction of Working Costs.*—At the present time attempts to reduce working costs are carried too far, and economies in timbering, waste packing, pillar cutting, and other safeguards to underground workers all tend to increase the possibility of accidents. This Department has more than once had to insist on an increase in the safety measures mentioned above, which has certainly for the present added to the working costs of the mine. In capital expenditure also a large saving can be and has been effected by straining mining risks to the utmost, and working a large area with a single shaft. The decrease, however, in efficiency and safety resulting from this departure from one of the most important rules governing mining makes it improbable that any further attempts will ever be made to save money by mining with single outlet.

'*Falls of Ground.*—Out of 824 persons killed underground in the Transvaal mines in 1912, 290, or 35%, met their deaths through falls of ground, and against accidents due to this cause and to explosives the technical staff of the Department is paying special attention. The beautiful strong hanging wall of the Reef that we have heard so much about in the past has had its day and has served its purpose, and where formerly stopes were cleaned out and left bare for hundreds of feet, proper mining has now got to be undertaken with necessary filling, packing and timbering. As the deep levels get opened up and the pressure increases, the strain on small pillars and open stopes reaches the breaking point; areas give way on weak fault planes or over improperly supported excavations, pillars burst and stopes and levels crush, all causing increased expense at lower levels, and killing, injuring, and frightening the underground labourers. This crush in the mines with its attendant accidents can only be met by systematic filling, packing and timbering; methods that will no doubt involve increased expenditure at first, but which will repay themselves in the future.

'*Sand Filling.*—Sand filling is fortunately becoming more general along the Reef, but in many instances has been delayed too long; tremendous areas of worked out ground having to be caught up before the mine as a whole can feel relief from the general strain.

'*Packs.*—Packing is still unpopular, and is seldom done systematically. Packs are expensive, a trouble, and to be efficient must be packed tight with waste, at times difficult to get. Proper packs are absolutely necessary to reduce our accident rate from falls of ground in stopes, and the criticism that packs anywhere near the face of a stope are necessarily blasted to destruction means bad blasting, bad pack arrangements, or both. There does not seem to be any advantage in a few large packs built in the stopes of some of our mines where the pack takes days to build, and for some time only resembles a large pile of loose rock. The efficient pack for safety purposes is one that can be erected quickly under weak hanging to take the place of temporary timber as the stope advances, and which will absorb rather than resist pressure. Fill and pack, and keep the filling and packing close to the face, is the only safe system of working ground under pressure, as with the extensive and continuous mining that has been going on all along the Witwatersrand for the last twenty-five years, it is reasonable to suppose that the pressure in the mines is now greater than ever before.

'*Timbering.*—It is unnecessary to comment on timbering in drives and shafts. Levels have got to be kept open, and travelling ways cannot be allowed to cave, so every working mine has got to attend to shafts and levels or cease to exist. Timbering in stopes, however, leaves much to be desired. In deep stopes loaded stulls should be put in more frequently as close to the working face as possible. Speaking generally, the use of timber, more or less temporary, should be far more extensive for the protection of all persons working in or under working places, and temporary propping should be insisted upon. Temporary props get blasted out, but they have served their purpose and can be put in again when the next shift goes to work. The system of a contractor doing his own timbering is not a good one, as he is more intent on making a big cheque than on taking safety precautions, and a large gang of native labourers is dependent upon what safety measures he may choose to adopt. A timberman cannot, on the other hand, be in every stope at the beginning of the shift, but the contractor should be encouraged to put in such props as are temporarily necessary until the timberman arrives to make a more permanent job of it. The question of an easily accessible supply of timber for all stopes is, generally speaking, in an unsatisfactory condition and will have considerable attention from inspectors in the coming year.

'*Rock Bursts (So-Called Air Blasts).*—The rock burst only appeared prominently on these fields some years ago and, like artificial ventilation, was enshrouded in a good deal of mystery. It is generally recognised that these pressure bursts are the result of altered strains and stresses due to large excavations and deep mining and intimately connected with faults, fault planes, and other lines of weakness. Extra pillars, unless large enough to prohibit economic mining, would not appear to stop the danger, and at times increase it to a very marked degree. The remedy is to fill and pack stopes, and to keep this filling and packing as near the stope faces as possible, to avoid stripping ore along fault planes where air blasts are occurring, and to sheathe drive and stope pillars in areas already under pressure.

'*Resting.*—I am afraid that any suggestion for giving workings a rest will meet with very little sympathy on the Witwatersrand, but it is a recognised fact that ground under pressure becomes uneasy after blasting, and if left for a while settles down, the cracks and bursts dying away as the strains are rearranged. By increasing the number of working faces a deep-level mine would be able to allow a settlement after blasting in its worst workings and avoid some of the dangers attendant on trimming down a working near uneasy ground.

'*Machine Drills.*—A fruitful source of accidents from falls of ground is the indiscriminate use by incompetent persons of heavy machines for stoping. Many stopes are 'tender' and 'uneasy' and only require the jar of one or more heavy drills to bring large portions of the hanging wall down during working hours. Many persons do not appear to realise the bad effect that drills, even working at a distance of hundreds of feet, have on surrounding strata. Here again the 'rush' for rock compels the use of a dangerous uneconomical machine, as many a stope that could be worked safely with hand stoping and small blasts, becomes dangerous when stoping with machines and heavy blasting are carried on. The skilled miner with his better knowledge and training will only utilise his rock drill for breaking ground along the natural lines of weakness, calculating his holes to break a maximum of ground with a minimum of explosives, but the ordinary ganger of to-day, knowing the power at his disposal, often breaks indiscriminately into the foot or hanging wall, and ruins the supporting power of the enclosing strata.

'*Shaft Sinking.* — The danger in shaft sinking where from thirty to sixty persons may be crowded together in a shaft bottom has caused this Department to make every effort to minimise the risk of missfires in shaft sinking. In the new regulations a rule has been included making it compulsory to have special supervision in a sinking shaft with arrangements for blowing over the shaft bottom with air or water under pressure as an aid to detect missfires after blasting. In addition to these precautions, a sketch taken after blasting showing the position of all misfires and sockets is also compulsory. These precautions in shaft sinking have been adopted on several of the diamond mines for some years past, but whether their low accident death rate is due to these special safeguards entirely, or is caused by the absence of rushing methods and more careful mining, cannot as yet be ascertained. Certainly these new shaft-sinking regulations have met with a good deal of disfavour amongst managers along the Rand, partially, in my opinion, on the ground of the expense of the extra supervision, and partially from the suspicion that the Rand record in smart shaft sinking might be lowered by delays caused by these restrictions. A few representative persons appeared to be genuinely afraid of directing air or water under pressure into cavities containing fragments of explosives. To show that sinking accidents can be kept within a reasonable limit, we have 6,300 ft. of sinking done at the South Rand and Turf Mines shafts with a death rate far below the average for the Witwatersrand shaft sinking, all of which work was done at a fair speed and reasonable cost. As a prevention against missfires in shaft sinking especially, it has been suggested to employ a better fuse and a different method of priming and charging. This matter is worth attention and is being investigated. For the time being at any rate, inspectors have inaugurated a special examination and blowing over of all shaft bottoms, including a sketch, and the necessary extra attention that this method involves. This regulation will be given a fair trial and will be rigidly enforced.

'*Missfires—General.*—Accidents from missfires in stopes, rises, winzes, and development faces are far too numerous, and, where practicable, the regulation governing the extra precaution to be taken in sinking shafts has been extended to these workings. Numerous missfire accidents are still caused by the carelessness of the ganger in not properly examining his bench before rigging his machine and laying out new holes, and the Department is determined to adopt the most severe measures to prevent and punish carelessness of this description. At times the machine-man is killed or injured at his machine sharing the risks with his boys, but more often he is away from the danger point, and death and injuries fall on the native workers, who have been left to themselves during the dangerous period of starting to drill a new round.

'*Amalgamated Properties.*—If the comparatively recent large amalgamations of mining properties have brought about any of the anticipated advantages to the owners, their accident death rates leave a great deal of room for improvement, as shown by the following figures:—

Name of Amalgamated Property	Accident Death Rate per 1,000.
East Rand Proprietary Mines...	... 5·46
Crown Mines	... 4·28
Geldenhuis Deep...	... 4·05

'*General.*—If the foregoing text contains criticisms against the owners and administrators of the Witwatersrand mines, it is only fair to make any deductions possible in favour of their increased attempts to reduce the mining death rate. The past year shows an improvement of 0·2 per thousand over 1911, and we can analyse the possible cause. It is not contended that inspection has been any more rigorous or useful than it was in the previous year, nor have the attendant dangers in mining been any less. Miners have been no better than they were before, neither has the native labourer become more intelligent. This small death rate reduction must, therefore, be attributed to the united efforts of controllers and managers, who, in more ways than one, have given proof that they realise what large room there is for improvement. If the frank criticisms contained in these remarks draw their further attention, and secure, as in the past, their co-operation in remedying the evils indicated, a useful purpose will have been served.

'The Department received continued assistance from those interested in mining in its attempts to reduce accidents, and is quite willing to acknowledge that among both owners and administrators there are men who regard the protection of persons as essential in mining. But, unfortunately, there is the passive resister, the polite person who talks nicely, but will do nothing, and against this class of man, if he will not advance with the times, the Department intends to bring the full force of the law to bear."—R. N. KOTZE, *Science and Art of Mining*, October 21, 1913, p. 122. (A. R.)

TESTING ROCK DRILLS.—"The New Jersey Zinc Company, at their Franklin Furnace, N.J., mine, are paying unusual attention to the efficiency of the drills employed. A series of tests begun six or seven years ago, and still in progress, has resulted in greatly increasing the efficiency of the drilling and in bringing out some curious points in the performance of rock drills.

Every machine under consideration by the company is taken underground and its air consumption and drilling rate measured, this being done in a particular crosscut, where the rock is uniform. The holes are drilled for the most part vertically in calcite. The air pressure used is between 95 lb. and 100 lb., and, together with the temperature, remains fairly constant for any one test. The criterion upon which the comparison of the machines is made is a figure obtained by dividing the number of inches drilled per minute by the air consumption per inch. No machine is accepted for further trial unless it drills at least 7½ in. per minute on 60 cub. ft. of free air or less.

The important factors of convenience, repair expense, retention of original efficiency under wear, steel breakage, &c., are determined by further tests in the harder formation of the stopes. One anomaly developed in the tests is the fact that a machine will often drill as well or better with one wing of the cross-bit broken off. More curious still are the contradictory results obtained with the same machine under apparently identical conditions. A hole may be collared and a test run made; then with no further change than a new piece of steel, a second run may give a much higher rate of drilling in continuing the hole. The calcite is too uniform to permit of this being explained by a change in rock quality. A plausible theory is that the blow of the hammer and the anvil block on the steel is transmitted through the latter as a wave. Such a wave will be reflected when the bit transmits it to the rock in the hole bottom, and its return may oppose or coincide with the next blow of the hammer. When conditions are such that these rebounds reinforce the successive hammer blows, the maximum drilling effect is attained. If the rebound and the new blow exactly oppose each other the drilling power is appreciably cut down. The usual conditions will be a varying degree of interference with correspondingly different results. An attempt to analyse and control the phenomenon, however, disclosed the existence of too many variables, the chief of which are the length of the steel and its resiliency."—*Times Eng. Supplement*, October 15, 1913, p. 23. (E. P.)

Wireless Telephony in Mines.—"The first installation of wireless telephony in a coal mine in Great Britain has, *The Times* says, just been fitted up at Dinnington Main Colliery, South Yorkshire, and is giving satisfactory results. The system is the invention of Herr J. H. Reinecke, of Bochum, Westphalia, and has already been adopted in German collieries. Its introduction into this country is due to Mr. Maurice Deacon, managing director of the Dinnington Main group of collieries. The instruments are similar to those in ordinary use."—*Page's Weekly*, September 19, 1913, Vol. xxiii, No. 471, p. 365. (J. G.)

Cement Injection for Consolidating Rocks.—"At the Trelys Colliery, in the department of Gard, it became necessary to drive through measures that had become much dislocated in consequence of a violent outburst of carbonic-acid gas. A cement stopping was put in, behind which 25 tons of cement 'milk,' or wash, composed of 10 parts of cement to 15 of water, were injected, under the pressure of compressed air, through pipes of about 2 inches (40 to 60 millimetres) diameter, flattened at the end, and drilled with small holes. A fortnight after the injection the cement had set sufficiently to permit of the continuation of driving without fear of a fall at the face."—M. de Marillac (*Soc. Ind. Min., Comptes Rendus Mensuels*, 1912, p. 251-255), *Transactions Institute of Mining Engineers*, July, 1913, p. 744. (E. P.)

MISCELLANEOUS.

Mine Sanitation.—"The old wooden dry, of a comparatively few years ago, is now being replaced by the modern one with ample hot and cold water, individual basins or buckets, lockers for the street and mine clothes, and drying and ventilating systems which effectually take care of the wet clothing of the miner. The plan of suspending the wet clothes by means of a rope or chain, with individual lock, has much to commend it. The shower bath, which is a part of all the modern drys, has made it possible for the men to bathe who have not the proper facilities at home. Their constant use has demonstrated their value.

The introduction of water closets in the drys has not always met with complete success, but in the majority of cases it is working very advantageously and improving the underground conditions.

Dry closets should be placed in every main level and they should be provided in proportion of one to every 25 men. These closets should be made of wood or iron of a size to be conveniently handled by two men. They should be taken to the surface and washed out with the hose when emptied. Lime should be kept convenient to the boxes and used regularly. There is no more fruitful source of disease than the human excreta left in the mine workings.

The water that the men drink should have the most careful attention and under no circumstances should the men be permitted to drink water that is not known to be pure.

Anything that will lessen those diseases frequent among our miners, such as tuberculosis, typhoid, and other contagious or communicable diseases, should be carefully considered. There should be a more rigid quarantine in cases of contagious diseases, and it is desirable that the mining companies co-operate more fully with the health officers and that such regulations be made and enforced so that our unenviable record in contagious diseases, especially in diphtheria and scarlet fever, may be greatly improved upon.

Owing to the relation that the physicians bear to the mining companies, they hesitate to suggest things which might appear as criticism of existing conditions or methods of mine management, and should therefore be given full authority to correct undesirable conditions. I believe we could get better results in this matter of sanitation by employing the physicians, at a stated added remuneration, to make periodical inspection of the entire properties.

I believe that there is a duty that the mining companies owe to men employed by them and the community in which they live. One of the most efficient ways of fulfilling this obligation is in providing satisfactory homes, with healthful surroundings for the men and their families. Any work of this kind must be wisely done and in such a way that the men will be led to co-operate in it so that there will be no suggestion of paternalism in its methods or manner of enforcement. The importance of providing houses for rental is generally recognised, but after the house is provided it is just as essential that its condition be not neglected. We have seen houses with no provision for waste water, which must be thrown out upon the ground. The only good things that may be said of this is that it usually provides a good place for the propagation of fish worms, but it also assists in the propagation of other less desirable things.

The old boxed-in cupboard sink may yet be found which is a place for refuse and vermin, and is so frequently a rotten, dirty, slimy hole. If there are such, let them be torn out.

The drainage around houses and other properties requires careful attention. Ditches should be dug and kept open even if the boys fill them up, and no stagnant water left to breed mosquitoes and infection.

Garbage and refuse should be deposited in covered receptacles, preferably galvanised iron cans, and where such services are not rendered by the city or village, the companies should provide for its regular collection and disposal not less frequent than once each week. It has been found feasible in some places to have the occupants of the premises purchase the cans, which are supplied at wholesale rates, or through the company at cost, with the understanding that proper collection is assured. Rubbish should not be dumped around promiscuously, but deposited in a designated place, collected, and disposed of. Few of the men have time or facilities for economically doing this, and if not looked after by the city or the company it will not be taken care of at all satisfactorily. A refuse burner, costing very little, can be so located as to make the disposal of rubbish a simple matter and comparatively an inexpensive one.

It is desirable in many cases to permit the keeping of a cow or horse, but provision should be made for the care of the manure. It should not be allowed to accumulate from month to month, but should be removed promptly. It also should be treated with chloride of lime, or with a kerosene-carbolic acid mixture.

The companies should encourage the beautifying of home surroundings, but it is even more important to see that the sanitary conditions are what they should be. In those localities having no sewer connections, the question of outhouses is a serious matter. Too often the outhouse is set on the ground with no proper receptacle for the human excreta which often spreads over the ground, even if a pit is dug. This is frequently allowed to collect through the season, thus becoming a menace to the family. Often the houses are more or less open; vermin enters without hindrance; the ever-present disease-carrying fly freely finds access, goes from there to the house, feeds on the baby, and has even been known to be on the household food. We have just begun to appreciate the danger from these pests. The companies which have already set us an example in their campaign against the fly should be congratulated, and it should also stimulate the rest of us to make an equal effort. These outhouses should be provided with a proper receptacle which should be emptied at regular and not to separate intervals, and properly cleaned. Either earth, slaked lime or chloride of lime should be constantly used.

The value of the physical inspection of school children is not only just beginning to be appreciated. It should be in effect in all of our communities. It is a health precaution of great merit.

The most efficient agency in extending the benefits of this work of sanitation is the visiting or public-health nurse. A number of mining companies in the Lake Superior region have already introduced this service with much success. No other person can have the same opportunity of reaching the homes and the members of the familes and assisting in the understanding of the benefits to be derived from fresh air, cleanliness and other sanitary precautions. The importance of fresh air cannot be emphasized too strongly.

The advantage of all this is a better home life, the prevention of serious contagion, and the men in better condition for work. Anything that tends to the more regular work of the men is well worth consideration."—W. H. MOULTON, *Lake Superior Mining Institute*; *Mining and Engineering World*, Oct. 25, 1913, p. 739. (A. R.)

WEIGHING LIGHT MATERIAL ON ORDINARY SCALES.—"It is often desirable to weigh with some accuracy articles or material of light weight when only the ordinary platform scale, used for heavy weights, is at hand. In such cases a method devised by W. B. Crowe (*Met. and Chem. Eng.*, June, 1913) is available and is a great convenience.

The method is based on the relation of the beam weight to the platform weight. In many ordinary scales this proportion is 1 : 100, 1 lb. on the beam balancing 100 lb. on the platform, and this proportion can be used as an example, any other proportion being treatable in the same way. To find the weight of an article too light for weighing in the usual way on the platform, it is attached instead of the beam weight and a convenient material placed on the platform until equilibrium is reached. Then the article is detached and the material is weighed as usual. The weight of the material used, divided by 100, equals the weight of the light article hung for the beam weight. With scales accurate to 0·25 lb. on the platform, the sensibility to objects on the beam would be 0·04 oz., or 1·133 gm. This ingenious expedient should be of use in metallurgical plants, as by it small amounts of precipitate, bullion, flux, etc., can be weighed with sufficient accuracy for ordinary purposes."—*Engineering and Mining Journal*, Aug. 23, 1913, p. 355. (A. R.)

ARTIFICIAL COAL FROM A WOOD STEAMER.—"The author has investigated a hard brownish black deposit which formed the accumulation of several years in the bottom of a wood steamer in a brown mechanical wood pulp mill. The deposit resembled in part lignite, in part anthracite and in part consisted of practically unchanged wood particles. There appears no doubt that the deposit was formed by the carbonization of wood by long-continued steaming under pressure. In elementary composition, appearance, behaviour on dry distillation and reactions with alkalis and acids the greater part of the material closely resembled lignite. The composition of the ash was different from that of lignite, a large amount of Al silicate being present, but this had doubtless been derived from the cement lining of the digester. The formation of lignite by partial carbonization of woody material due to steaming at 75 lb pressure for extended periods is in direct agreement with the work of Bergius (*C. A.*, *6*, 3322), who converted cellulose into a coal having all the properties of anthracite by heating from 8 to 64 hr. at very high pressure and temperatures. It follows that by treatment along similar lines the conversion of lignite to anthracite could most probably be speedily effected."—EMIL HEUSER (*Z. angew. Chem.*, *26*, *Aufsatz*, 393-6), *Chemical Abstracts*, Vol. 7, No. 18, September 20, 1913, p. 3215. (J. G.)

INTERNATIONAL CONGRESS OF MINING.—"One of the largest of the great scientific and industrial congresses is to be held in London in the early part of June, 1915. This is the sixth international congress of mining, metallurgy, applied mechanics, and practical geology. These congresses take place at intervals of five years, and the last, which was brilliantly successful was held at Düsseldorf in 1910, previous congresses having been held in Paris and Liège. The attendance at the Düsseldorf congress was more than 2,000, and it is anticipated that the attendance in London in 1915 will be equally large. An influential committee has been formed to make the necessary arrangements, and the movement is being actively supported by the University of London, Imperial College of Science and Technology, Geological Society of London, Institution of Mechanical Engineers, Iron and Steel Institute, Society of Chemical Industry, Institution of Mining Engineers, Institution of Mining and Metallurgy, Institute of Metals, South Wales Institute of Engineers, Cleveland Institution of Mining Engineers, West of Scotland Iron and Steel Institute, Staffordshire Iron and Steel Institute, Sheffield Society of Engineers and Metallurgists, and by numerous firms interested in the various industries represented."—*Nature*; *The Valve World*, July 1913, p. 223. (A. R.)

Reviews and New Books.

METALLIC ALLOYS: THEIR STRUCTURE AND CONSTITUTION. By G. H. Gulliver. Second Edition. Charles Griffin & Co., Ltd., London, 1913. Pages 409 + vi., including Index. Chapters XI., with 310 illustrations. Price 10s. 6d. net.

"The rapid advance in the study of alloys makes the re-appearance of this book of special interest at the present juncture.

It is divided into 11 chapters, which deal with investigation, physico-chemical equilibra binary alloys, transformations which occur in solid metals and alloys. Structure of metals and alloys, the bronzes, brasses and other copper alloys, and alloys of iron. The final chapter is one of considerable interest as it deals with the microscope in engineering practice. The illustrations are particularly good, and the tables of references to standard works on each subject will be of great value to the student and chemist. This second edition has been greatly enlarged, and largely re-written. It should be in the hands of all those interested in metals and their working."—*The Chemical World*, Nov. 1913, p. 360. (J. W.)

THE ELEMENTS OF CHEMICAL ENGINEERING. By J. Grossmann, M.A., Ph.D., F.I.C. With a preface by Sir William Ramsay, K.C.B., F.R.S. Second edition, revised. Pp. 152. Charles Griffin & Co., Ltd., 1913. 3s. 6d. net.

"We are pleased to see that a second edition of Dr. Grossman's useful little manual has been called for. Students' text-books of chemistry are plentiful, but text-books of chemical engineering are few, and 'Elements of Chemical Engineering' should be in the hands of all students who are leaving the lecture-room for the works. The author's method of showing the transition from laboratory apparatus to the plant of the chemical works is very effective."—*Chemical Engineering and The Works Chemist*, Sept. 1913, p. 286. (J. W.)

THE NICKEL INDUSTRY: WITH SPECIAL REFERENCE TO THE SUDBURY REGION, ONTARIO. By A. P. Coleman, Ph.D., Canada Department of Mines, Government Printing Bureau Ottawa.

This report consists of 206 pages, with 63 photographs, 18 drawings and 9 geological maps.

"*Nickel Ores of Cape Colony* are referred to on p. 122:—'Nickel ores were found at Insizwa, Cape Colony, many years ago, but have only recently attracted much attention. As described by Mr. Du Toit of the Cape Survey, they resemble rather closely the Sudbury deposits, consisting mainly of pyrrhotite with chalcopyrite and pentlandite, though a few other nickel and copper minerals occur also, as well as the metal platinum. The ores accompany gabbro or norite which formed a basin-like sheet as at Sudbury. The sheet is 2,000 or 3,000 ft. thick.

Two trial shipments of the ore, of some five tons each, sent to Messrs. Johnson, Matthey & Co., in England, for examination, are reported to run as follows:—

	1.	2.
Copper	3·40%	3·50%
Nickel and cobalt ...	4·90%	5·25%
Gold (per ton) ...	6 gr.	6 gr.
Platinum (per ton)	2 dwt. 12 gr.	12 gr.
Silver (per ton) ...	10 dwt.	12 dwt.

(Dept. of Mines. 15th An. Rep. Geol. Com., Cape of Good Hope, 1910, pp. 111-142.)

If the ore represents the average of the deposits, they seem to be of high grade, and if the ore is present in large amounts the region may be of importance."

Prof. Coleman visited South Africa with the British Association, when some of us had the pleasure of meeting him.—(J. W.)

Carr, William Kearney. Matter and Some of its Dimensions (Library of Living Thought). Cr. 8vo., pp. 122. *Harper* Net 2s. 6d.

Classeo, A. Quantitative Analysis by Electrolysis. Translated from the 5th German edition by W. T. Hall. 8vo. *Chapman & Hall.* Net 10s. 6d.

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Rector, F. L. Underground Waters for Commercial Purposes. Cr. 8vo. *Chapman & Hall.* Net 4s. 6d.

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Royal Society. Celebration of the 250th Anniversary. July 15-19, 1912. 4to., pp. 176. *H. Milford.* Net 6s. 6d.

Scanlen, J. J. The Mutilated Hand and the Workmen's Compensation Act, 1906. Having special reference to "Missing" Fingers. Cr. 8vo., pp. 98. *Scientific Press.* Net 2s. 6d.

Scientists' Reference Book and Diary (The) 1914. 18mo., pp. 132. *Letts.*

Smith, Ernest A. The Sampling and Assay of the Precious Metals. 8 vo., pp. 476. *C. Griffin.* Net 15s.

Stanley, H. Practical Science for Engineering Students. Cr. 8vo., pp. 174. *Methuen.* 3s.

Stewart, A. W. A Manual of Practical Chemistry for Public Health Students. Especially arranged for those studying for the D.P.H. Cr. 8vo.,, pp. 82. *Bale.* Net 3s. 6d.

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Tilden, Sir William A. The Progress of Scientific Chemistry in Our Own Time. With Biographical Notices. 2nd Edition. Cr. 8vo. *Longmans.* Net 7s. 6d.

Webster, R. W. Diagnostic Methods, Chemical, Bacteriological and Microscopical. 3rd edition, revised and enlarged. 8vo. *H. Kimpton.* Net 18s.

Wimperis, H. E. The Principles of the Application of Power to Road Transport. Cr. 8vo., pp. 146. *Constable.* Net 4s. 6d.

Year Book of the Scientific and Learned Societies of Great Britain and Ireland. 1912-1913. 8vo. *C. Griffin.* 7s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 555/13. Albert Williams Daw (1), Zacharias William Daw (2). Improvements in hammer rock drills and in percussive hammers and similar tools. 20.11.13.

(C.) 556/13. Dr. Carlo Rossi. Improvements in or relating to electrodes for use in the production of nitric oxide. 20.11.13.

(P.) 557/13. Harry Ernest Rygor (1), Abraham Isaacs Wiesholtz (2). Improvements in safety devices for skips running in inclined shafts. 20.11.13.

(C.) 558/13. Ernest Loesser. Improved apparatus for making dops for holding precious stones to be ground. 20.11.13.

(C.) 559/13. Ernest Loesser. An improved dop for holding precious stones to be ground. 20.11.13.

(C.) 560/13. Ernest Loesser. Apparatus for grinding and polishing precious stones. 20.11.13.

(C.) 561/13. Ernest Loesser. Apparatus for grinding and polishing precious stones. 20 11.13.

(C.) 562/13. John Emery Bucher. Method of producing metals. 20.11.13.

(C.) 563/13. John Emery Bucher. Improvements relating to the fixation of nitrogen. 20.11.13.

(P.) 565/13. Frederick Victor William Swanton. Improvements in power producing apparatus. 20.11.13.

(P.) 566/13. James Snodgrass. Improvements in the concentration of gold ores. 22.11.13.

(P.) 567/13. Charles Young Weir. Improved haulage jockey for trucks for use in mines and the like. 22.11.13.

(P.) 568/13. Anthony William Richard Schieder (1), William Spence Wilson (2), George Innes Smith (3). Improvements in apparatus for a portable chute or stope box for use in underground workings of mines or the like. 24.11.13.

(P.) 569/13. Otto Stuhang Tonneson. Improved means for extracting jumpers, drills and such like. 24.11.13.

(C.) 570/13. Robert Blakoe. Improvements in and connected with the wheels of cycles, motor cars and other vehicles. 25.11.13.

(P.) 571/13. Arthur Joseph Seguin Fisher. Improvements in percussive apparatus. 26.11.13.

(C.) 572/13. Fearless Brush Company, Ltd. (1), Alfred Gerhold (2). Improvements in and relating to brushes. 28.11.13.

(P.) 573/13. Bertram Campbell Travers Solly. A new and improved clamp and nozzle holder for water spray or jet to be used with rock drilling machines. 28.11.13.

(C.) 574/13. Charles Washington Merrill. A precipitant for recovering metals from cyanide solutions. 1.12.13.

(C.) 575/13. Charles Washington Merrill. Process for precipitating and recovering valuable metal bearing materials from solutions. 1.12.13.

(P.) 576/13. Samuel Thomas Tregaskis. Improved support for light power driven hammer drills. 2.12.13.

(P.) 577/13. Robert Caldwell. Improvements in means for locking skips to prevent their premature tipping. 2.12.13.

(P.) 578/13. John Gillis Munro. An improved portable machine for the treatment of diamondiferous gravels and soils. 3.12.13.

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337.

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NOTICE.

The declaration of a General Strike throughout South Africa by the Transvaal Federation of Trades on the 13th January was followed immediately by a proclamation by the Government declaring Martial Law in force in certain districts. This affected the Witwatersrand in particular, and as a consequence the Council were unable to call the Ordinary Monthly Meeting of the Society in January. In order that members should not be deprived of their *Journal* owing to this trouble, it has been decided to print the paper which was to have been read at the January meeting, together with such discussion on previous papers as was available, as well as the other matter usually issued to members.

The next monthly meeting of the Society will be held on Saturday, the 21st February, 1914.

THE PREVENTION OF DUST IN UNDERGROUND WORKINGS.

By B. C. Gullachsen, M.I.M.E. (Member).

The prevention of dust in mine workings is a subject of paramount importance, especially so in the case of the mines on the Rand. As is well known, the terrible scourge of Miners' Phthisis is due to the dust created by mining operations being breathed into the lungs, and it therefore follows that in order to prevent the disease mining operations must be carried on in such a manner that as little dust as possible is produced. That dust will be entirely eliminated from the underground workings is, of course, impossible. There can be no doubt that the amount of dust present in the air of most mines has, during the last three or four years, been considerably reduced, but I am of opinion that considerably more could be done to further prevent it in very many of the mines. In order to prevent dust there is only one efficient remedy, and that is the application of water in a systematic and effective manner. Strange to say, text books on mining have little to say on the important subject of the watering of mine workings, except in the case of coal mines, where, of course, watering is carried out for the purpose of damping the coal dust, and thus preventing the extension or intensifying of an explosion of gas. The principal object to be aimed at in the use of water underground is to utilise it in such a way so as to eliminate the possibility of dust being carried into the air from its point of origin ; and, secondly, when that is impossible, as in the case of blasting, to remove the dust from the atmosphere as soon and as effectively as possible. In connection with the use of water underground the Mines Regulations Act (Transvaal) requires in Regulation 101 (1) that:—"No person shall in the drilling of holes use or cause or permit to be used any percussion machine drill unless a water jet or spray or other means equally efficient is provided and used so as to prevent the formation of dust by drilling, and unless the floor and sides of the working place to a distance of at least twenty-five (25) feet from the face be kept constantly wet." Regulation 101 (2) requires that:—" No person shall in any part of the mine move any broken rock or ground or cause or allow the same to be moved if such rock or ground is in a dusty condition, unless and until it and the floor, roof and sides of the working place to a distance of at least twenty five (25) feet have been effectively wetted and kept wet so as to prevent the escape of dust into the air during removal." Regulation 106 (34) requires that the ganger shall in all development faces (except in the case of a winze being worked single shift) immediately after lighting up put into action the water blast which he shall previously have tested. If as a result of such test the water blast be found to be not in order no blasting shall take place."—The above three regulations, if carried out to the letter, would undoubtedly cause a great improvement, and we must look to the miners of the Rand themselves to see that these regulations are effectively carried out; on the other hand the management is responsible for causing all plant, material and other things necessary for compliance with the requirements of these regulations to be provided and maintained in good order and repair. Regulation 158 (6*a*) requires that the manager of a mine shall "Provide or cause to be provided an adequate or constant supply of water which is clear and odourless at every working place which is not naturally wet. Such supply shall be sufficient for effectively wetting the broken ground and for preventing the formation of dust caused by drilling operations. Such water shall be supplied in metal pipes not less than 1 inch in diameter at a pressure of not less than 30 lbs. to the square inch at each working place when all sprays and jets supplied from the same pipe are working. Such pipes shall reach to within fifty (50) feet from the face and from there a sufficient length of hose shall be provided and used to bring water up to the face. (*b*). Cause the surface in all working places, travelling ways and shafts which are not naturally wet to be kept wet as far as practicable. (*c*). As far as practicable cause such old stopes as tend to shortcircuit or add dust to the ventilating current to be closed off and cause the ventilating current to be circulated along the working face. (*d*). Provide in shafts which are not wet ring sprays at suitable points to prevent dust being carried with the air current. (*e*). Provide or cause to be provided a water blast as near as practicable to all development faces (except winzes being worked on single shift)."

Having pointed out what is required by the Mining Regulation for the prevention of dust, I can now proceed to the consideration of the more practical side of the question.

Watering Stopes.—All stope faces and the broken ground lying in the stope must be watered before drilling or shovelling may be commenced. In the deep level mines where the temperature of the air is fairly high, it naturally follows that stopes watered at the commencement of a shift are quite dry again before the end. In such mines it seems to me to be very important that the stopes should be watered a second time just previous to blasting. It is specially necessary that the stopes should be wet just at the time of blasting in order to prevent the dust that would otherwise be present being raised and set in motion in the atmosphere of the mine; besides this, if the roof, floor and sides of the stope are wet at the time of blasting, they will tend to collect a large quantity of the very fine dust caused by the blasting. Very little dust should be caused in stopes by the *drilling* of holes seeing that in nearly every case the holes are wet ones. When a hole is being "collared" dust is often produced, as the natives will often drill the holes a few inches before commencing to use water. In hammer stopes the new Regulations require that a "swab" shall be used around the drill at the collar of the hole when drilling dry holes, but how the "swab" is to be held in that position is not mentioned, and in very many cases I fail to see how it can be done unless special appliances are used for the purpose.

Watering Development Places.—In all development places the face must be in a wet condition before drilling can be commenced, and the floor, roof and sides must be wet for a distance of 25 feet from the face. Personally, I hardly consider the distance adequate, as when blasting takes place the air is driven forward at a high velocity and all dust outside the 25 feet will be raised in a cloud and remain in the air for a very considerable time. The use of water blasts would only partly prevent this dust, seeing that they are as a rule placed as near the face as possible. Of course, if a second water blast was brought into use further out in the drive, it would no doubt catch the dust driven forward by the force of the explosion. The objection to having either a second water blast, or one at any considerable distance from the face, is that the stream of air and water tends to act as a cork in the neck of a bottle and prevent the escape of smoke and fumes along the drive, unless the water blast is pointed towards the shaft. As with stopes, I consider it highly necessary that all development places should be watered just previous to blasting, and certainly in dry places the roof, floor and sides should be damp for a considerably greater distance than 25 feet from the face.

Rises.—More dust due to drilling is caused in rises than in any other development place, this, of course, being due to the fact that all the holes are dry ones. Spraying the collar of the hole is partly effective, but often results in the jumper becoming fast. A small pipe inserted into the hole at the side of the jumper, and through which a stream of water is passed to the end of the hole, has given good results, but if they are not carefully looked after they are liable to become choked up with small stuff or flattened out, thus preventing the flow of water.

Drives.—Drilling in drives should not be the cause of much dust, as only two holes in the round are dry ones. Water, if thrown, or, better still, sprayed on to the hole as soon as collaring commences, will prevent the escape of dust caused by that operation, and in the case of all holes, with the exception of the "back" holes, water can be poured into the holes, thus entirely preventing the escape of dust. In order to prevent the dust caused by the two dry "back" holes, small pipes may be inserted into the holes and water injected by that means. Atomisers may be attached to the air chest of the machine which project a fine spray of water into the collar of the hole, but as they tend to fill the drive with a form of Scotch mist they are objectionable and may even be harmful. Wherever dry holes are to be drilled it seems to me the only real effective remedy is to have machines using hollow steel, through which a constant supply of water is forced right to the end of the hole, as by this means no dust can possibly be formed. Such a machine as the Leyner-Ingersoll, which works on this principle, is giving excellent results, but it remains to be seen how it will stand the wear and tear and the use of what is often slightly acid water.

Winzes.—In winzes, as all holes are wet ones, no dust should be produced if care is taken to use water as soon as collaring the hole commences. The same remark applies to shaft sinking.

The Water Supply.—According to the Mines Regulations the supply of water shall be clear and odourless. In order to carry out this requirement it would, on very many of the mines, be necessary to use water from the town mains. At most of the mines on the Rand mine water is being used, and I am giving no secret away when I say that such water in the majority of cases cannot possibly be considered clear or odourless. Personally, I cannot see the necessity of using drinking water for watering a mine, and in most cases little harm could possibly result from using the ordinary mine water. In most cases the mine water is acid, and, in order to save the pumps and water pipes, lime has to be added in order to neutralise it. Such water is seldom clear and may often have an odour, but certainly no injurious effects could result from its use. The fact of the water remaining slightly acid appears to be, to a certain extent, an advantage, as the acidity of the water will no doubt cause the destruction of any bacteria with which it comes in contact. Would it not be an advantage to add to the water some cheap form of disinfectant? I hope that our medical members will have something to say on this side of the question.

Another point about which I trust the medical members will give us information is the humidity of mine air and its danger to the health of the miners. In our deep level mines, where the air temperature is so high, more water will naturally have to be used to keep the workings, etc., in a wet condition, and the higher the temperature of the air the higher will be its point of saturation; thus, where large quantities of water are used, the humidity of the air is sure to be great, which naturally will have a detrimental effect upon the work and health of those employed in it.

Fixed Sprays.—It appears to me that this is one of the methods for preventing the carriage of dust in the mine air which does not receive sufficient consideration. I refer to sprays which as a rule are fixed overhead in drives. These sprays, if placed in the principal tramming levels, are constantly being turned off, and even broken, by the tramming boys, who rather naturally object to having an enforced cold douche each time they pass to or from the stopes. I consider that an excellent place for fixed sprays is at the top of stopes if the ventilating current is upcast, or at the bottom, say, in the mouth of the boxhole nearest the face, if the ventilation is downcast. When a stope is first being opened up there is, of course, only one outlet at the top, and all dust coming up the stope must pass this point: thus a spray fixed in that position would catch all the dust and fumes caused by blasting. As the stope becomes extended more sprays would be necessary, but when the faces are some considerable distance apart it would be sufficient to have only one spray at the top of each face, at the nearest manhole to the face. The direction of the ventilating current would, of course, be the chief thing to be studied when deciding the position of the sprays. With sprays fixed in the positions I have suggested, it would only be necessary to turn them on just prior to blasting.

Pipe-Fitters.—With regard to pipe-fitters, a sufficient number should always be employed to cope efficiently with the work. In most cases a pipe-fitter is allowed two or three boys to assist him, but it seems to me that in many mines he might with advantage have more, as it often happens at the beginning of a shift that the water service pipes are broken or out of order in more than one place, with the result that each place has to wait its turn before it can be repaired. This either causes delay in starting machines, or what more often happens, drilling and lashing are started before the water service is got into order. With more boys the pipe fitter could hurry round his section, see what was required, and start his boys on each job as required. Now that it is a matter of such great importance, would it not be advisable to send the pipe-fitters down the mine an hour or so previous to the shift, in order to have the water service in perfect order on their arrival. No excuse could then be found for not using the water as required.

Underground Dams.—If dams are used underground for the storage of water, they should wherever possible be constructed in duplicate. Trouble is often experienced when a dam becomes partly filled up with sludge, which the inflowing water stirs up and causes very dirty water to go into the water-service pipes. Where two dams are built, one only is used until the accumulation of sludge becomes troublesome, when the water is turned into the other dam while the first one is being cleaned out.

Ore-Passes and Ore-Bins. — On many mines more attention might be paid to the adequate watering of both ore-passes and ore-bins, which are often a fruitful source of dust. Ore-passes should always be kept wet, and to do so sprays should be provided under the grizzly-bars. It is not sufficient that the rock tipped into an ore-pass should be wet, as the rock in passing down an ore-pass strikes against the sides and is broken up and causes dust. Ore-passes are also often main airways, and any dust generated in them is carried away by the ventilating current and passes into the mine workings. For this reason sprays should always be fixed at the upcast end of ore-passes. The same remarks apply to shaft ore-bins, especially so in downcast shafts, where any dust set at liberty in the shaft is carried by the air into the workings. If the rock drawn from a bin is not wet a large quantity of dust is caused during its passage down the shoot and into the skip. Ore-bins should be provided with sprinklers in order to keep the rock wet, as even if the rock is wet when tipped into the bin, it may become dry before it is drawn from it, as when it has remained there during a week end.

Labour.—The employment of what is practically an uneducated savage is a large factor militating against the prevention of dust in the mines on the Rand. The native does not like water, as anyone with only a brief acquaintance with him can assure you. The result of this is that he does not care to use any apparatus for the prevention of dust which may mean a wetting for himself. This is especially the case in rises, where in the majority of cases a native will only use water as long as someone is there to enforce its use. Natives should be fined more frequently for not using water as directed, and notices should be posted in the compounds stating the danger resulting from not using it. The educated natives in the compounds should be given short papers dealing with the cause of phthisis, and instructed to read them to the other natives.

The Contract System v. Day's Pay.—It may seem rather far-fetched to say that the contract system is the cause of considerable quantities of dust, but I think underground men will bear me out in this. If the water service is off, how often will a contractor be willing to wait until it is in order before commencing drilling? In nine cases out of ten he takes the risk, as he wants to get his "round" drilled during that shift; dry holes are drilled and dust produced. I will be told that efficient supervision will prevent this, but shift-bosses cannot be everywhere at the same time, and a contractor on the bottom levels of a section knows that it is very unlikely that the shift boss can be down to his level until late in the shift.

Molasses.—A mixture consisting of 50% solution of molasses and water has been used on the City Deep, Nourse Mines, and in a lesser degree on other mines of the Central Mining Group. This mixture is sprayed on the roof and sides of development places, where it forms a slightly sticky coating, all dust settling on this surface is retained in a damp condition, thus preventing the usual cloud of dust raised by the concussion of blasting, whereas if water alone had been used the dust would have dried up after an interval of only a few hours, and left to be disseminated at the next blast.

I am not aware if any other mines besides those mentioned above are using this mixture, but if it has proved beneficial on these mines, why is it not adopted elsewhere?

In coal mines, calcium chloride has been employed for keeping coal dust in a damp condition. In some cases a solution of the chloride is sprayed on, and in others the salt itself sprinkled on the coal dust, which consequently becomes damp owing to the moisture being taken from the air by the calcium chloride. I am not aware if calcium chloride has been tried on any of the Rand mines.

Measurement of Dust.—I believe difficulty has been experienced in obtaining a machine which will give accurate results, but even if an error of 5% is likely to occur, the information would still be very valuable, as it would give us at least an idea of what improvements were taking place from time to time. That a machine would give satisfactory results in the measurement of dust was proved by Mr. A. McArthur Johnston, who conducted a series of experiments on the Simmer Deep from Sept., 1911, to April, 1912, during which time improved conditions of watering were brought into use, as indicated by the experiments, which showed an average decrease of 77% of the dust present in the air.

Why has this important matter been apparently allowed to drop? Surely the testing of the underground air for dust is equally as important as testing for CO_2.*

I shall, no doubt, be informed that many of my suggestions are being carried out on certain mines; but it is of little use to the mine workers of the Rand if only two or three mines have excellent arrangements for the prevention of dust.

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* See Interim Report of the Miners' Phthisis Prevention Committee, this *Journal*, Vol. XIV., Aug., 1913, pp. 98-100.

If miner's phthisis is to be stamped out on the Rand within the next few years, then it is absolutely imperative that every mine, without exception, from Springs to Randfontein, must adopt every possible precaution for the prevention of dust.

Electric blasting is a step in the right direction towards making the mines less harmful to those employed in them, seeing that all persons can be removed from the mine previous to blasting; but I contend that adequate and efficient use of water during the time the men are at work, and the systematic use of sprays and water blasts after blasting, are the only methods which will ultimately make the mines of the Rand as healthy to work in as any other mines in the world.

My remarks have purposely been brief, and considerably more might have been said under each heading, but they have been so in the hope that enough has been said on each point to cause discussion upon them.

In conclusion, the prevention of dust is only possible by means of water everywhere, plus the strict enforcing of the regulations for its use.

THE EFFECT OF CHARCOAL IN GOLD-BEARING CYANIDE SOLUTIONS WITH REFERENCE TO THE PRECIPITATION OF GOLD.

By Morris Green (Associate).

(Discussion at Meetings of Institution of Mining and Metallurgy).

Dr. A. L. Simon, in introducing the paper, said that the author was to be congratulated on his thorough work which showed why charcoal was a gold precipitant from cyanide solutions, while graphite was not.

The author found the cause for the precipitation to be the gases, particularly CO, occluded in the charcoal. The experiments carried out left little doubt as to the correctness of his conclusions.

In support of the author's experiments he would like to show two test tubes which he had prepared some two years ago when working on the precipitation of gold from gels. Both tubes originally contained a silicic acid gel with chloride of gold. A piece of graphite had been added to one of the gels; a piece of charcoal to the other. The graphite had precipitated very little of the gold, the gel still being covered yellow, whereas the charcoal was covered with crystalline gold, and the gel appeared colourless.

Mr. H. K. Pickard said he looked upon the paper as of theoretical rather than of practical interest, as no one would propose to employ charcoal precipitation in a modern cyanide plant. Nevertheless he thought it was a valuable contribution to what had always been a debatable question, though he was not sure that the author had conclusively determined the reactions involved. He thought that with the object of limiting the length of his paper the author had omitted some details of his experiments which would have been of interest, and would have led to a clearer understanding of the subject. For example, in the first series mentioned at the top of p. 2 the author did not give the strength of the solution in gold, nor the proportion of charcoal used. The latter might well have an important influence on the results, and it would be interesting to know whether, had twice or three times the quantity been employed in another parallel series, similar results would have been obtained. The same remark applied in the next series, in which the author obtained some charcoal carrying 5 mg. of precipitated gold, but he did not tell them the percentage of gold in the charcoal.

The conclusion which he thought could be drawn from that series was that a solution poor in gold, though rich in cyanide, would not be precipitated by charcoal, as the solution originally containing 0·25 KCy, and no gold, dissolved 1% of the gold previously present in the charcoal, thereby producing a very weak gold solution which evidently had not been reprecipitated.

At the bottom of p. 3 also, in the cornet series of tests, a little more detail would be of interest. The author here said 'a certain amount of gold' was found to have entered into the solution. This was somewhat vague: one would like to know how it compared with the previous tests, and whether the test was carried on until no free gold was left in the form of original cornet.

In the experiments on p. 4, the amount of charcoal used was given, and if he followed the tests correctly, this finally carried about 1·7% of gold. He (the speaker) would refer to Lowles' paper, read before the Institution in 1899,* wherein it was shown that in practical precipitation work the charcoal contained about 0·5%. It was this which led him to suggest that the results might possibly vary with the amount of charcoal used, and that had the author employed three or four times the amount of charcoal he might not have got such close agreement in the results of tests A and B.

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*Trans. vii., 190-196.

The reference to the use of porous substances other than charcoal was interesting, though one would hardly expect that pumice or kieselguhr would effect any precipitation. It would have been interesting (in view of the author's later experiments) if he had determined the absorptive power of pumice and kieselguhr for carbon monoxide and hydrogen (in replacement of the absorbed air) and then noted whether precipitation took place with material so treated. It was, of course, well known that the gases absorbed by charcoal, platinum and other highly extended surfaces exhibited chemical characteristics which differed greatly from their normal properties.

He did not think the tests on the gases called for other comment than that they were excellently designed and carefully carried out, and provided matter of considerable interest. The author said that it had been stated that the gas held by charcoal upon its surface was in a highly condensed condition, and suggested that might account for the precipitating action: this seemed a very reasonable proposition in the absence of any other explanation.

He agreed with Dr. Simon in thinking that a useful practical point in the paper was that the author showed (not for the first time, as he had already published the fact a year ago) that the presence of graphite during cyaniding did good rather than harm.

Mr. C. O. Bannister said there had been a large amount of work done on the mechanism of the reduction of gold from gold chloride and from gold-bearing cyanide solution by means of carbon. He really did not think the author had got the full amount out of his subject or made the best of the manipulative skill he had shown.

Referring to the figures on the second page, to which Mr. Picard had already drawn attention, he would like to add that the author ought to have shown how much gold was precipitated in less than three hours. The author showed that the same amount was precipitated in 3, 6, 9, 12, 15 and 18 hr., but not how much in shorter periods, and the experiments should have been continued in order to determine the shortest time necessary to obtain maximum precipitation. In some cases the gold was precipitated very rapidly, for instance, it was well known that from a gold chloride solution one could get practically the whole of the gold precipitated by simply filtering through animal charcoal; so it was quite possible that the maximum amount of gold was precipitated long before the three hours, which was the shortest time used. He thought full credit was due to the author for pointing out the difference in the precipitating power of the carbon monoxide which was loosely held by the charcoal and that which was not so loosely held—in other words, what was merely absorbed in the pores as against what was absorbed by the charcoal.

The author pointed out one rather interesting and quite unexpected result on p. 10, that on heating charcoal *in vacuo* a rush of gas had come off at about 500°. This was an absolutely unexpected phenomenon to have noted, and he should say it probably wanted confirming. He thought the author had assisted very much in showing that carbon monoxide was the active reducing gas, and this was confirmed by the results of previous workers who had shown that as a result of the precipitation carbon dioxide was formed.

Mr. H. N. C. Cobbe said he understood a previous speaker to say that from a practical point of view there was not much importance to be attached to a paper on the action of charcoal in the precipitation of gold. This view, he thought, was not quite giving the subject due credit. Charcoal, as an intentional precipitant, had certainly been given up in most works that had ever employed it, but the problem of cyaniding gold-bearing material which contained charcoal presented another form of the subject and surely still deserved consideration. Tailing heaps, which either accidentally or through carelessness had got mixed up with ashes and charcoal were often met with, and apart from the assay value, it then became a question for determination whether it was worth while to cyanide such accumulations in face of 'unauthorised precipitation' which took place in the treatment plant itself.

Speakers seemed liable to quote their own experiences when specific cases were on the tapis, but in this case there was no special instance before the meeting, and therefore he might mention that he had recently seen, in one spot, several hundred thousand tons of alluvial tailings which were washed all over with ashes and charcoal from the boilers of the power plant which had been generating the steam for the pumping station.

There was no doubt that the total gold content of the accumulation in question (the gold was mostly in a very fine state) was most attractive it could be extracted by cyanide, but little seams of charcoal and ashes could be found right through the whole accumulation, and, of course, it was often just the same with mill tailings. More research, not less, might show exactly what it was in or about charcoal that actually caused the precipitation of gold—this would be an approach to the time when it might be shown how the undesirable and premature precipitation might be obviated in the presence of charcoal, and such tailings would then be available for cyanide treatment. It seemed to him that the practical value of the subject lay in that direction; the paper did not, of course, aim at showing that charcoal was a commercial competitor with zinc for purposes of precipitation.

The President (Mr. Bedford McNeill) said that he was exceedingly sorry that the author was in South Africa and could not be present with them. He thought the paper was an interesting one, and very much liked the author's method, the way in which he gradually eliminated the various possible alternatives, and then, having established the facts, reversed the procedure and proved the truth that he had ascertained at first.

He wished to congratulate the author himself upon his work, and he certainly congratulated the Institution on having him as one of their students. He could safely predict that this student would do good work, when in due time he attained the status of full membership. (The foregoing from *Bulletin*, No. 110, Nov. 13, 1913.)

Mr. Paul T. Brühl: I was much interested in Mr. Green's paper. The statement that graphite does not precipitate gold from cyanide solutions attracted my attention. Although Mr. Green certainly appears entitled to hold the opinion he does on the result of his experiments with Mt. Morgan graphite, his assertion, I consider, is not entirely correct. In some of the West African gold mines graphitic schist is associated with the lode, and it is found that the extraction in the cyanide department is usually in the neighbourhood of 50%. Laboratory experiments have shown that if the percentage of graphite in the sands is increased, the recovery of gold by KCN solution is markedly diminished. Obviously, then, there are two varieties of graphite—one has no harmful effect on the gold solution, the other has. The precipitating agent in the graphite is probably (as in the case of charcoal) carbon monoxide. It would be instructive to know the percentages of this gas in the two varieties of graphite. The problem of premature precipitation is of importance on mines where large tonnages of moderate value are dealt with, and where it is not practicable to roast the sands before cyaniding them. At present poor extractions seem inevitable, and the only palliative that suggests itself is the drawing of the cyanide solution through the sands charge as rapidly as possible. (The foregoing from *Bulletin*, No. 111, Dec 11, 1913.)

THE SOLUBILITY OF IODINE IN SODIUM IODIDE SOLUTION.

(Read at December Meeting, 1913.)

By Harold W. Gill, B.Sc.(Lond.), A.I.C. (Member).

DISCUSSION.

Mr. J. McCrae (*Member*): Mr. Gill's method of working as described on page 291 of the *Journal* would lead one to believe that what he determined was the amount of iodine contained in a definite weight (10 gm.) of a solution which had originally contained a known percentage (5, 10, 15 or 20) of sodium iodide. Mr. Gill does not indicate precisely what he means by a "10%" solution of sodium iodide but as he *weighed* the solutions saturated with iodine it is probable that the original solution was also made up by weight and contained 10 gm. of sodium iodide in 100 gm. of solution. Mr. Gill's figures would be more valuable if he would inform us exactly what they are.

In Table I. we are told that y is the amount of iodine dissolved by 100 gm. of x% sodium iodide solution: this does not appear to be what Mr. Gill determined, nor do the data given in the paper allow this to be calculated. From the description of his method of working Mr. Gill found, presumably, that 10 gm. of the solution (first on Table I.) contained 0·499 gm. of iodine capable of reacting with thiosulphate: he assumes that the 10 gm. of solution also contain 0·5 gm. of sodium iodide but such is not the case since a change in volume would take place by the solution of the iodine. Therefore the proportion y/x in Table I. can only be calculated approximately since the amount of sodium iodide is not accurately known.

Mr. Gill announces the startling conclusion that "each 1% of NaI present brings about the solution of 1% of iodine from which it would seem that a compound $NaI_2 2H_2O$ exists in solution." In the first place, a very conclusive proof will be required before we accept the statement that a compound with two molecules of water exists in aqueous solution. In the second place, even if the *molecular* proportions showed one atom of iodine dissolved per molecule of sodium iodide the conclusion that the compound NaI_2 exists would not be justified. To argue from *weights* of substances, without taking into account atomic or molecular relationships, is surely an error of a very elementary type.

It is unfortunate that the author of the paper has not referred to previous work on a cognate subject. Jakowkin (*Zeit. Physikal. Chem.*, 1894, *13*, 359 ; 1896, *20*, 19), Dawson (*J. Chem. Soc.*, 1901, lxxix, 238), and Bray and MacKay (*J. Amer. Chem. Soc*, 1910, xxxii, 914) have shown that when iodine is dissolved in aqueous potassium iodide a triiodide (KI_3) is formed and Bray and MacKay suggest that a small quantity of pentaiodide (KI_5) is also formed.

In order to utilise Mr. Gill's experimental results in any way, it is necessary to assume that the figures in Table I. represent x = grams of sodium iodide per 100 gm. of solution saturated with iodine, and y = grams of iodine per 100 gm. of solution : this is not what the table-heading states, but Mr. Gill appears to have used his figures with this meaning. Calculating the figures to *molecular* proportions we find :

Sodium Iodide		Iodine		
Grams. per 100 gr.	Gram-molecules.	Grams. per 100 gr.	Gram-Atoms (I).	
x	a	y	b	b/a
5	0·033	4·99	0·0393	1·19
10	0·067	9·96	0·0784	1·17
15	0·1	14·93	0·1175	1·175
20	0·133	20·02	0·1576	1·185

This shows that the experimental results do not support the conclusion as to the existence of a compound NaI_2. If these figures led to any conclusion it would be to the existence of a compound $NaI_{2\cdot 2}$ (Na_5I_{11}) : but such conclusion is absolutely invalid, because no account is taken of the equilibria of ionisation and dissociation products :

$$NaI_3 \rightleftharpoons Na^{\cdot} + I_3'$$
$$I_3' \rightleftharpoons I' + I_2$$

presuming that sodium triiodide is formed.

Mr. Gill would appear to base his conclusion as to the formation of NaI_2 on the figures given in Table I. It is purely accidental that the ratio of the weight of iodine dissolved to weight of sodium iodide in solution has been found by Mr. Gill to be approximately 1. It is surely a fortuitous circumstance that he chose the temperature 25° C. That the ratio of the weights is not 1 to 1 at other temperatures is shown by the figures quoted in Tables IV. Using x and y in the sense in which Mr. Gill uses them in his Table I. we select the following figures from his Table IV.:

t	x	y	y/x
15	10	9·27	0·927
30	10	10·365	1·0365
29·75	20	20·63	1·0315
10·5	10	10·88	1·088
40·5	20	22·01	1·1005
60	10	12·57	1 257
60	20	21·70	1·235

It would be interesting to know what deductions Mr. Gill would draw from these ratios.

Whatever practical significance may attach to the results recorded, it is certain that no theoretical deductions may be drawn from them.

Obituary Notices.

DR. JOHNSTONE BROWN.

The death is recorded with much regret of Dr. Johnstone Brown, which took place on December 28th at the General Hospital, following an operation for internal abscess.

Dr. Johnstone Brown was born in Lanarkshire in 1871. He was the son of the Rev. Matthew Brown, a Presbyterian minister. After taking the M.D. degree at Glasgow University, the deceased physician travelled to the East as a ship's doctor, and shortly afterwards started practice at Ficksburg, O.F.S. When the war broke out, he was taken prisoner at Ficksburg by General De Wet. On being released he made his way to Wepener, and went through the siege of that town by the Boer forces. When the town was relieved, Dr. Johnstone Brown took up duty in the General Military Hospitals, first at Standerton, and later at Cleveland. On the conclusion of the war he started practice in Market Street, Jeppe, where he resided till his death.

Dr. Brown held the positions of Senior Physician to the General Hospital, Surgeon Captain of the Witwatersrand Rifles, and Hon. Treasurer of the Witwatersrand Branch of the British Medical Association. He was also physician to a large number of Societies and to the Fire Brigade, and took a lively interest in the work of the South African Red Cross Society. He was well-known to a large number of members of this Society, which he joined as a member in August, 1904, and he acted as one of the judges in the two competitions which have been held for the Society's Ambulance Shields.

The funeral on the 30th December was very largely attended, the coffin being borne on a fire engine and military honours accorded by the Witwatersrand Rifles. The sincere sympathy of the Society will go out to the widow in her sad bereavement.

MR. WILFRID TAYLOR.

The death has been announced of Mr. Wilfrid Taylor a former member of the Society. Mr. Taylor was well known to many on the Witwatersrand where he had held many positions, including that of General Manager of the Saxon Gold Mines, Ltd. He resigned his membership last year on taking up farming in Rhodesia.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ASSAYING BY IRON-NAIL METHOD.—"W. J. Sharwood, Lead, S.D. (communication to the Secretary*: It seems possible that one of the principal differences between nail-method slags and the slags formed in the ordinary lead assay, and perhaps the chief cause of the less satisfactory character of the former, may be the presence of a larger proportion of ferrous sulphide, or, as usually stated, of ferrous alkaline sulphide. This is due to the relative smallness of the usual lead charge and its commonly lower proportion of sulphur. Thus with the ordinary 5 gm. lead assay charge, a sample of pure galena would carry only 0·67 gm. of total sulphur. With an ore consisting of one-third each of galena, blende, and pyrite, there would be 1·66 gm. of sulphur. The proportion of alkaline flux per unit of ore, calculated to Na_2O and K_2O, is commonly somewhat greater in the lead assay than in the mixture used by Messrs. Hall and Drury, and the large ratio of potash to soda also makes it decidedly more fusible than an unmixed soda flux. I believe the general experience of assayers is that the nail method tends to give decidedly low silver results, but is reasonably accurate in regard to gold; while the nitre fusion, if carefully performed, is fairly satisfactory for both gold and silver. In some assays made several years ago, with charges of 0·5 and 1 assay ton of Frue concentrate containing up to 50% of pyrite and pyrrhotite, using nails or iron filings, a few slags were analyzed, and it was found that nearly half the sulphur remained in the slag as sulphide. The late Richard Smith, when teaching assaying at the Royal School of Mines, used to recommend the cut nails then in vogue, used head downward to expose the greatest surface possible; alternatives being lengths of nail rod or pieces of 0·5 in. hoop iron, sometimes bent U-shape; or substituting a suitable amount of iron filings or turnings. The manipulation necessary when large nails or long pieces of metal are used and removed, is a time-consuming inconvenience, while the waste of metal is considerable, as it is rarely safe to use a nail more than once. It seems desirable, therefore, to use iron in some form exposing more surface, adjusting the amounts to the requirements of the case, so that all may be consumed, leaving no excess to be removed. The idea is not a new one. In the case of the sulphide ores of lead, which are partly decomposable by sodium carbonate alone, partly by iron alone, but completely by both together, the use of iron filings is mentioned by Mitchell and by Percy, both of whom refer to still earlier work. In fact, this seems to have been one of the earliest forms in which iron was used for assay purposes. In Cramer's work on assaying (Mortimer's translation, 2d ed., 1764) one section is devoted to 'Precipitation by Iron and Lead of Silver out of a Mixture containing a great deal of Sulphur,' in which the reader is cautioned: 'You must not use Filings quite spoiled with Rust: For they have no Virtue for absorbing the Sulphur.' At the Homestake assay office the practice, introduced about 10 years ago, is to use small steel wire nails, a number of which, suited to the material to be assayed, are stuck in the top of the charge before adding the borax cover. A size found well adapted to charges of tailing carrying from 3% to 5% of sulphur is a ⅞ in. brad of 18 gauge wire, the diameter of which is about 0·045 in. An old Californian book—Barstow's 'Sulphurets,' I believe—recommended small tacks for this purpose. The advantage of using normal sodium carbonate (soda ash) as a flux, rather than the bicarbonate to which, for some unexplained reason, so many assayers still adhere, is very obvious. It may be worth while to point out here that soda ash usually costs less per pound, and that a pound of it contains over 50% more actual fluxing material than a pound of bicarbonate, though the latter gives off about 90% more gas per pound before action ceases. For every gm. of Na_2O available as flux, one must use 1·7 gm. dry sodium carbonate (soda ash) or 2·7 gm. of bicarbonate. If both are pure the volume of gas given off by the soda ash is just one-third of that evolved by an equivalent (not equal) weight of bicarbonate. Thus the amounts finally given off per gram of Na_2O available are nearly 500 and 1,500 cc. respectively if measured at 100° C., or about double these volumes at 400° C. A reagent which presents some advantages, and was recommended by Percy but is now little used, is red lead as a substitute for litharge. It contains at least 25% more oxygen, retaining it up to a fairly high temperature, so that the extra oxygen is available for the oxidation of sulphur or other reducing agents. Some of the redder samples of litharge in the market contain an appreciable amount of extra oxygen. While it would not be permissible to make the slag highly acid in carrying out the nail assay, it seems desirable to add some silica whenever the percentage of pyrite is high, if only to check the corrosion of the crucibles. I believe this is common practice. One writer on assaying (W. L. Brown) has gone to the extreme of recommending the addition of silica to all assay charges—even with quartzose ores."—W. J. SHARWOOD, *Bull. Amer. Inst of Mining Engineers*, July 1913, p. 2685. (H. A. W.)

DANGER OF BARIUM PEROXIDE AS PART CARGO.—"At the inquiry into the loss of the *Volturno*, Mr. P. V. Dupré, chemical adviser to the Home Office, said the cargo contained several oils which had a very high flash point. It also contained peroxide of barium, which, when subject to friction, burst into flames. He suggested that the fire was probably caused by barium peroxide which had escaped from the iron drums, and, owing to the rolling of the ship, had become ignited by friction with either wood or iron. This occurrence, together with a recent case in the Law Courts, will tend to open up the whole question of the shipping of inflammable chemicals."—P. V. DUPRÉ, *The Oil and Colour Trades Journal*, Dec. 13, 1913, p. 2095. (J. W.)

THE DETECTION AND ESTIMATION OF NICKEL BY MEANS OF α-BENZILDIOXIME.—"A 0·02% alcohol or acetone solution of α-benzildioxime which has been made somewhat ammoniacal will immediately produce a copious intensely red precipitate of $C_{28}H_{22}N_4O_4Ni$ with a solution containing 0·002 mgm. Ni in 5 cc. (1 part in 2,000,000). 10 cc. of an ammoniacal solution containing 1 part per 1,000,000 of Ni and 100 times that amount of Co gives a distinct and immediate test for Ni. Fe, Ag, Mg, Cr and Mn do not interfere with the qualitative determination. Large amounts of nitrates seriously affect the determination and must be removed by evaporating with H_2SO_4 before adding the reagent. In the quantitative determinations amounts containing less than 0·025 gm. Ni should be used. Add with stirring a slight excess of a warm solution of the reagent in alcohol, to which NH_3 has been added, to the ammoniacal Ni solution and warm for a few moments on the water bath. Precipitation is complete in 1 minute. Filter on a gooch, wash with 50% alcohol and hot water and dry at 110°. The precipitate contains 10·93% Ni. Directions for the quantitative determination of Ni in the presence of other metals are given. α-Benzildioxime is made by boiling 10 gm. benzil (not necessarily pure) with 8—10 gm. hydroxylamine-hydrochloride in MeOH solution; after boiling 3 hours the precipitate is filtered off, dried, washed with hot water, then a small amount of 50% alcohol and dried, when it consists of pure α-benzildioxime (m. 237°). A further yield may be obtained by boiling the filtrate with hydroxylamine-HCl."—F. W. Atack (*Analyst*, *38*, 316-2; *J. Chem. Soc.*, *103*, 1317-21), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3291-2. (J. G.)

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Aluminium Corroded by Mercury.—"An interesting report of an accident caused by a corroded aluminium vessel in a brewery is related in the *Tageszeitung fuer Brauerei*, of Berlin. In consequence of the breaking of a thermometer, the quicksilver came in contact with an aluminium vessel, and within a few hours holes were eaten into the tank. Where aluminium vessels are used, thermometers should be filled with alcohol."—*The Chemical World*; *Tageszeitung fuer Brauerei* (Berlin), Sept. 1913, p. 295. (J. W.)

Fixation of Nitrogen.—Progress in Scandinavia.—"This article describes the gradual development of the processes for the manufacture of calcium carbide, cyanamide, ammonia and nitric acid, using atmospheric nitrogen. The Nitrogen Products and Carbide Co., Ltd. was formed to acquire the rights of the new process for almost the whole world and to establish works for the manufacture of nitric acid and nitrate of ammonium and also to develop existing carbide and cyanamide factories. It has been decided to establish works in England—on the Thames and at Manchester, in Scotland and Ireland, calculated to reduce the dependence on overseas sources for supplies of raw materials for use in the manufacture of explosives."—*Times Engineering Supplement*, Oct. 15, 1913, p. 21. (T. D.)

Detection of Nitrous Acid.—"Nitrous acid is liberated from its salts by the addition of a syrupy solution of citric acid, and detected by the formation of the brown ring at the surface of contact of a superimposed ferrous sulphate solution. Nitrates not being decomposed by citric acid do not respond to the test."—Leclère, *Journal Soc. Chem. Industry* (from *J. Pharm. Chim.*, 1913, *8*, 299), Oct. 31, 1913, p. 975. (T. D.)

A Simplified Method for Coal Distillations, etc.—"In a combustion tube, 500 × 13 mm., closed at one end, is weighed 15 gm. dried and finely powdered coal, followed by an asbestos wad. Next is placed 12 cm. of crushed firebrick (3-5 mm.) and another asbestos wad. The next 5 cm. are left empty, and the remainder is filled with weighed cotton wadding. The tube is placed in a combustion furnace up to the empty space, an asbestos plate serves to keep radiation from the rest of the tube, and the 15 cm. containing the cotton are immersed in a boiling water bath. Next follow a bulb with 1 : 3 H_2SO_4 (*S*); a $CaCl_2$ tube (C_1); 2 bulbs with 1 : 3 KOH (*K*); another $CaCl_2$ tube (C_2); and finally, a large bottle for a gas holder, filled with H_2O, and provided with a manometer, a thermometer, and a siphon connection to a lower receiver. The apparatus is put under suction by this siphon, and, if tight, is closed by a cock which is opened as soon as the distillation furnishes a slight positive pressure. The burners under the firebrick are first lit, and when the bricks are hot the first burner under the coal is lit. After 15 min., the next, etc., being careful to keep up a uniform rate of distillation by regulation of the burners and the furnace tile. Two to 3 hr. suffice for the distillation. Break the tube between the firebrick and the cotton, weigh the end containing the tar and cotton, clean the tube, and weigh. The weight of coke may be determined by warming the tube, plunging into cold water, and drying the coke. The increase of (*S*) and (C_1) = gas liquor. NH_3 is determined in (*S*); increase in the weight of $(K + C_2) = CO_2 + H_2S + HCN$. Titrate a portion of the contents of (*K*) with I_2 to obtain $H_2S + HCN$; another portion is distilled with $Pb(NO_3)_2$, absorbed in KOH, and the KCN titrated with $AgNO_3$. The method yields only relative results, that is, a coal is distilled as above and the relation of the yields so obtained to the works yields is determined. This relation can then be carried over to other coals. The accuracy of the results depends on careful and uniform distillation."—I. Schramm, *J. Gasbel.*, *56*, 589-91; *Chemical Abstracts*, Vol. 7, No. 17, Sept. 10, 1913, pp. 3013-4. (J. G.)

The Estimation of Nitrites by means of Thiocarbamide and the Interaction of Nitrous Acid and Thiocarbamide in the Presence of Acids of Different Strength.—"In presence of a weak acid such as acetic, $CSN_2H_4 + HNO_2 \rightarrow HSCN + N_2 + 2H_2O$. If acid such as HCl is present, $2CSN_2H_4 + 2HNO_2 \rightarrow C_2S_2N_4H_6 + 2NO + 2H_2O$. Analysis of the gas evolved when nitrites react with CSN_2H_4 in solution containing one of the more common organic acids shows the presence of N_2 and NO in proportion dependent on the acid employed. A small amount of N_2 appears instead of NO in the gas evolved when HCl is present. However, for analytical purposes, either reaction may proceed, as the volume of gas evolved is the same in both cases. For a rapid and accurate gasometric estimation of HNO_2 in a nitrite proceed as follows: Place 1 cc. H_2O in the cup of the nitrometer, dissolve in it 0·1 gm. $CS(NH_2)_2$, and a quantity of sample corresponding to a milligram molecular proportion of HNO_2; admit solution to the nitrometer, rinse out the cup with another cc. H_2O, and add 1 cc. acetic acid (20%). The reaction is complete in a few minutes. The accuracy of results is not affected by nitrates even when they are present in large amounts. The method is more rapid and cleaner than the similar one in which $CO(NH_2)_2$ is used."—M. E. Coade and E. A Werner (*Proc. Chem. Soc*, *29*, 188; *J. Chem. Soc.*, *103*, 1221-8; cf. *C. A.*, 7, 947), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3288. (J. G.)

The Estimation of Zinc as Zinc Ammonium Phosphate and as Zinc Pyrophosphate.—"The gravimetric method of determining Zn by precipitation with $(NH_4)_2HPO_4$, drying at 100—105° and weighing as $ZnNH_4PO_4$ or igniting and weighing as $Zn_2P_2O_7$ gives accurate results when applied to solutions of pure salts. In presence of considerable amounts of Na salts, add 5—10 gm. NH_4Cl before precipitating with $(NH_4)_2HPO_4$. A double precipitation followed by ignition to $Zn_2P_2O_7$ is necessary if large quantities of K salts are present. The precipitation with $NaNH_4HPO_4$ as recommended by Low (*Tech. Meth. Ore Anal.*, 1911, p. 30) gives irregular results, sometimes quite accurate, and sometimes very unsatisfactory. Some Zn appears to remain in the solution, but the greater part of the trouble is caused by the acidity of the solution, as 1 cc. acetic acid is added after neutralization. Under such conditions the precipitate supposed to be $ZnNH_4PO_4$ contains basic phosphate, $Zn(OH)_2$ and $ZnNaPO_4$. As precipitants, Na_2HPO_4 and K_2HPO_4 are unsatisfactory. Preference is given to $(NH_4)_2HPO_4$."—T. M. Finlay and A. C. Cumming (*Proc. Chem. Soc.*, *29*, 153; *J. Chem. Soc.*, *103*, 1004-10), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3290.

Analysis of Minerals Containing Phosphates and Small Amounts of Uranium.—"The authors have tested all the methods available for separation of U and P_2O_5. The only satisfactory method was by dissolving the mineral in HNO_3, freeing from SiO_2, adding alkali until only slightly acid, and boiling after addition of Na acetate; all P_2O_5 was precipitated as Fe ''' phosphate and the remaining Fe in the mineral was precipitated as the basic acetate."—C. Lepierre and A. de Carvalho (*Rev. Chim. pura app.*, *9*, 45-8), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3288. (J. G.)

A Rapid Method for the Determination of Phosphorus in Steel.—"Wash the yellow NH_4 phosphomolybdate precipitate, obtained from steel in the usual manner, with 1% HNO_3 until free from Fe; wash out HNO_3 by 1% KNO_3 solution. Transfer filter and precipitate to a 200 cc. flask, add 20 cc. 0·1 *N*/NaOH and 2 drops of phenolphthalein and titrate the excess of alkali with 0·1 *N*/HCl or HNO_3. Each cc. of 0·1 *N*/NaOH is equivalent to 0·000129 gm. of P. The method is said to give very accurate results in good agreement with those obtained by the gravimetric method."—Haripada Bhattacharyya (*J. Chem. Soc. Ind.*, *32*, 738-9), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3288-9. (J. G.)

Determination of Magnesium as Ammonium Magnesium Phosphate and Ammonium Magnesium Arsenate.—"A series of experiments is described working with solutions of $MgSO_4$ and Na_3AsO_4 to determine effect of temperature and varying amounts of NH_4OH of different concentrations. It is concluded that errors in the usual procedure are due to ignition of the filter and solubility of the precipitate. It is recommended to filter on a Gooch crucible and wash with 3·5% NH_3; dry in an oven at 100°; the gooch is placed in a large porcelain crucible containing a little asbestos; heat slowly over a Bunsen burner for ¾—1 hr., then 1 hr. over a Teclu burner, and finally blast to constant weight. The crucibles were covered during ignition. Table is given showing results with dilute and concentrated NH_4OH, precipitating warm and cold. In each case the precipitate was allowed to stand at room temperature 12 hours. When precipitating warm, the NH_4OH was heated before adding."—M. Wunder and C. Schuller (*Ann. chim. anal.*, *18*, 221-2), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3290. (J. G.)

Wine, Spirits and Vinegar Act, 1913. Union of South Africa.—"Wines may be treated with calcium sulphate equivalent to not more than 140 grains of potassium sulphate per gallon, and with potassium metabisulphite or sulphurous acid (in dry wines a maximum of 14 grains total, or 1·5 free SO_2, and in other wines 25 total, or 2·25 grains free SO_2 per gallon). All brandy or whisky must be labelled brandy, wine brandy (cognac type), grape brandy, whisky, malt whisky or blended whisky. The addition of silent spirit or of colouring or injurious matters to rum and gin is prohibited. Vinegar must contain at least 4% of acetic acid. Spirit- or distilled vinegar must not be artifically coloured. Vinegar substitutes must be labelled 'vinegarine,' and contain not less than 4·5% of acetic acid but no other acid or injurious substance. All the products mentioned are defined. The Act takes effect from November 1, 1914, for products on order for importation, from April 1, 1914, for those actually imported before June 1, 1913, and from July 1, 1913, for other products"—*Journal of the Society of Chemical Industry*, September 15, 1913, p. 879. (J. W.)

A New General Method fop the Detection of Hydrocyanic Acid.[*]—"The reaction with benzidine and Cu acetate is not characteristic for HCN, several substances behaving similarly; however, in the presence of Na_2HPO_4 and in proper concentration, the reaction takes place only with HCN. Reagent: 1 drop of a 3% Cu acetate solution + 5 drops of saturated benzidine acetate solution + 1 cc. of 10% Na_2HPO_4 solution. The solution tested must be poured into the reagent. HCN gives a blue precipitate. 0·000027 gm. HCN in 1 cc. can be detected. NaH_2PO_4 or Na_3PO_4 can not be substituted for Na_2HPO_4. If solution is *high* in HCNS, the latter may give the reaction in above method; to obviate this and make the test of general application the HCN is forced over, by a current of CO_2, into the reagent: 1 drop of 3% Cu acetate solution + 5 drops of saturated benzidine acetate solution + 0·5 cc. H_2O. *Application*: CO_2 generator connected to a flask containing soda solution connected with a 2nd flask containing solution studied from which an exit tube dips into reagent. *Sensibility*: 0·000007 gm. HCN in 10 cc. *Application*: the substance to be tested is boiled 5—10 min. with soda solution and the filtered solution treated with CO_2 as above. If metals, *e.g.*, Fe, Co, Mn, etc., are present that form double salts with cyanides in alkaline solution, the Ag salts are precipitated with $AgNO_3$ in weak HNO_3 solution, the precipitate (in complex cases may be AgCN, AgCl, AgBr, AgI, $Ag_4Fe(CN)_6$, $Ag_3Fe(CN)_6$, AgCNS) is washed well and warmed in a beaker with Zn dust and H_2O made alkaline with $Ba(OH)_2$, filtered and filtrate treated as above. The method is suggested for the detection of N in organic substances (Na method), HCN in toxicological work, gas and foul air. Guaiacum resin tincture (Schönbein's reaction) in the presence of a Cu salt behaves similarly to benzidine, giving a blue colour with iodides and many other substances; with freshly prepared tincture the addition of Na_2HPO_4 prevents the reaction with bromides, iodides and thiocyanates, but not with HCN."—C. Pertusi and E. Gastaldi (*Chem. Ztg.*, *37*, 609-10), *Chemical Abstracts*, Vol. 7, No. 19, October 10, 1913, p. 3292. (J. G.)

METALLURGY.

Titanium and its Uses.—"Despite the fact that titanium was isolated by Berzelius so far back as 1825, its utilisation on an important commercial scale may be said not to have started more than five years ago, although certain of its compounds had previously found a limited use in the dyeing industry. . . . According to the estimate of one authority, titanium is one of the ten most common elements found in a state of combination on the earth's surface. In the mineral kingdom this element most commonly occurs as the dioxide (TiO_2), either alone, as the mineral *rutile*, or in combination with ferrous oxide as *ilmenite*. . . . The function of titanium in steel manufacture is not to give new properties to the product, such as, *e.g.*, do tantalum, nickel, chromium and manganese, but to act as a scouring agent in removing minute traces of certain substances, such as slag, nitrogen and metallic oxides, which exert a deleterious effect on the wearing properties of finished steel. Much evidence has been adduced during recent years that brittleness in steel is to some degree caused by the presence of nitrogen, and it has been demonstrated by E. von Maltitz that Bessemer rolled steel which contained 0·014% of nitrogen was greatly improved, physically, by treatment with ferro-titanium, and the nitrogen content was thus reduced to 0·005%. . . . In the steel industry titanium is generally employed in the form of a ferro-alloy, containing from 10% to 25% of titanium. Alloys containing over 25% of the metal are difficult to use, owing to their high fusion point rendering them difficultly soluble in the molten steel. . . . The alloy which finds most general use is one containing from 10% to 15% of titanium, about 10 lbs. of the alloy being added for each ton of steel. The alloy, which is usually added as the steel runs into the ladle after recarburisation, causes the impurities to rise to the surface, and is itself oxidised and passes to the slag; thus in the finished steel hardly a trace of the titanium remains. . . . In the textile dyeing industry titanium salts, particularly the lower chloride, $TiCl_3$, and sulphate, $Ti_2(SO_4)_3$, are gradually finding an extensive use on account of their great power as acid reducing agents. . . . In the tanning industry potassium titanium oxalate has for some time past been used for mordanting leather, and more recently the titanium lactates, introduced by Dr. Dreher, of the Freiberg Tanning School, have found widespread application. As titanium salts readily form 'lakes,' they can be utilised in this connection either as mordants or 'strikers.' . . . It has been clearly shown, by the numerous experiments of A. J. Rossi and others, that good iron can be economically produced from these ores, and it is stated that the iron ore from the Adirondack Mountains, New York State, which was successfully smelted for 20 years, contained from 9% to 15% of titanic oxide. Attempts in the United Kingdom to smelt iron ores containing high percentages of titanic oxide have, however, been mostly abandoned soon after starting. At the present time it is very difficult to negotiate a sale, in Britain, of iron ore carrying over 2% of titanic oxide."—Sydney J. Johnstone, *The Chemical World*, Sept. 1913, pp. 283-284. (J. W.)

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[*] See also Moir, this *J.*, April, 1910, p. 342, and October, 1910, p. 155, also Proc. Chem. Soc., London, May 5, 1910.

The Simplification of Gold Ore Treatment.—"The successful application of the cyanide process in the extraction of gold from a great variety of ores has led to much theorizing on the question of the simplification of treatment; and attention has lately been drawn to the so-called advantages arising from the abolition of amalgamation in favour of direct cyaniding, whatever the coarseness of the gold in the ore. As the matter cannot be satisfactorily disposed of by a few expressions of opinion, it may not be inopportune to draw attention to the various considerations involved, and the probable advantages and disadvantages arising from such a change of policy.

System of Nomenclature.—The actual treatment of a gold ore is generally preceded by a series of experiments; and the results of these and the deductions therefrom are embodied in a metallurgical report. I would like to suggest the adoption of some definite system of nomenclature in such reports because at present the advantages of thorough amalgamation are apt to be overlooked. A report of experimental cyanide treatment generally gives a misleading estimate of probable recovery; and too often the fact is overlooked that an additional percentage must always be deducted from possible solution before it can be compared with, or used in conjunction with, possible recovery. The assaying of the ore before and after experimental amalgamation, or the actual recovery of gold during experimental amalgamation, denotes an actual possible recovery or extraction in practice. The assaying of the washed ore before and after experimental cyaniding indicates nothing more than a possible solution of gold, and must be distinctly differentiated from a possible recovery. When the percentage of possible recovery by amalgamation is added to the percentage of possible solution by cyanide the total is by no means a possible extraction of gold; but is less in error than the result of a direct cyanide test where the solution of gold is made to indicate a possible extraction.

Simple Amalgamation.—As a preliminary I would like to consider the use of the term simplification when used in connection with the suggestion of the abandonment of amalgamation in favour of all-cyaniding in free-gold milling practice. Simple amalgamation is too well known to need further description. Suffice to say the gold is obtained in its original state of purity. The method of recovery is by far the most direct and the most economical; and the actual mechanical losses before realisation are so small that they may be left out of consideration. There is an actual recovery of the total extraction of gold; and in the great majority of cases there are no refining fees to pay on bullion so recovered. The so-called 'simplification' consists in the following as an alternative. The amalgamable gold is ground and agitated in cyanide solution and a certain percentage is dissolved. The balance may be considered, so far as the treatment under review is concerned, as an unrecoverable loss; and is an actual addition to the residual loss which would have occurred had amalgamation been practised. The gold so dissolved is then, during the course of treatment, intimately mixed with the ore. Owing to the higher metal content in the pulp, a higher cyanide strength is necessary, and this results in a higher chemical loss to the solvent. During treatment the stability of the solution is impaired by the increased metal content, and the premature precipitation rendered proportionately more probable. Filtration follows thickening and agitation, and the purpose of this step in the process is to remove as much as possible of the dissolved gold from the residue. The efficiency of this separation varies with the class of apparatus used and the condition of the pulp; but let it be noted that, given the same scheme of treatment and filtration, the loss in dissolved gold is strictly proportionate to the original value of the solution in the pulp. This dissolved gold loss may be as low as 1% of the dissolved gold in the cyanided product; or it may be as high as 10%. On a homogeneous pulp a high displacement efficiency may be obtained in various types of vacuum filters or in the frame filter-press. On a non-homogeneous pulp of too high a dilution, or in other types of filters or presses, the displacement efficiency may leave much to be desired.

Precipitation Losses.—Zinc precipitation may be presumed to follow filtration. The increased gold content necessitates a higher free cyanide percentage in the solution entering the precipitation plant. This results in a higher zinc and cyanide consumption. A percentage of the gold is precipitated. Acid treatment of the zinc sludge may be assumed to follow, after which the precipitate is washed, generally dried, fluxed, and smelted. A certain mechanical loss during these operations is inevitable, and there is a further loss in slag and matte in which, even if time, trouble and expense are unstinted, the whole gold content is unrecoverable by ordinary methods. A percentage is generally realised by direct sale.

When the elaborate method of cupellation, preceded by litharge smelting, is practised, the resultant bullion is high grade. Under ordinary conditions, however, cyanide bullion is lower in grade than the bullion obtained from the amalgamation process; and refining fees have often to be considered as an additional expense.

Enough has, I think been said to show that the so-called 'simplification' cannot refer to other than the elimination from the flow-sheet of fundamental units for the recovery of gold. In actual practice the solution of gold can never be more than partly successful; and unavoidable losses occur at every stage of a treatment process rendered more complicated by an unnecessary addition of gold to the material being handled. The result is an eventual recovery of but a percentage of the gold which would have actually been obtained had amalgamation been practised.

Insistence on the theory that amalgamation is unnecessary is the natural sequence of milling in cyanide because, under such conditions, amalgamation must play a secondary part in the scheme of operations. The only direct argument which can be used against the practice of amalgamating a free-milling gold ore is that it is not up-to-date.

Treatment of Free-Milling Ore.—For the purpose of comparison I append some theoretical estimates which will serve to illustrate my contentions as to the importance of amalgamation in the treatment of a free-milling gold ore. The ore is assumed to carry $20 in gold, 50% of which may be directly recoverable by amalgamation. It is further assumed that the tailing can be cyanided and a solution of the remaining gold, to the amount of 90%, can be obtained. Before the gold is ready for a mint, a loss of 2% of all the gold which passes into solution is assumed. A solution of 98% of the amalgamable gold is assumed in the case of direct cyaniding.

1.—*Amalgamation and Cyanide Treatment.*

		Per ton.	Per cent.
Recovered by amalgamation ...		$10.00	50.0
Dissolved by cyanide, 90% of $10		—	—
Recovered by treatment, 98% of $10		8.82	44.1
Total Recovery		$18.82	94.1
Loss:			
Dissolved and precipitated gold. 2% of $9	$0.18		
Undissolved gold ...	1.00		
		1.18	5.9
Value of ore		$20.00	100·0

2.—*All-Cyanide.*

		Per ton.	Per cent.
Amalgamable Gold:			
Dissolved by cyanide, 98% of $10		—	—
Recovered by treatment, 98% of $9.80		$9.60	48.0
Unamalgamable Gold:			
Dissolved by cyanide, 90% of $10		—	—
Recovered by treatment, 98% of $9		8.82	44.1
Total Recovery		$18.42	92.1
Loss:			
Dissolved and precipitated gold, 2% of $18.80	$0.38		
Undissolved gold ..	1.20		
		1.58	7.9
Value of ore		$20·00	100·0

These figures are, I think as lenient as possible toward all-cyaniding, but by no logical rearrangement could they be made to show a result in favour of the abolition of amalgamation. An exceptionally high figure has been assumed for the solution of amalgamable gold; and the assumed loss (2%) of the dissolved gold is below the average.

It may be said that when the pulp is high in dissolved gold the filtration treatment might be preceded by a system of decantation, or decantation and replacement with barren solution, which would reduce the amount of dissolved gold in the pulp, and also the ultimate dissolved gold loss in the residue. Such a method might be followed but it would be equally applicable in the case of the treatment of the tailing from the amalgamated ore; so that given similar schemes of treatment, better results would always be obtained from the handling of material from which the greatest quantity of gold had been removed before cyaniding was commenced.

In most gold ores the abolition of amalgamation prohibits the application of leaching to the sandy portion of the product. In an ore carrying high or coarse gold content it is rarely possible to obtain a sample from the tube-mill circuit (or from the classifier discharge when the tonnage is being forced) which does not contain, in the absence of efficient amalgamation, sufficient free gold in a condition to make the leaching of the sandy portion impracticable. Unless the sand and the accompanying gold is pulverized and flaked to an exceptional fineness there is a possibility of the residue, after a complete agitation treatment, showing free gold in the pan. This is the more liable to happen when the viscosity of the mill pulp is high on account of the presence of clay in the ore.

Use of the Tube Mill.—As already mentioned, the abolition of amalgamation is closely associated with milling in cyanide; and much of the benefit said to accrue from the latter practice when used in treating amalgamable gold ores is due to the fact that, with the abolition of amalgamation, the tube-mill assumes importance as a prime factor for the solution of the excess gold. The effective action of the pebbles in bringing the solution in contact with the gold admits of no doubt; neither is there any question that a considerable percentage of amalgamable gold can be dissolved in a tube-mill circuit using cyanide solution as a diluent. But from this fact can be adduced no argument in favour of the solution of gold rather than amalgamation; it indicates no economic advantage in neglecting to recover the greater portion of the gold in the shortest time and by the most direct method.

One of the most interesting examples of metallurgical advance is indicated in descriptions of the Dome mill, Porcupine, where the efficiency of modern equipment is helped by a combination with previously proved practice. From a published account* it is to be noted that a recovery of 95% is being obtained, and 65% of this is the direct result of amalgamation treatment. After the removal of the bulk of the gold it has been found that cyaniding is so successful that only economic restrictions limit the ultimate recovery which may, from all accounts, be raised at will to an exceptionally high figure. The question of leaching the fine sand from the milling plant has been raised and the thoroughness of the amalgamation treatment will doubtless make this an economic possibility. On the other hand, and although such minor modification may be instituted, it would seem impossible to improve upon the general scheme of treatment, both as regards economy of method and efficiency of operation."—A. W. ALLEN, *Mining and Scientific Press*, Aug. 16, 1913, p. 254. (H. A. W.)

HYDROMETALLURGY. — "Hydrometallurgy is a broad term if it be held to include every process in which the metal to be won is, at any stage of the operation, in solution.

The difficulties met with in solution processes are many. For the most part they are not based upon theoretical error. The troubles encountered, a few of which we shall present, are partly due to manipulatory obstacles, partly to underestimates of expense. We cannot specify individual cases, whether prospective or operative, for obvious reasons. Nor are the difficulties to be described peculiar to any one process, or even to any allied group. We hope to show in what directions miscalculations have been common, as a warning both to over-enthusiastic inventors and to prospective investors

Every metallurgical process involves the operation of chemical laws. This may be generally recognised to-day; possibly not always in times past.

'*In Spite of Science.*'—In a by-gone day the writer, then in St. Louis, Mo., was visited by an official of the Missouri Tin Company, a corporation formed for the purpose of extracting tin from porphyry. In justice to the promoters, they did not so state it in their prospectus. The functionary aforesaid, after much earnest protest against the assertion that there was no tin in the porphyry, a statement for which the writer was responsible, wound up with the following amazing dictum: '*Why, we know as well as you do that the tin in our ore isn't susceptible of chemical detection.*'

It was almost prophetic—for did not the renowned 'Metallurgical Murphy' years later, issue his immortal report: 'The ore is of metallic contents, but basic constituents. Its values, while susceptible of chemical detection, are not capable of metallurgical extraction!'

Chemical Ignorance.—We have said that difficulties encountered in actual practice were for the most part not based upon theoretical error. However, this is hardly true of many processes which are still in the purely prospective stage.

By 'ignorance' of chemistry we do not mean total ignorance of text-book knowledge of the subject. It is a frequent happening, however, that the inventor of a chemical process knows just enough chemistry to harm him. He has read of, possibly practised, certain reactions, and tries them under conditions wholly favourable to their success. Of reactions in complex solutions, or the influence of mass; of the solvent powers of his own solutions for supposedly 'insoluble' substances—of a hundred other complications, familiar to all operative chemists, he knows little or nothing. We divide the style of ignorance too commonly found in attempted chemical metallurgy, into 'scientific, manipulatory and financial.' In the following examples of the 'left-wing' cases (*i.e.*, frauds pure and simple), we divide our criticism between the fakir and the dupe; for the latter ordinarily has ample means and opportunity to discover in advance the false or fraudulent grounds upon which he is asked to invest.

Frauds, Fakirs and Fables.—In the above classification we have not included the ignoramus *in toto*. We nevertheless shall give one frightful example of him. King Solomon has disposed of him in a well-known phrase, which we may put into modern technical language by saying that one may pound him to forty mesh in a ten-stamp mill without changing his mental characteristics. Most metallurgical men will recall without difficulty several cases of attempts at the impossible. We have at the outset to dispose of the ever-recurrent metallurgical fraud—fool he may be, fraud he certainly is—whose mission is to create matter, or in brief, to extract metal, not from where it is, but from where it isn't! 'Ven ve are dead,' said poor old Rip van Winkle, 'how soon ve are forgotten!' And these recurrent spasms of alchemistic foolery, which appear 'each in their turn to make the vulgar stare,' how soon they are forgotten in this busy world. Yet within the memory of thousands of living men, we have had three schemes of continental fame, and half a dozen of more local celebrity, all claiming the production of gold, some under one, some under another disguise, as to method; but all in fact, if not in form, claiming the rediscovery of the art of transmutation.

For the strictly technical reader, it may be a waste of time to read over accounts of mere trickery Nevertheless some of these now historical transactions are reproduced. 'To what end?' may be asked. Well, as a very slight contribution to the science of psychology. There is no appeal to science in these wornout juggler's tricks, but if we venture to classify them, as we think we can, and show that they fall under certain definite forms of appeal to ignorance, prejudice, credulity or greed, we may possibly indicate to some readers the earmarks by which he may recognise a fraud, without having too closely to investigate the alleged technical details.

'*Classified Ignorance.*'—These hydrometallurgical attempts then, in so far as they try to get the general public interested, are divided into:

(1.) Appeals to total ignorance.

(2.) Appeals to anti-scientific prejudice.

(3.) Assumed appeals to scientific principles, under a further assumption that the inventor (and he alone) has learned how to apply these profound truths to practical ends.

First: Appeal to Sheer Ignorance.—The best example that occurs to us is that of the 'Wynne' process, started something over a decade ago in Denver, Colo. The promoters so far succeeded that they actually sold stock in the process to no small amount. The 'recipe' was kept secret, but the results were apparent. Prospectors and mine owners were invited to send in samples of their ores, and did so. Invariably the 'return' was made showing values far in excess of report made by any other

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*H. A. Megraw *Eng. & Min. Jour.*, Nov. 23, 1912, and this *Journal*, vol. xiii., March, 1913, pp. 433-7.

assayer. But the new concern went further than this; in many instances it actually turned back to the sender of the ore sample the gold 'extracted.' Not the 'ignorant class,' if you please, but some men of prominence and presumable education were taken in by this particularly stupid trick. 'Of course,' you will say, 'there was some attempted explanation of the phenomenon?' No. Not so far as the public who were invited to invest were concerned. On its naked, self-asserted production of gold from rock, barren or nearly barren, the delusion found many believers. It would seem that the return of the metal in a few selected cases was productive of 'results.' The 'inventor' died. His stockholders were heirs to his chemical 'recipe.' They tried it, reporting with refreshing naïveté, that 'it wouldn't work,' No wonder. It is before us now, a silly mess of potash, nitric acid, ammonia, lithium carbonate and other ingredients. Possibly a fair bug poison, but fit for nothing else that we can imagine. The stockholders pocketed their loss and the public forgot about it in a fortnight.

Second: Appeal to Anti-Scientific Prejudice—We should like to annex the proper name to this little bit of local history, but we regard the 'feelin's' of some possible readers, who may still be a little sore. We shall call the camp by what it is in fact, not by its map name. Busted Camp, then, had 'ore' and had also many enthusiastic promoters. The 'formations' were identical with those of Cripple Creek. (Of course, for the discovery was made in the days of first development of the wonderful riches of the latter famous camp.) Everything was lovely at Busted, except that there was no gold in the ore. However, the settlers were true 'Boosters' and weren't going to let a little thing like that discourage them. They invented a new assay method, which they called a 'colour method,' but we shall not try our reader's patience with a description of the fool thing. Suffice it that the great argument used was based upon the fact that no 'scientific' man or 'regular' assayer or chemist could find the gold here. *That,* it seeems was the main point of value! For, there is no doubt about it, there is an element in every community who have deep-seated convictions that 'science' is a fraud. They could not define 'science' to save their dollars—but as a noted foreign agitator said as he landed in New York, 'whatever your government is, I'm against it,' so there are certainly those who say 'whatever science says about this, I don't believe it!' And of these were the investors in Busted Camp: the 'poor but honest' and later, bankrupt people, who invested in their own ignorance—and lost. The perpetrators of this fraud knew, of course, what they were doing. They also knew how to calculate upon ignorance and prejudice. That is the characteristic of fraudulent promoters everywhere. They are shrewd enough to gauge the prejudices of those with whom they deal. They descend to their level in their solicitations. Finding people who were already prepared to believe in a thing *in the precise ratio in which it had no scientific bearing,* they play upon that particular form of ignorance, and by skilful sneers at 'scientific' work, induce the belief that work which is *not* 'scientific' must be correct! Is the logic apparent? It was at least successful, and many a 'share' was sold. It was another proof of the prevalence of scientific illiteracy. We may roughly state the proportion of scientific illiterates at ninety per cent. of the population. The only connection of this with 'hydrometallurgy' is found in the fact that the assay method by which they found the gold that wasn't there, was a 'wet' method, and that the proposed method of extraction was also 'hydrometallurgical.'

The Ocean a Gold Mine.—Third in our classification comes the assumption of scientific principles. The 'Electrolytic Salts Company' had a wonderful history. Mr. Jernegan issued circulars stating among other things that it was 'well known' that sea water contained gold. The whole statement was a mixture of falsehood and of semi-truth. The object of the enterprise was not to extract sunbeams from cucumbers, but gold from the ocean. It is indeed a 'well-known' fact that for many years a paragraph has been floating round the world stating that sea water contains a definite proportion of gold. One metallurgist has taken the trouble to trace the statement to its source, with the result that the trail 'ran up a tree.' No authoritative statement of the contents of gold 'per ton of ocean' could be discovered. But it was enough to start the remarkable exploitation now briefly to be described. The inventor stated the amount of gold to be one grain per ton of sea water. Possibly there is some infinitesimal trace of gold there. Certainly it is not that much—but the amount matters very little. The ingenuity of the attack (on the public) consisted in the declaration that the inventor was well aware that he couldn't handle a ton of material for four cents, which is about the value of one grain of gold. He added, however, that if we could persuade 'Nature' to do the job for us, that little difficulty would be overcome. Nature has so shaped the Bay of Fundy that tides of extraordinary altitude characterise its upper reaches. 'Here we shall place our mill,' said the inventor. The tanks or reservoirs to hold the sea water are to be so placed that they will fill at high tide, and empty themselves at low tide. Here is Nature's handling of the material. But how about the extraction of the gold? Why, that was so simple that 'it was to smile.' Here we have a gigantic battery of one fluid (the sea water) and one cell (the reservoir). There were, of course, 'plates' for the deposition of the gold. In flows the water, down goes the gold. The only 'work' is to scrape off the beautiful stuff—and there you are. It was indeed a lovely scheme. The hocus of the original statement aided by the pocus of the 'battery' needed only the locus (Bay of Fundy) to bring it to a focus (subscriptions and stockholders). And the subscriptions flowed in; the mill was actually built. The scoffers scoffed, of course, but suddenly they were put to shame. Lo, a dividend! A real fifteen-thousand-dollar dividend, paid in hard cash. With it, perhaps the most ingenious statement of all, viz.: 'The yield was a great disappointment, being but one-fourth of the total gold content. From this trivial portion the dividend has been declared. But the cause of the small yield has been discovered, and the matter is easily rectified,' and so on. Now indeed, money came in fast. There had been just enough apparent appeal to scientific principles to satisfy those to whom electrolytic operations are a fact sufficiently well known, but who have never bothered themselves as to any details. Doubts were dispelled by the money. The stale old trick of a dividend paid from stock subscriptions had worked once more. Then came the closing of the mill, the flight of the promoter to a more congenial climate, and other highly uninteresting details. It is said that the fugitive was caught abroad, but we are not able to give the closing chapters of this miserable history." Regis Chauvenet, *Metallurgical and Chemical Engineering*, Sept. 1913, p. 486. (H. A. W.)

EFFICIENCY OF HARDINGE MILL.—"The writer gets a production of 392 mesh-tons per horse-power day for the Chilean mill, and 478 mesh-tons per horse-power day for the Hardinge, an increase of some 22% in favour of the Hardinge. (The term *mesh-ton* represents the increased surface produced by crushing all particles of a ton of rock to a diameter whose reciprocal is one greater than in its previous condition. Diameter should be in inches, although of course this can be adapted to other units. For example, a ton of evenly sized pieces 1 in. in diameter would have 1 mesh ton of surface; a ton of similar pieces just passing a hole 0·01 in., and retained on a screen with holes the reciprocal of whose diameter was 101, would have 100 mesh-tons of surface; the difference between two lots of 1 ton each, whose diameter reciprocals were respectively 99 and 100, is 1 mesh ton.)

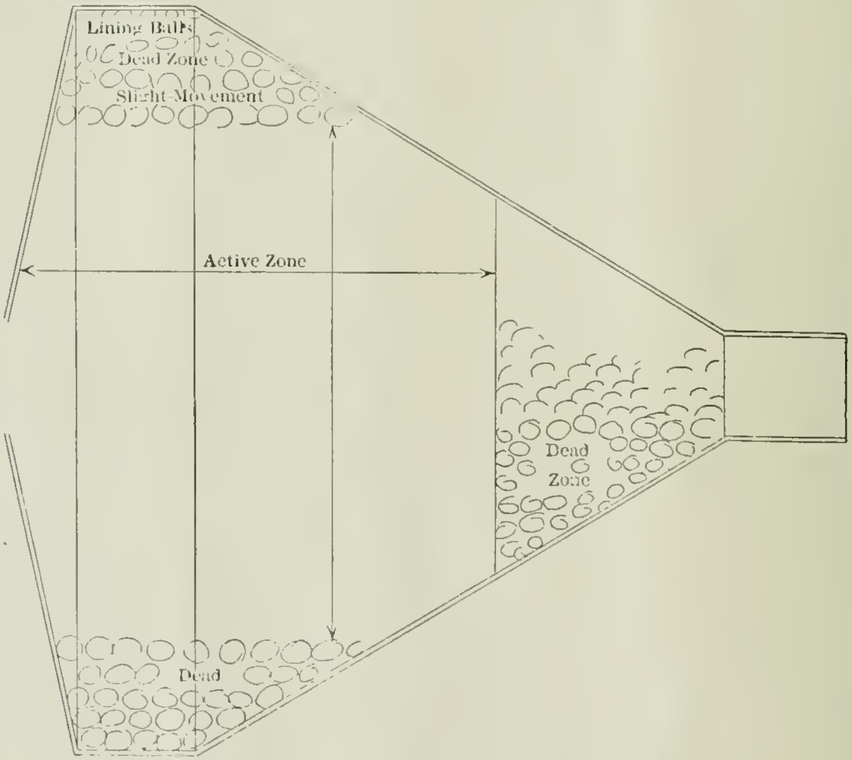

FIG. 1.—Active and Dead Zones in the Hardinge Mill.

In spite of the commercial success of the Hardinge mill, and the increased economic results accomplished by its introduction into concentrating mills, I wish to criticise the statements that are so frequently made as to the value of the cone, its segregating action on the pulp, and the graduation of forces, intensities of energy, or inertia, so that each particle gets just the right blow. I have never seen any published results of screen analyses of material taken at different points along the cone, and I do not think screen analyses taken at all these points will bear out the claims made for this feature. Analyzing the mill on the assumption that the greatest diameter is to produce the greatest effect in crushing, we find that the weight of crushing pebbles is proportional to square of the diameter (machine half full); that the energy per unit pebble weight is something nearer the square than the first power of the diameter; and that the velocity with which the ore or pulp being crushed passes through the mill is inversely proportional to the square of the diameter. The result is that the energy applied per pound of pulp at various points along the cone is inversely proportional to about the sixth power of the diameter. This means that half way towards the apex of the cone, only 1/64 as much work is done as at the cylindrical portion, while three-fourths of the way towards the apex only 1/4000 is done. This means that the energy applied along the cone is so small that the force exerted by the falling or rolling pebbles is not sufficient to break the coarser particles, with the result that the work in the cone is largely done on the lines! This is as logical as the generally accepted explanation.

But the Hardinge mill is not run at such speeds that the effect of the large diameter is obtained; it runs at such a speed (750 feet per minute peripheral speed for the 8 ft. diameter size, according to Mr. Hardinge) that centrifugal force at the periphery is about 1·2 times that of gravity, and therefore at least one layer of pebbles in the periphery is useless except for the purposes of lining. For at least a foot in, the possible fall of the pebbles is so slight as to be valueless for crushing. The result is, neglecting part of the apex of the cone, where the energy is too small to be effective, the Hardinge mill resolves itself automatically into a short tube mill, the 8 ft.

size producing about the same effect as a 5 x 5 or 6 x 6 ft. tube mill, as in Fig. 1.

Perhaps I am like the sailor's mother who could credit his story about the mermaids, but refused to believe what he told her about the flying fish. The segregation of the pebbles is entirely reasonable, the survival of the most energetic, but how can the fines separate themselves from the coarse in the turmoil taking place within the crushing zone? The particle has got to go where it is knocked, the agitation is too great for it to follow any laws of classification, and it with the others passes through by displacement and chance, perhaps getting through without being hit at all; or, again, a single particle in the final pulp may be the result of perhaps a hundred blows.

In Figs. 2 and 3 are plotted some graphical results of calculation of what goes on inside a tube mill, particularly of the Hardinge type. The concentric circles in Fig. 2 represent planes through the cone. The velocity of each of these circles, based on 29·75 rev. per minute, is indicated in the lower right hand quadrant. Applying the principle of mechanics, it will be found that pebbles will become free to fall when reaching the half-circle drawn in the upper right-hand quadrant, going up, and their paths from that time on will without interference follow the paths P_2, P_5, P_8, etc. Centrifugal force is too great on the outer ring of pebbles to let them move. Supposing pebbles to leave this half-circle along each of the concentric circles at the same time, lines of equal interval of time have been drawn so that one may judge velocities. I have not attempted to locate the landing place of the pebbles very accurately, although I have shown possible landing places by curved lines across the path lines. When these landing places are located properly the resultant velocity can be determined graphically, energy then being proportional to the square of the velocity.

In Fig. 3 (see next page) are plotted away from the horizontal axis of the mill, weights and energy available per foot of length, for different positions along the length of the mill, and below, the rate of flow of pulp through the mill, and the energy per pound of pulp imparted by the action of the pebbles. The curves are plotted to relative units, not absolute.

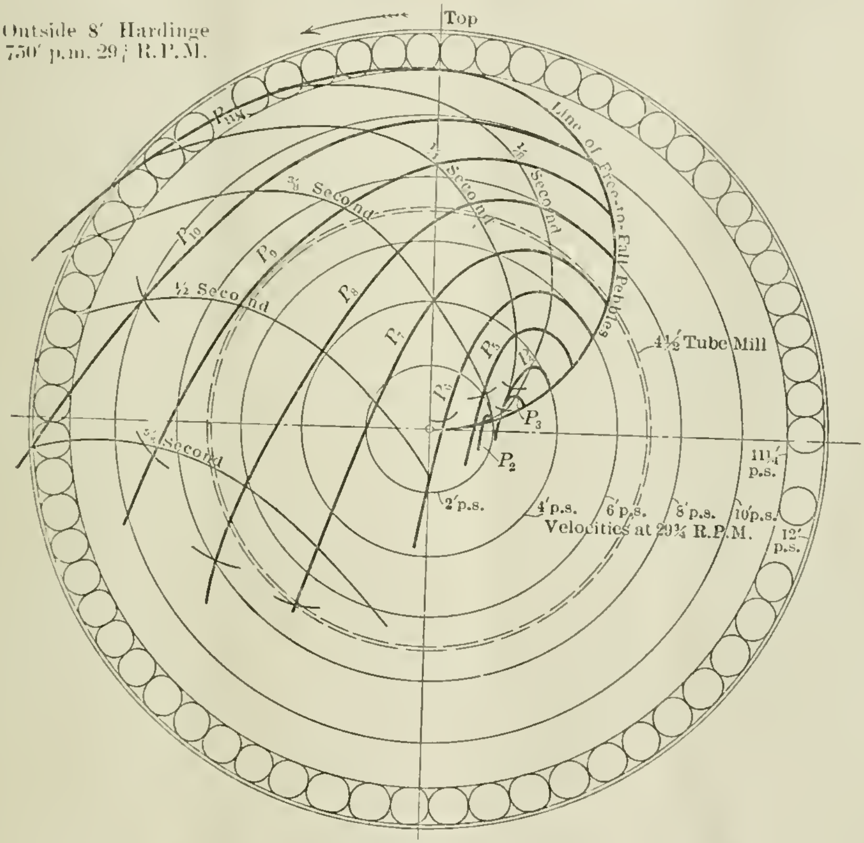

FIG. 2.—Diagrammatic Representation of the Action in a Tube Mill.

I predict that the fine-crushing machine of the future concentrating mill will be a short tube mill, followed by an efficient sizer to remove more of the fine material than is done at present, and followed by a second short tube mill. And by a short tube mill I mean short, 1 or 2 ft. long, and perhaps of large diameter, the pulp travelling through it rapidly so that the fines are not subjected to repeated crushing."—ARTHUR O. GATES, *Bull. Amer. Inst. of Mining Engineers*, July 1913, p. 2702. (H. A. W.)

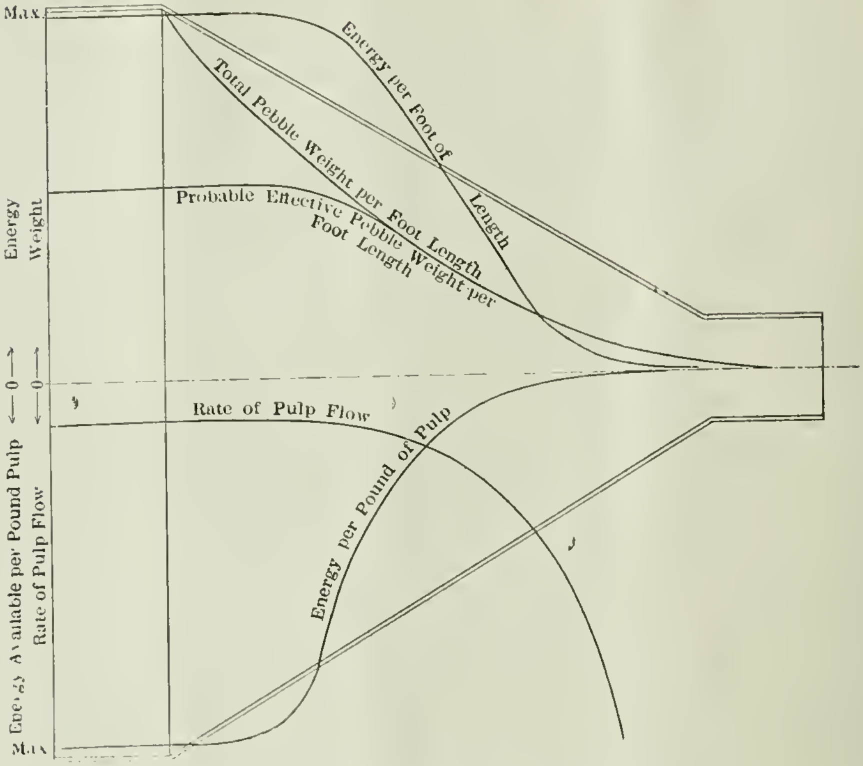

FIG. 3.—Curves of Energy, Pebble Weight, and Flow of Pulp.

SOLUTION CONTROL IN CYANIDING.—"The generally accepted equations of zinc-box reactions indicate the formation of a double cyanide of zinc and an alkaline earth as a by-product resulting from the deposition of the gold and silver. This reaction undoubtedly occurs in alkaline as well as neutral solutions. Silver nitrate titrations of such solutions may indicate either the content of free cyanide or of total cyanide. In the former case the result shows, roughly, the amount of potassium, sodium, or calcium cyanide present, which is generally recorded in terms of the potassium salt. On the other hand, a total cyanide estimation serves to indicate the content of free cyanide plus the equivalent, also in terms of potassium cyanide, of the double cyanide present; and also the hydrocyanic acid.

Estimation of Free Cyanide.—Since the commencement of the industry it has been generally taken for granted that free cyanide content may be estimated by the direct addition of standard silver nitrate to the working solution; and total cyanide by the same method, after the addition of alkali. Since cyanide treatment invariably includes the precipitation of gold and silver by zinc, and since the great majority of cyanide operations are conducted with alkaline solutions, it is exceedingly difficult to understand how any line of demarcation can be drawn. By the adoption of such a method it will be seen that if the alkali is added in the plant the subsequent titration of the solution would reveal a high free cyanide content; if added in the laboratory, a high total cyanide content. The adoption of such a line of reasoning must necessarily mean the general acceptance of the theory that the addition of alkali to a working solution containing the double cyanide of zinc and an alkaline earth results in the entire regeneration of the cyanide in the double salt, and its conversion into potassium, sodium, or calcium cyanide. This, again, is tantamount to saying that in a normally alkaline solution, gold and silver are precipitated in the zinc-boxes and free cyanide regenerated. My own experiences of the behaviour of plant solutions in connection with the solution and precipitation of gold and silver are entirely opposed to this idea. As far as I know, there is no evidence to show that any such regeneration occurs in actual practice, or that the double cyanide in question may be formed by the addition of acid to a normal plant solution carrying zinc salts, which is an obvious corollary.

Titration Results.—In protectively neutral solutions the titration with silver nitrate gives free cyanide content. In protectively alkaline solutions the titration gives free cyanide content, plus the equivalent in terms of potassium cyanide of the double salt, in proportion to the amount of protective alkali present. In the presence of sufficient alkali the estimation becomes a total cyanide one, although often reported as free cyanide. I therefore suggest that the result of a simple titration of a working solution with silver nitrate is of no definite value or significance, since only in the absence of free protective alkali would the result represent actual free cyanide content. In the presence of free protective alkali the result might indicate anything from available free cyanide to unavailable or only partly available double cyanide.

In a neutral or acid solution the titration would exclude the hydrocyanic acid. By a comparison of actual solution tests I have evidence that this substance is equally powerful with free cyanide as a solvent of gold; and, as a result of the definite recognition of its solvent powers in the Gitsham process, it may be suggested that the occasional success attending the practice of heating cyanide solutions is, in some measure at least, due to its liberation, added to the fact that, under conditions of low alkalinity, its content is not included as an available solvent in the result of the simple titration test.

It seems, therefore unreasonable that the efficiency of a solution should be arrived at by a method, which, under certain conditions of alkalinity, would exclude a powerful solvent; and, under other conditions of alkalinity, would include in the result the cyanogen in a zinc salt which makes the solution unworkable when the content rises above a certain percentage.

Precipitation of Zinc.—An inevitable result of precipitation with zinc is the inclusion of zinc salts in the working solution. With an excess of free cyanide and a moderate alkalinity in the solution entering the zinc-boxes, the ordinary titration method will give a higher free cyanide content at the tail than at the head of the box. In other words, the solution will appear to have a higher efficiency after the addition of zinc than before it. This fact may be cited as a striking instance of the absurdity of this method of testing, since it is obvious that ultimate fouling of solution may be entirely due to the addition of zinc as a concomitant of precipitation.

The following figures of actual titration results of solution entering and leaving the zinc-boxes are of interest:—

	Incoming, Per cent.	Outgoing, Per cent.
Total cyanide	0·175	0·175
Free cyanide:		
Direct titration method ...	0·145	0·150
After neutralization of solution on basis of protective alkalinity arrived at by Green's method	0·120	0·095

It is my opinion that the lack of a uniform method of determining the available and partly available cyanogen contents of a working solution, is, in the main, responsible for many of the opinions expressed from time to time as to the effect of alkalinity on extraction. For example, a definite 'cyanide strength,' based on the simple titration with silver nitrate, is being maintained, for some reason the alkalinity rises and, in proportion to the amount of rise, the zinc double salt appears in the result of the titration. The 'cyanide strength' is then brought down to normal by a decrease in the amount of fresh cyanide added. The solution or precipitation of gold becomes unsatisfactory and the high alkalinity is suspected as the cause. This is then lowered. The 'cyanide strength' drops in proportion and is raised to normal by the addition of fresh cyanide. The extraction or solution troubles disappear and the previous poor results are attributed to the alkalinity. In all probability the method of titration of the solution was at fault and indicated a 'cyanide strength' when sufficient alkali was present which bore an entirely different ratio to the actual available cyanide than did a corresponding titration when the solution was neutral or of a low protective alkalinity.

Chemical Reactions.—The following equations probably represent the reactions occurring during the titration of a solution containing free cyanide and zinc potassium cyanide.

(1) Free cyanide

$$AgNO_3 + 2KCy = KAgCy_2 + KNO_3$$

(2) Zinc-potassium cyanide

$$^{*}2AgNO_3 + 4KOH + K_2ZnCy_4 = 2KAgCy_2 + Zn(OK)_2 + 2KNO_3 + 2H_2O$$

In each case the end point is indicated by the precipitate of silver cyanide formed by the excess silver nitrate on the potassium silver cyanide.

If sufficient acid be carefully added to an alkaline plant solution so that neutrality, as far as protective alkalinity is concerned, is obtained, then none of the cyanogen existing in the form of simple cyanides of the alkaline earths, or the double cyanide of zinc and an alkaline earth, will be affected; and a titration can then be made with neutral potassium iodide as indicator, which will give an estimate of free cyanide present. On the other hand, the raising of the alkalinity by the addition of excess caustic alkali will not affect the behaviour of the standard solution toward the free cyanide present; but the cyanogen existing in the form of the double cyanide of zinc and an alkaline earth will, together with the hydrocyanic acid and in presence of the excess alkali, postpone the end point until all the cyanogen existing in the form of free cyanide, hydrocyanic acid, and double cyanide, has been recorded.

The following results of the careful addition of weak acid to a plant solution carrying free cyanide and double cyanide are of interest; and serve to demonstrate that there is no loss of cyanogen.

Series 1.

	Prot. alk., Per cent.	KCy, Per cent.
1	Neut.	0·125 (free)
2	0·005 (NaOH)	0·137
3	0·007 ,,	0·138
4	0·010 ,,	0·140
5	0·012 ,,	0·140 (total)

Original solution tested with alkaline KI and $AgNO_3$. Result, 0·14% (KCy). Protective alkali then neutralised with $N/10$ HNO_3; then gradually raised with NaOH. Solution tested with $AgNO_3$ and neutral KI at definite intervals of alkalinity.

Series 2.

	Prot. alk., Per cent.	KCy, Per cent.
1	Neut.	0·160 (free)
2	0·003 (NaOH)	0·162
3	0·006 ,,	0·165
4	0·008 ,,	0·170
5	0·011 ,,	0·175 (total)

Original solution tested with alkaline KI and $AgNO_3$. Result, 0·175% (KCy). Solution then treated as in Series 1.

The necessity for adequate solution control work was exemplified in a recent experience. Routine titration for 'cyanide strength' had been made regularly and showed no material alteration in the composition of the solutions. Precipitation went from bad to worse in spite of the adoption of all the usual methods of regaining control. The solution was then analysed and was found to contain a high zinc content and also a considerable amount of copper. Acid treatment was tried for the purpose of removing the double salts, but was abandoned because no satisfactory method of application could be evolved. The use of potassium permanganate was also tried and abandoned on account of the precipitation of the gold in considerable quantities (as with the acid method) together with the zinc and copper. The difficulty of collecting the precipitate from either reaction, or of doing anything with it when collected, prohibited the practical utilization of either method. Electrical precipitation, on an experimental scale, was also tried and the idea abandoned.

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*Clennell, "Chemistry of Cyanide Solutions" p. 37.

Solution Control.—The question ultimately resolved itself into one of solution control. As the source of copper was temporarily at an end the free cyanide strength of the solution was dropped to a minimum and the bulk of the copper allowed to deposit on zinc shavings, the latter being removed and replaced by fresh zinc as soon as thoroughly coated. This took about two days, during which time there was practically no precipitation of gold. As soon as all the copper had been removed in this manner the free cyanide content in the solution entering the zinc-boxes was raised considerably, the idea being that a point of saturation had been reached with the normal strength and that further solution of the zinc was impracticable without an increase in the normal free cyanide content. This solution of the zinc was considered necessary for the formation of the hydrogen which, in a nascent form, was an essential element in the breaking up of the potassium gold cyanide in the solution entering the zinc-boxes.

The outgoing solution showed an immediate drop in valuable content, and as soon as satisfactory precipitation had been secured the discharging of barren solution with the residue was commenced and the usual water wash abandoned. By these means the excessive amount of zinc was removed from the solution, the coppery zinc was gradually replaced in the boxes, and the free cyanide content was allowed to drop to normal without any interference with the efficiency of precipitation. In subsequent operations a mixed solution and water wash was used, and sufficient barren solution (amounting to about 15 % of the tonnage treated) was discharged with the residue to prevent the accumulation, above a workable limit, of undesirable compounds.

Zinc Losses. It is interesting to note in passing that the quantity of zinc so discharged in the residue is but a small fraction of the amount which disappears daily from the plant. The loss of zinc from plant solution is a mystery, which has yet to be definitely solved. Virgoe suggested the formation of an insoluble double carbonate as a result of a possible combination of zinc potassium cyanide with atmospheric carbon dioxide. Whatever the cause, it is evident that the automatic reduction of zinc content prevents the fouling of cyanide solutions in the great majority of cases. Under other conditions, however, as in the present instance, the reaction stops at a certain point, after which active measures must be taken for the removal of an undesirable accumulation of zinc. The strength of the cyanide solution entering the zinc-boxes must often be increased in proportion to the amount of zinc in such solution; and an unnecessary consumption of both solvent and precipitant results from an endeavour to effect satisfactory precipitation from foul solution.

As soon as facilities were secured it was deemed advisable to go more into detail in the matter of solution control, especially with regard to the estimation of zinc content. Independent tests were then made for total cyanide, free cyanide, and protective alkali. An additional test for hydrocyanic acid was also included and a modification of Bettel's* was decided upon. The result of this test gives free cyanide plus hydrocyanic acid, which is practically an available cyanide result in the absence of protective alkali.

A resumé of the complete series of tests is as follows:

Estimation.	Principal Salts indicated in Results.	Preparation of Solution	Indicator.	Titrated with	End Point.
Total cyanide	$KCy, NaCy, CaCy_2, K_2ZnCy_4, HCy$	Add excess caustic alkali	KI	$AgNO_3$	Yellow opalescence
Protective alkali		Add K_4FeCy_6: then slight excess $AgNO_3$ over that required in previous test	Phenolphthalein	$N/10\ HNO_3$	Disappearance of red colouration
Free cyanide, *i.e.*, available cyanide in presence of protective alkali	$KCy, NaCy, CaCy_2$	Add $N/10$ acid till protective alkali is neutralised: *i.e.*, amount shown necessary in previous test	Neutral KI	$AgNO_3$	Yellow opalescence
Available cyanide, in absence of protective alkali	$KCy, NaCy, CaCy_2, HCy$	Add $NaHCO_3$	Neutral KI	$AgNO$	Yellow opalescence

The effect of dilution in the titration for total cyanide of a solution carrying the double cyanide has been discussed by other investigators. As a means of emphasising the necessity for strict uniformity of operation, the following is instructive.

Total Cyanide Estimation Result.

Plant solution:	(KCy), %
Undiluted	0·167
Diluted with equal bulk of water ...	0·172
Diluted with twice bulk of water ...	0·175
Diluted with three times bulk of water	0·175

Experiments made by me showed that reverse titrations for total and free cyanide gave almost

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*Proc. *Chem. Met. & Min. Soc. of S. A.*, Vol. I. p. 165.

identical results as direct ones, indicating that the amount of silver nitrate added by the ordinary method in quantity sufficient to give a distinct end-point, although in actual excess of the amount required to satisfy the equation, gives for all practical purposes a substantially correct estimate.

Protective Alkali.—Green's method, as outlined, seems to be the most satisfactory, provided strict uniformity is observed in the amount of excess silver nitrate added.

It has been found preferable in estimating protective alkali under conditions of low alkalinity to count the number of drops of dilute acid necessary to neutralize a small amount of cyanide solution, rather than to attempt to read the burette. In the subsequent estimation for free cyanide the slow addition of acid is essential; and this is best made by adding the exact number of drops shown to have been necessary by the preceding test.

The free cyanide method is one of many given by Clennell,* and in my opinion, gives uniformly satisfactory results.

Hydrocyanic Acid.—The bicarbonate should be added only in moderate excess, and a check test occasionally made to correct for the presence of carbonate. The inferred definition of 'available cyanide' in connection with the result of this test is consistent with my experience of actual practice.

In all of the above tests for cyanide the formation of a cloudiness inevitably preceded the appearance of distinct opalescence. The latter indication has been looked upon as the only reliable end point. In the present instance control testing is not considered in the light of an analysis of the solution, but rather calls for a quick and sufficiently reliable method of arriving at results of comparative accuracy.

The difference between the free cyanide result in the presence of protective alkali or the available cyanide result in its absence, and the total cyanide result, gives an approximation of the amount of the double salt present, in terms of potassium cyanide.

Volume.			Cyanide (as KCy) Total, %.	Cyanide (as KCy) Free, %.
50% working solution	...	..	0·114	0·057
50% added alkali	..	...	...	...
Numerical average	..	...	0·057	0·028
Titration average	...	...	0·057	0·028

One fourth of this amount gives an estimate for zinc in the form of the double cyanide.

Use of Burettes.—Standard solutions, indicators, and correctives, and are all measured from burettes. These are mounted on revolvable stands with the result that the complete series of tests can be made very rapidly: and absolute uniformity of condition is assured for each estimation.

The final estimation for zinc content as double cyanide has been checked by analysis for total zinc in the same solution made by the sodium sulphide, ferric sulphate, and potassium permanganate method. The direct titration results were slightly higher than those from analysis, the greatest difference amounting to 0·005 %. These differences were consistent and comparisons were made from solutions carrying variable amounts of zinc. The results indicated the comparative accuracy at least of the direct titration method on the solution in question.

Acidity and Regeneration.—At one time I made some experiments to determine whether it might be possible, under ordinary conditions, for the double zinc salt in a plant solution to be locally decomposed in contact with an acid ore or an acid water, according to the equation:

$$K_2ZnCy + H_2SO_4 = 2HCy + ZnCy_2 + K_2SO_4$$

The conclusions arrived at were that such decomposition was possible in a solution of low free cyanide content; but that the hydrocyanic acid formed is so entangled with the precipitate of zinc cyanide that subsequent contact with alkali would result in a reversion to the original compound thus:

$$ZnCy_2 + 2HCy + 2KOH = K_2ZnCy_4 + 2H_2O$$

rather than conversion of the hydrocyanic acid into a cyanide of the alkali.

Alkalinity and Regeneration.—The equation upon which the suggestion of regeneration of free cyanide by means of alkali is as follows:

$$K_2ZnCy_4 + 4KOH = Zn(OK)_2 + 4KCy + 2H_2O$$

This equation must be assumed in all cases where the simple titration of an alkaline working solution is expected to indicate free cyanide content.

According to Clennell* the alkaline equivalents should be as follows:

			Methyl Orange.	Phenolphthalein.
K_2ZnCy_4	..	...	4KOH	...
4KOH	...	...	4KOH	4KOH
$Zn(OK)_2$	...	...	2KOH	2KOH
4KCy ...	...	...	4KOH	4KOH

The addition of alkali to a protectively neutral solution carrying the double cyanide should therefore result in an increase in the sum total alkalinity to phenolphthalein, and a decrease in the sum total alkalinity to methyl orange. A number of tests that I have made would seem to indicate that the supposed reaction does not occur. The end points with the phenolphthalein are slightly uncertain, but in no instance was it possible to show an increase of alkalinity to this indicator. It was equally impossible to show a decrease in the alkalinity due to the methyl orange.

The figures in the following test will serve to indicate the method employed:

Alkalinity (as KOH) To Phenolphthalein. Prot., %.	Alkalinity (as KOH) To Phenolphthalein. Total, %.	Alkalinity (as KOH) To Methyl Orange. Total, %.
...	0·015	0·115
0·052	0·052	0·054
0·026	0·033	0·084
0·026	0·031	0·084

In the above test the amount of alkali added was calculated so that the recognized direct titration method of testing, as advocated in the most recent text-books on metallurgical practice, would indicate the whole of the cyanogen content of the solution as free cyanide.

Mechanical Mixture.—In considering the figures it is interesting to note that as regards total alkalinities, the figures favour the assumption that nothing more than a mechanical mixture has taken place. It is also noticeable that the actual titration average of the protective alkali is in agreement with the numerical average. Had regeneration occurred there would have been a reduction in protective alkali content, free alkali being used up to complete the reaction and the alkaline free cyanide so formed being reduced to neutrality as a result of the addition of silver nitrate, after the ferrocyanide, in the protective alkalinity test.

Another point worthy of notice is that the neutral solution, free or available cyanide result, minus the alkaline solution, total cyanide result, gives a definite idea of zinc content. The direct titration method is of no value in this connection and, as a result, the fact is often overlooked that a solution may be as foul with zinc after the addition of alkali as before it. This quantity of zinc in solution is an important factor in the efficiency and economy of operations. Its presence invariably affects the precipitation and sometimes the solution of gold, and a regular estimation of its content is second only in importance to an estimation for free or available cyanide. The cumulative ill-effects of adding cyanide in lumps at the head of the zinc-boxes in an effort to improve precipitation is indicated by the subsequent determination for zinc content. The system is an unsatisfactory palliative rather than an actual remedy, precipitation only being effected by an increased addition of zinc and a further fouling of the solution.

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* "Chemistry of Cyanide Solutions," pp. 25 and 26.

* "Chemistry of Cyanide Solutions," p. 60.

Satisfactory precipitation, from an economic point of view, is more a question of ultimate value than actual metal content. The 'barren' solution after gold precipitation may contain a few grains per ton as compared with a corresponding 'barren' solution after silver precipitation which may contain as many pennyweights. Each may be referred to as the residue after satisfactory precipitation, but it is obvious that the question of zinc content is more important in the case of the treatment of gold ores where a complete precipitation is essential, a small percentage of the metal representing a high money value. In either case the precipitation is at maximum efficiency at the commencement of operations and deteriorates with the accumulation of zinc in the solution.

In conclusion it may be added that, in piecing together the above notes, a deal of recognized information on zinc precipitation has been included for the sake of clarity. I wish to acknowledge the fact and would make especial reference to J. E. Clennell, whose hand book on cyanide solutions is of especial interest and value."—A. W. Allen, *Mining and Scientific Press*, Sept. 20, 1913, p. 448. (H. A. W.)

MINING.

Sand Filling.—"The extraction of pillars of rich ore, and permanent support of the old workings, has been a problem for most mine managers from the days of comprehensive lode mining. The most general practice has been the close timbering of heavy ground with expansive timbers, but this has prevented in most instances the subsequent mining of portions of the ground, and compelled the management to abandon numerous pillars of rich quartz, as the timbers have needed the support of the column of rock to sustain the tremendous weight of the hanging-wall. Not only has the adequate support of the underground workings compelled the constant attention of the manager during the active life of the property, but even after the passing of its productive period, because of surface disturbances. The subsidence of the old mine workings has often endangered portions of towns and cities, and legislation has been frequently threatened against the mine owners to prevent trouble of this character. The first grade of timber is certain to weaken and collapse under the crushing strain of millions of tons of settling earth in the course of years even when supporting pillars are permitted to carry the greater portion of the strain. In many of the Rand mines, and numerous American properties, it has been found that the ore pillars show decided signs of crushing after carrying the prodigious weight for some years, even with the reinforcement of close sets of timbers. This is particularly marked when the vein-system has a fairly steep dip. Various expedients have been proposed to overcome the problem, and in several instances efforts have been made at its solution by employment of steel timbers and reinforced concrete supports. Objections to such practices have developed, but the method has generally proved satisfactory when compared with old-time provisions. It has fittingly remained for the great Rand mining field of the Transvaal to devise and perfect a method at once satisfactory and inexpensive—a method that has so convincingly demonstrated its merit that its use is becoming general throughout the greatest gold field of the world on the Rand. Mining is conducted on a stupendous scale hardly appreciated in other fields, and the question of adequately supporting the mine workings at low costs has developed into a vital consideration. Practically all the premier companies have devoted special attention to the subject and after years of systematic experiments sand-filling has been selected as the ideal solution of the vexing question. The employment of the method has been particularly marked during the past two years, and its use is making rapid advances throughout the entire field. The success of the experiment on the Rand has claimed some attention from American operators, but is deserving of a better reception than has been generally extended it on this side of the hemisphere. American mine managers are proverbially slow to adopt the practices developed in foreign fields, which may account for the apparent indifference accorded the method developed by the Transvaal operators. One of the most successful of the sand-filling plants on the Rand is that operated by the Witwatersrand Deep, Ltd., which has been in commission about eight months. The plant was designed after the method had been thoroughly tried out by several other operators, and embodies all the good features of earlier installations, together with many original improvements of merit. The mill of this company crushes about 38,000 tons of ore per month and 60 per cent. of the reduced product is sand, or approximately 22,000 tons. All of this is turned back into the mine for filling of old stopes and other workings. The Witwatersrand Deep claims are traversed by the great East Rand dyke which practically cuts the property into north and south sections. In both portions of the mine sand-filling is proceeding, and not only has it facilitated removal of the rich pillars hitherto employed as supporting agents, but also enables the operators to mine large areas of ground formerly inaccessible under ordinary working conditions. The sand is sent from the surface into the northern portion of the mine through a hole bored for the purpose, and the south section is supplied by way of a winze. At the receiving terminals are placed rows of tanks commanding belt-conveyors. These deliver the product of two sludge pumps where the sand is mixed with about four times its own weight of water. In this state it is too wet for direct loading into the workings and first passes to the dewatering station where six Caldecott dewatering cones, for each station, reduce the moisture to about 28 per cent. The amount of water to be used forms an important point, and varies somewhat in different properties because of natural conditions. Without a sufficient percentage of water the sand cannot be effectively delivered to the various portions of the mine, and if the amount of water is excessive it must be pumped out again. The quantity used at the Witwatersrand was decided on after a comprehensive series of tests, and has proven satisfactory in this particular instance. The costs attending surface handling of the sand are approximately the same as are entailed in dumping the residue, whilst underground costs are slightly over 4 cents per ton. It is thus readily apparent that the process is fairly inexpensive. As a result of the installation the management states that the safety element has been materially increased, and damage to surface buildings by sliding ground virtually eliminated. It has also enabled the carrying on of operations on a larger scale, inasmuch as more extensive areas of ground may be worked at once without danger of caving. Besides, it permits the extraction of columns of commmercial quartz formerly left to aid the timbers in sustaining the weight of the hanging-wall. In the old workings of this property several of the pillars show signs of yielding to the excessive strain and sand-filling has proven far superior to timbers in assisting the pillars in bearing the load when it is not deemed advisable to extract the supporting columns. The operation of the sand-filling plant is practically automatic throughout, and requires little attention, save at the surface loading stations and points of application. The process employed by the Witwatersrand Deep Co. is the one most generally favoured by Rand operators, and may be considered a standard method of sand-filling. The sand is taken direct from the tanks, and the free cyanide neutralised by feeding potassium permanganate into the pulp. Tests of the treated product are made regularly to detect any trace of free cyanide and prevent its passage into the mine workings. It is essential that the sand used in filling be sufficiently moist to pack well, as too dry a product is more difficult to handle and flows through the supplying pipes and launders less readily. Yet the product must not be too moist, or the necessity of pumping out the excess water means loss of time and added expense. While the management of the Witwatersrand Deep use a mixture of seventy-two parts sand and twenty-eight parts water, the property is an exceedingly wet mine and the use of a very moist sand is carefully guarded against. The company is pumping 1,500,000 gallons per day at present, and a few months ago was handling 2,000,000 gallons. The excessive wetness of this mine is attributed to the big transverse dyke which cuts through a portion of the Witwatersrand Deep and neighbouring properties. Within a short time it is expected the powerful new pumping system of the East Rand Proprietary Company will relieve the Witwatersrand Deep of a considerable portion of the water, in which event it is possible the proportion of water used in the sand-filling will be increased. In many of the dry mines of America it would probably be found desirable to employ a mixture containing considerably more water to insure best results. Sand-filling may be carried on very much as ordinary timbering, as the timberman in charge of the work completes the placement of the sand in the workings as the ore is removed. In this way there is no necessity for leaving large open chambers unsupported for any length of time, and caves or movements of ground towards the shaft are effectively controlled. The total cost of sand-filling on the Rand varies from 10 to 24 cents per ton, including surface handling. It costs about 10 cents per ton to handle the sand from the vats and store on the dumps in ordinary practice. It is thus apparent that the sand can be sent down to the mine levels almost as cheaply as it can be stored on surface. The danger of caves in deep mines, and resultant airblasts of terrific severity, is intensified by the room and pillar method of mining and similar practices, and it is under such conditions that the use of sand for filling of the old working claims particular attention. The pillar and room method means large open spaces, and a large open stope is a source of positive danger. In many mines worked by this method the men labour under cover, the guarding pillars protecting them from sudden caves of the hanging-wall. The pillars are subsequently removed by top-slicing. But there is always the possibility of the ground crushing down the supports, unless a large number of pillars are provided. The method is dependent for success upon the bringing down of the capping evenly and regularly, and it is economically necessary that the maximum quantity of ore be caved down in the shortest possible period of time. The method has been developed along particularly successful lines in the Lake Superior district, and it is notable that airblasts are very common in this region. As before stated, an airblast is caused by intense compression of air in a confined space. The fall of an enormous tonnage of rock into a large empty stope hurls a crushing wave of compressed air through the mine very much as a similar wave is set in action by a gas or dust explosion in a coal mine. The rushing blast of air sweeps timbers, cars and men along with tremendous force, and such blasts have often caused heavy loss of life and considerable damage to property. It is not to be concluded that airblasts are always the result of caving methods of mining, for disastrous blasts of this character have frequently occurred in more restricted workings. They were not unknown on the Comstock when that famous lode was at the zenith of its glory, and have developed in most of the deep mining regions of the world. A convincing demonstration of the terrific force of an airblast, and its deadly properties, was recently evidenced at the Miami copper mine, where three men were killed, seven seriously injured, and others hurt in minor ways by the swirling blast of air driven from the 245-foot level by the collapse of capping estimated to comprise 3,000,000 tons. In this mine the men work under protecting pillars, and the falling rock itself caused little damage. But the cave drove the compressed air forth at terrific velocity, carrying death and destruction into the nearby drifts. The resistless nature of the blast was evidenced by the driving of a 7-ton motor and 15 ore cars along the track for 200 feet, despite the desperate efforts of the motorman to check the terrific force of the rushing wind. In foreign fields the airblast has been a most destructive agent, and the frequent occurrence of such accidents on the Rand was one of the prime reasons for adopting the sand-filling method. The danger of airblasts has been given little consideration by metal miners in the past, although numerous precautions have been taken to guard against the danger of falling masses of rock. The series of airblasts recently occurring in many districts, however, particularly in deep mines and where large stopes are worked, have awakened managers to the presence of an element of peril that must be guarded against as carefully as other potential sources of danger. It is universally conceded that the most dependable guardian against the airblast is an adequate support of the roof of the underground workings, but conditions are frequently of such a nature that close timbering is economically impracticable. In many districts timbers are difficult to secure, and costly to instal, while the margin of profit attendant on mine production is so limited that the management does not feel justified in assuming further expenses. Under the present exigencies of the commercial era, each mining company is endeavouring to rush production. Practically every progressive manager is striving to extract the largest possible quantity of ore within a specified time, and the earning of maximum profits forms the chief consideration. But such a practice has its drawbacks and penalties, and the insufficient support of heavy ground too often entails subsequent costs and delays. When the lessons of Rand mining are studied and analysed, and it is realized that the sand-filling method has only been adopted after years of comprehensive and intelligent trials by the leading Transvaal companies, it seems strange that the practice has not been given greater approval by American operators. The method was not employed by the Rand companies before its merit had been definitely established, and while most managers admit there are many opportunities for improving the system, it has incontestably saved an immense sum to several operators, and facilitated extraction of ore from sections of ground previously inaccessible. Not only has sand-filling proved of inestimable benefit in protecting the mine from the effects of disastrous caves, but has also increased the important factor of personal safety whenever employed. And the human factor, the adequate protection of the employees, must be considered. Unless the miners know they are working in a fairly safe stope, with all possible precautions exercised to secure them from accidents, they cannot be expected to remain satisfied with conditions, nor to do their best work. No man can give his best when he knows that danger constantly hovers near. And no man will work for long in a property he knows to be absolutely unsafe. Furthermore, it is the duty of the company to protect its men to the utmost of its ability. And as a means of protection the sand-filled workings have demonstrated their worth. The subject is commencing to claim some attention in America, and has been given a trial in some instances, but has yet to be given the support by the mining fraternity to which Rand results prove it to be entitled." A. H. MARTIN, *Mexican Mining Journal*, Oct. 1913, p. 177. (H. A. W.)

ACCIDENT WITH A BENZOLE LOCOMOTIVE AT THE ROSSLEBEN POTASSIUM SALT-MINES.—"In front of a pillar-and-post working place eight workmen were rendered insensible by fumes from a benzole locomotive. One of them died. The investigation instituted showed that the accumulation of gas in part came from the benzole locomotive, the machinery of which had been running disconnected for about an hour. Owing to a loose screw in the mechanism which produced the ignition, the latter frequently missed, and thus benzole flowed into the workings. Meanwhile, a ventilator intended to assist the current had, owing to insufficient supply, drawn air through an untight brattice, instead of round it, and thus produced short-circuit. Carbonic-oxide gas, which issued in small quantities from the salt, had also collected, and the two gases—benzole and carbonic oxide—had together overcome the men. In future, the running of the machinery without load is to be avoided as far as possible. In regard to the ventilator, care is to be taken that, in carrying the current to remote places in connection with brattices, at least the quantity of air suitable for the suction is provided." ZIEBARTH (*Zeitschr. f. Berg., Hütt.- u. Salinenw.*, 1913, Vol. lxi., p. 214-222), *Transactions Institute of Mining Engineers*, July, 1913, p. 756. (E. P.)

IRIDOSMINE FROM THE NEW RIETFONTEIN MINES.—"This mineral, which has been found in several South African gold mines, is an alloy or intimate mixture of Ir and Os with generally some Ru. It was collected from battery concentrate, black sand residues, and die sand, where, because of its high sp. gr., it was with difficulty separated from Au. The amounts found were too small for it to be of other than scientific interest. A quantitative analysis of a concentrate, after removing the Hg used in amalgamating gave Au 1·24, Pt 0·61, Fe (as Fe_2O_3) 1·29, Ni (as NiO) 0·66, and 'Osmiridium' 95·52, the latter containing about 45% of Ir, which indicated that the mineral was essentially iridosmine. Part of the Fe was in the free state and probably came from the stamp used in crushing the ore. The iridosmine and associated metals have not been recognised *in situ* underground nor in hand specimens; they are associated with Au and pyrites; they seem most likely to occur in that auriferous banket reef known as Carbon Leader, which is distinguished by the abundance of C, and further they carry Ni. These metals are of secondary origin and occurred in minute proportions as primary segregations formed by magmatic concentration in basic eruptives. By the last or pneumatolytic phase of eruptive activity they were extracted from the dikes by superheated gases. Later hydrothermal action probably played an important part in concentrating them in the banket reefs."—C. BARING HORWOOD, *Chem. News*, *107*, 230-2, 244-5, 253-5; *Chemical Abstracts*, Vol. 7, No. 17, Sept. 10, 1913, p. 2917. (J. G.)

EXPLOSIVE EFFICIENCY.—"This article, taken from the *Mexican Mining Journal*, (17, 344-6) places special emphasis on the use of strong detonators. No. 8 detonators are recommended for gelatine explosives and at least No. 6 for other grades. Strong detonators ensure complete detonation and therefore greater efficiency." —ANDERSON, *Chemical Abstracts*, No. 20, 20/10/13, p. 3510. (T. D.)

WORKING COSTS IN CANADA. "The costs of the Hollinger Gold Mines, Porcupine, Ont., for February, 1913, covering 9,250 tons, were as follows:—

Hollinger Mining Costs.

Account.	Costs per ton. Ore Milled.	Per Cent. Labour.	Per Cent. Supplies.
General mining charges	$0.053	75	25
Superintendence ...	0.126	100	...
Diamond Drilling ..	0.027	65	35
Crosscutting	0.043	36	64
Drifting	0.338	60	40
Raising	0.007	66	34
Stoping	1.581	54	46
Timbering stopes ...	0.219	90	10
Track laying	0.013	58	42
Tramming and mucking	0.438	99	1
Pipefitting underground	0.042	34	66
Mine drainage	0.050	28	72
Hoisting	0.170	54	46
Landing and dumping	0.063	100	...
Drill repairs	0.017	86	14
Drill sharpening ...	0.111	98	2
Distributing steel ..	0.050	100	...
Mine sampling	0.041	95	5
Assaying . ..	0.017	47	53
Change house	0.008	100	...
Mine lighting	0.005	65	35
Handling explosives ...	0.025	98	2
Handling waste ...	0.016	100	...
Surveying	0.008	100	...
Total	$3.588	65.5%	34.5%

Hollinger Milling Costs.

General mill charges ...	$0.160	75	25
Superintendence ...	0.075	100	...
Heating	0.076	55	45
Lighting	0.020	14	86
Watchman . . .	0.009	100	
Sampling	0.006	100	..
Assaying	0.033	48	52
Coarse crushing ...	0.073	61	39
Conveying	0.034	73	27
Stamping	0.181	47	53
Classification and tube milling	0.277	19	81
Concentration	0.057	79	21
Handling concentrates	0.012	25	75
Grinding concentrates	0.036	48	52
Handling pulp	0.022	35	65
Thickening	0.012	82	18
Agitation	0.029	38	62
Filtration	0.147	67	33
Acid washing	0.013	77	23
Neutralizing	0.023	52	48
Classification and precipitation	0.074	15	85
Smelting and retorting	0.063	26	74
Pumping solutions . .	0.020	61	39
Cyanide	0.041	...	100
Alterations	0.200	52	48
Total	$1.693	49%	51%

Total Working Costs.

	Per ton Milled.
Administration, management, insurance, etc.	$0.407
General charges	0.209
Clearing surface roads, etc.	0.015
Mining	3.588
Milling (not including alterations)	1.493
Operating camp	0.261
Loss on temporary boarding houses	0.042
Alterations to mill and plant	0.221
Strike expense	0.508
Total	$6.744

The costs are expected to be brought down to approximately $5.50 per ton under normal conditions. The wages paid range from $3.25 to $3.75 per day for skilled labour and from $2.50 to $3 for unskilled. Eight hours are worked in the mill and nine in the mine. Bed and board are furnished for 60c. per day."—*Engineering and Mining Journal*, Oct. 18, 1913, p. 739. (A. R.)

MISCELLANEOUS.

RAND TERMINOLOGY.—"As is generally known, the pioneers of the Witwatersrand adopted several Australian terms and called their lodes 'reefs.' Finding a type of ore with which previously they were unfamiliar, they borrowed a Dutch word, and called it 'banket.' Further, being more eager to make money than to preserve the English language, they spoke of their workings as 'drives,' not drifts: they sought 'high values,' meaning ore having a high assay-value: they delved in search of 'good gold,' not doubting that all gold was good, but desiring gold in goodly proportion. Even educated engineers borrowed the language of trammers and timber-men, so that ore was described as 'payable,' instead of profitable; waste became 'mullock;' shale was loosely labeled 'slate;' and the locality itself became 'these fields.'

'Banket' is a genuine addition to the English language. It is the word used by the Dutch for almond cake; it is significant and full of local colour. A 'banket' stands for a conglomerate containing sufficient gold, or any other valuable metal, to be exploited as an ore deposit. 'Banket' in its etymology harks back to the Dutch domination of the Transvaal, and to the great series of gold-bearing conglomerates or almond-rocks associated with the modern mining development of South Africa. It is convenient to use 'conglomerate' in petrography, and 'banket' in economic geology. Every banket must be a conglomerate, but not many conglomerates are bankets. By giving precision of meaning to words, and by setting aside special words for special duties, we gain accuracy of expression.

Another useful South African addition to technical English is 'deep-level.' The first properties developed from the surface were known as 'outcrop' mines; these had no extralateral rights, and were estopped from trespassing beyond their side lines projected vertically downward. The next mine on the dip of the lode became known as the 'deep-level' mine or 'deep' of the corresponding outcrop property; thus the Ferreira Deep exploits the ore beyond and below the limits of the Ferreira. In the same way the Tamarack, in the Lake Superior copper region, is the 'deep' or 'deep-level' of the Calumet & Hecla. On the other hand, the habit of writers on Rand geology to use such terms as 'reefs,' 'reef-matter,' 'values,' and the like, tends to obscure the basic economic feature, namely, that the conglomerate is 'ore.' We have read articles on the Witwatersrand in which the words 'ore' and 'lode' were absent to such a degree as to warrant the reader in forgetting, or failing to apprehend, that the deposits under discussion were the object of profitable industry. The word 'reef' was introduced into mining by the sailors and stewards who left their ships to participate in the rush to Ballarat and Bendigo in 1851. To them a rock projecting above the water was a reef, and when they saw outcrops of quartz they likened them to one of the perils of navigation. However descriptive the term may be when applied to an orebody emerging above the plain, it is wholly a misnomer for lodes that have no outcrop and for orebodies found far below the surface. Any significance that 'reef' may have in mining is entirely lost when applied to ore discovered or intersected far from sunlight, especially when, as in Rhodesia, it may be a mass either lenticular or without definable shape. Fortunately, it is a local term peculiar to Australia and South Africa. In other equally important mining regions it is not legal tender, but one of the many spurious verbal coinages of illiteracy. Nor is it required. We have 'lode' and 'vein.' Indeed, the use of 'reef' has tended to obscure the fact that the ore deposits of the Witwatersrand or 'white waters range' are not unique, except in their persistence. Beds of conglomerate containing gold have been found elsewhere in many regions. Gold is found enriching rocks of every kind, both crystalline and clastic. The deposits exploited in the environs of Johannesburg are lodes as much as, and no less than, those of other celebrated mining districts. A 'lode' is a deposit having such continuity of structure as will lead a miner to the discovery of ore. The word comes from the Anglo-Saxon word *lad*, from *lithan*, to go, and is related to *leadan*, to lead, another derivative from

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[*] Includes cost of repairing and lining three tube mills.

lithan. 'Vein' may not fitly describe the bed of conglomerate, because it is not typically a ramifying fracture or fissure filled or followed by ore, but a seam of banket is certainly more nearly a bedded vein than a 'reef,' in so far as these terms have any precise meaning. 'Drive' is an error when used as a synonym for 'drift.' The miner drives his working, which results in a 'drift.' 'Mullock' is a wretched Australian vulgarism that only obscures meaning. It stands for waste rock, the reject from ore. When the Australian sailors and shepherds introduced 'good gold,' 'poor gold,' 'rich gold,' and so forth, to express the relative richness of their ore, they garbled their speech. No need arises for educated men to adopt such illiteracies. Thus also 'values,' with 'high' and 'low values,' was originally an abbreviation for ore valuable by reason of its metallic contents. As now used it is only a misleading colloquialism. To seek values in a mine is comparable to the pursuit of a quadratic equation in an automobile. Whatever excuse there may be for carelessness in casual talk, there is none for the introduction and use of such phrases in scientific writing. Again we say that accuracy of thought is fostered by precision in the use of the vehicle of thought, namely, language, whereby we transfer ideas from one to the other.

'Shale' and 'slate' are different rocks. No educated engineer should use one term for the other. A 'shale' is fissile along its lamination, which is its line of original deposit. A 'slate' is fissile along lines independent of the stratification and usually at a high angle with it. 'Payable' is a mere vulgarism. Ore does not 'pay' nor is it able to pay; it yields a profit; it is profitable. Even if we overlook the use of an intransitive verb as if it were transitive, it remains a fact that ore that only 'pays,' that is, meets the expense of exploitation, has no economic worth. To have that it must yield a profit; it must be profitable. As for 'fields,' this also is scarcely worthy of castigation. It is bucolic. As an abbreviation from goldfields, it exemplifies the hankering for an inexpressive plural, the squandering of a useful inflection, and the popular fancy for the abstract as against the concrete. These matters of terminology seem petty, but they are not. Apart from the clarification of thought by the filtering of language, it is worthy of remembrance that the English language is the common heritage not of the Briton, the American, the Australian, or the Africander, each and severally, but of the 400 millions of English-speaking peoples on both sides of the equator and in both hemispheres. All of us share a priceless inheritance, our common literature, and it should be our aim not to curtail the usefulness of it, especially in technical science, by the employment of terms half understood locally and misunderstood everywhere else."—T. A. Rickard, *Mining and Scientific Press*, Nov. 8, 1913, p. 711. (H. A. W.)

New Fertiliser Factory in Cape Colony.—"H. M. Trade Commissioner for South Africa reports that a company has been formed in Cape Town, with a capital of £20,000, for the manufacture of fertilisers. The company proposes to deal with fish caught in Table Bay and with the offal therefrom, a regular supply of not less than 50 tons of raw material being assured. Works are to be established at Woodstock."—H. M. Trade Commissioner, *Oil and Colour Trades Journal*, Nov. 1, 1913, p. 1552. (J. W.)

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Abstracts of Patent Applications.

(C.) 73/13. George Thomas Hill (1), Charles Graham Stone (2). Improvements in or appertaining to the manufacture or preparation of Portland cement. 13.2.13.

This application relates to the addition of magnesium silicate to Portland cement for the purpose of making cement structures water-proof. The subsidiary claims refer to the percentage of magnesium silicate added, and various means of adding and preparing the silicate and incorporating it with the cement.

(C.) 338/13. Wm. Blane Dodds (1), David Charles Bowen (2). Improvements in linings for the ends of tube mills and the like. 16.7.13.

This application relates to improvements in linings for ends of tube mills, and consists in providing transversely, tapered, radially disposed bars, carried by a fixed supporting piece and filling in between with flint or other suitable material.

The object of this invention is to form a number of dovetailed or undercut recesses, in which the "filling" is securely retained, or in the event of one or more of the blocks falling out, of limiting the effect to a small section.

(C.) 358/13. George Stott. Improvements in rollers for mine shafts, haulage ways, and the like. 23.7.13.

This application relates to an improved form of roller for carrying ropes on haulages or incline shafts. The ends of the rollers instead of being castings are of steel plate, and the shell of the roller is carried by and secured to lugs punched from the solid metal and bent inwards. The boss of the end plate is also formed by punching the plate outwards.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 579/13. Douglas William Stacey. An improved process and apparatus for the separating of a solution containing gold or other metals or material from finely crushed insoluble substances. 3.12.13.

(P.) 580/13. Gysbert Christiaan William Keet. The road splitter. 4.12.13.

(C.) 581/13. Guglielmo Marconi (1), Marconi's Wireless Telegraph Company Limited (2). Improvements in wireless telegraph transmitters. 4.12.13.

(C.) 582/13. Hans Andrew Hansen (1), Joseph Milton Schmueser (2). Improvements in linings for ball or pebble mills. 4.12.13.

(C.) 583/13. Thomas James Ehrhart. Improvements in lifting rigs. 5.12.13.

(P.) 584/13. Thomas Hurst. Electrical recorder for mine bell signals. 5.12.13.

(P.) 585/13. John Drinnan. Improvements in plug cocks. 8.12.13.

(P.) 586/13. Thomas Harrison. A new type of roller. 9.12.13.

(P.) 588/13. Alfred Edwin Davis. Improvements in and relating to apparatus for transporting material in mines. 9.12.13.

(C.) 589/13. Arnold Albert Musto. Improvements in percussive boring apparatus. 11.12.13.

(C.) 590/13. Walter Brewitt. Improvements in the process of welding rails or the like and apparatus to be used therein 11.12.13.

(C.) 592/13. Henry Squarebrigs Mackay. Process for extracting metals from low percentage ores. 11.12.13.

(C.) 593/13. Carl Davenport. Winding and haulage drums. 11.12.13.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

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Donkin, W., *l/o* England; Shamva Mines, Ltd., Shamva, S. Rhodesia.

Guthrie, F. G., *l/o* Cape Town; Consolidated Goldfields Laboratory, P. O. Box 108, Germiston.

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Vol. XIV. FEBRUARY, 1914. No. 8.

Proceedings
AT
Ordinary General Meeting, February 21st, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, February 21st, 1914, Mr. Alex. Richardson (President) in the chair. There were also present:

32 Members: Prof. G. H. Stanley, Messrs. J. E. Thomas, H. Meyer, C. Toombs, John Watson, A. Whitby, Prof. J. A. Wilkinson, Dr. W. A. Caldecott, A. F. Crosse, W. Cullen, W. R. Dowling, A. McA. Johnston, Dr. J. Moir J. R. Williams (Members of Council), R. W. Bennett, H. C. Boydell, J. Chilton, E. H. Croghan, R. Dures, H. W. Gill, B. C. Gullachsen, E. E. Hardach, J. T. Milligan, S. Newton, E. A. Osterloh, F. D. Phillips, J. C. Phillips, W. S. V. Price, D. F. Scott, P. J. Smith, A. L. Spoor and A. Thomas.

9 Associates and Students: Messrs. O. A. Gerber, C. Greathead, W. Human, L. T. Leyson, F. J. Pooler, H. Rusden, B. C. Smith, H. Stadler and H. Ward.

13 Visitors, and Fred. Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

NEW MEMBERS.

Mr. Wm. Cullen and Dr. James Moir were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected, as follows:—

Browne, Cyril H., City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Battery Manager.

Cooper, Richard Austin, East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand. Analyst.

Daveler, Erle Victor, Alaska Gold Mines Co., Juneau, Alaska, U.S.A. Superintendent of Mills.

Dealy, John, City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Cyanider.

Dennison, John Albert, M.Inst.M.M., A.M.I. Mech.E., Anglo-French Exploration Co., Ltd., 208-224 Salisbury House, London, E.C. Mining Engineer.

Dewar, Thomas Nimmo, East Rand Proprietary Mines, Ltd. (Angelo), P. O. Box 65, East Rand. Head Surveyor.

Fletcher, William Norman, City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Cyanider.

Gray, Charles John, Mines Department, P. O. Box 355, Krugersdorp. Deputy Inspector of Mines.

Hardy, Joseph Alexander, Warren Hill G. M. Co., Ltd., Klerksdorp. Reduction Foreman.

Hess, Rush Miner, P. O. Box 655, Guayaquil Ecuador, South America. Mining Engineer.

Philpott, Roy, C., P. O. Box 655, Guayaquil Ecuador. South America. Mining Engineer.

Thomas, Henry, East Rand Proprietary Mines, Ltd., 450, Double Storey Quarters, Vogelfontein. Assayer.

Williamson, Howard Baker, City and Suburban G. M. Co., Ltd., P. O. Box 1026. Johannesburg. Cyanider.

The Secretary announced that since the last meeting of the Society the following Associates and Students had been admitted by the Council:

As Associates—

Rodden, William, New Big River, Reefton, New Zealand. Mill Superintendent.

Waterson, Henry, East Rand Proprietary Mines, Ltd., P. O. Box 33, Boksburg. Chemist and Assayer.

As Students—

Bradley, Edward Gilbert, East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand. Laboratory Assistant.

Frielinghaus, Frederick Victor, East Rand Proprietary Mines, Ltd. (Cason), P. O. Box 80, East Rand. Mill Learner.

Patterson, Harry, 52, Westfield Road, Edgbaston, Birmingham. University Student.

General Business.

INVITATION TO VISIT DE BEERS.

The President: We have received a hearty invitation from the management of the De Beers Consolidated to pay a visit to the Kimberley diamond mines. Early in July will be the most convenient time for the company; and we hope to be able to arrange a visit of the Society during that month.

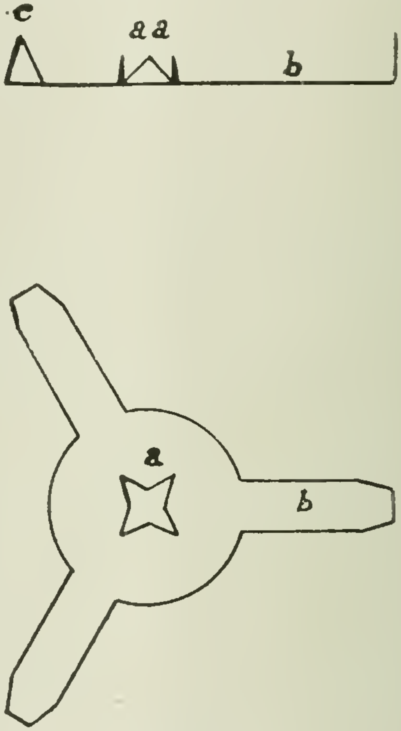
Mr. Wm. Cullen (*Past-President*): I should just like to say that, at the same time as you have been invited to Kimberley, the Annual Congress of the South African Association for the Advancement of Science and the Annual Medical Congress will be held there.

PRIMER CLIP FOR EXPLOSIVES.

Mr. Wm. Cullen (*Past-President*): Before I proceed to a description of the little device which I am placing before you to-night, I wish to read the following extract from the Mines, Works and Machinery Regulations, 1911 (Amended, 1913), p. 64, Chap. IX., Sec. 106, Sub-Sec. (19):—

> "shall in making up a charge, after the "fuse with attached detonator has been in-"serted in the primer cartridge, securely "fasten it to the said cartridge by means of "string or other suitable material so that the "fuse and detonator cannot be inadvertently "withdrawn."

If my observations are correct, this practice is not universally followed, and in certain mines it is a dead letter. It is asserted that the tying up of a cartridge takes too long, and when a large number have to be made up, as in the case of the Premier Mine, there is, no doubt, something in this contention. All makers of explosives, however, insist that it is absolutely necessary to tie the fuse with detonator to the primer cartridge, and as a matter of fact the non-observance of this precaution has led to a great many accidents. The little device, the clip, which I am placing before you, see sketch, obviates the necessity for tying. It can be made out of tinned iron, brass, or ordinary malleable iron painted, so as to comply with the Explosives Regulations, but the main point about it is that it should be malleable. Generally speaking, it is better to avoid brass or copper on account of possible metallurgical complications with the bullion. When the fuse has been cut off to the proper length it is slipped through the star shaped hole (*a*), and the detonator is secured to it in the ordinary way by means of pliers. If one tries to pull the fuse the little projections (*a.a*) grip the fuse, thus preventing it being pulled back. A hole is made in the primer cartridge in the ordinary way, the detonator with clip is slipped into the hole, and the three arms (*b*) are folded over the top of the cartridge. The little prongs (*c*) shown in the elevation grip into the cartridge, and if the metal is sufficiently malleable the grip is a very fast one. So far tinned iron has been found quite a suitable material. For electric detonators the star shaped hole (*a*) in the centre becomes a square-shaped hole.



Primer Clip.
(*a*) Star shaped hole (stamped out) with
(*a.a*) Points projecting up.
(*b*) Arms.
(*c*) Prongs for gripping cartridge.

These clips can be made very cheaply. So far as I can see this operation should be very much quicker than the ordinary one of tying, because it is not necessary to open the end cartridge as is generally done. All that one has to do is to make a hole for the detonator through the wrapping paper, slip on the detonator and fuse with the clip, fold down the projecting arms, and the cartridge is ready for use.

Mr. B. C. Gullachsen (*Member*): As far as I can see from the Regulations, there is no reason why a native should not make up the charges. According to the Regulations, the native is allowed to make up the charges under the supervision of the white man; at least that is the way I read the Regulations.

Prof. G. H. Stanley (*Vice-President*): I should like to ask Mr. Cullen whether under the Mining Regulations one is allowed to bring raw or naked iron into contact with dynamite; if brass or copper is debarred for these fitments there is only iron left.

Mr. Wm. Cullen (*Past-President*): Paint the iron and it is all right.

MINING EXHIBITION.

Mr. Wm. Cullen (*Past-President*): I presume that all members of the Society have been kept *au fait* with the progress of the arrangements for our forthcoming exhibition. The exhibition will not remain open for quite such a long period this year as formerly, the actual inclusive dates being from the 19th to 29th May. Members will, no doubt, remember that one of the original objects of the exhibition was to bring to the notice of those interested, devices which have to do with the saving of life and general minimising of risk, reduction of costs, and the increase in the efficiency of mechanical contrivances connected with the mining industry generally. The kind of devices I mean might be exemplified by the Fuse Clip which I have just brought to your notice. Then again, in order to bring the exhibition to the notice of everyone we have arranged for small handbills to be posted up on the notice boards of every Recreation Hall along the Reef, but even this will not suffice, and the Committee earnestly hope that—as this particular section of the exhibition can only be made a success by individual effort—members resident on mines will make it their personal and special duty to bring to the notice of everyone interested—whether members or not—this branch of the exhibition.

Then, of course, we shall have the usual trades exhibits—local and overseas manufacturers—and we hope to make a special feature of ambulance appliances.

Most of you have read in the local press about the "Safety First Committee." Very likely we shall be working in conjunction with the Committee which has this matter in hand, and to carry matters further we hope to give a series of popular lectures during the exhibition period on "safety first" measures. Otherwise the exhibition will be much on the lines of previous years, but thanks to the good-will of the authorities we shall have much more space available for exhibits.

REFEREE ASSAYS.

Mr. A. Whitby (*Member of Council*): The town assayer is occasionally called upon to make referee assays for the mines, I say occasionally advisedly, because so few of these cases come into our hands that they are scarcely remunerative; but it frequently happens that we receive inadequate quantities. Let me take a few concrete instances. It is not exactly fair to give the referee only four assay tons of screens for assay. He has no means of checking in case anything goes wrong with his assays; nor is it fair to send less than 12 assay tons of slime residues or less than 40 assay tons of solution. There have been occasions when I have only had two assay tons of the screen sample or wine bottles of solution to make assays of. Well, this is ridiculous.

There is one other little point I have in mind in connection with this. We frequently get water samples from the mines, but they are sometimes sent in wine or whisky bottles with apologies for corks. Only very clean stoppered winchesters should be used.

Mr. Andrew F. Crosse (*Past President*): I agree with Mr. Whitby. It is quite absurd to expect anyone to make correct assays when one has not a sufficient quantity of sample. Only a fortnight ago I made four assays of a certain material, and they differed very much. I had to make another half-a-dozen before I could get a decent average.

DESCRIPTION OF AN ELECTRIC PARTING APPARATUS.

Mr. F. Wartenweiler (*Member of Council*): The apparatus described has been gradually evolved by Mr P. H. Monckton, Assayer of the Village Deep, Ltd., and is in daily use.

The illustration shows the various parts and tools, as follows:—1. Electric parting furnace. 2. Electric drying and annealing furnace. 3. Porcelain parting dish of seven compartments. 4. Seven dampers by means of which heat is controlled over contents of each acid compartment. 5. Handle for removing parting dishes after annealing. 6. Lever controlling conversion of drying furnace to annealing furnace. 7. Tray upon which dishes are taken to balance-room. 8. Aspirator containing 50% solution HNO_3. 9. Aspirator containing distilled water. 10. Transferrer upon which prills are placed and from which prills are dropped simultaneously into the acid. 11. Anvil and hammer. 12. Steel forceps. 13. Decanting basin. 14. Handle for opening or closing the dampers when hot.

Manipulation.—Seven prills are placed on the transferrer (10), then dropped simultaneously into the porcelain parting dish (3), which has been previously filled with acid and placed in the parting furnace (1). The heat is regulated by means of the seven dampers (4). From the parting furnace, after decanting simultaneously and washing the seven assays simultaneously, the dish is moved to the combination drying and annealing furnace (2), where, by manipulating the controlling lever (6), the drying and then the annealing is effected by first keeping the dish some distance from the heating element and subsequently lowering it to a position immediately above the element.

The following are the advantages over the (i) test-tube and (ii) cup methods :—*a.* More rapid. *b.* Less acid consumption as compared with (i) and (ii). *c.* More cleanly. *d.* More perfect control of heat in each acid. *e.* The acid in each depression is boiling when prills enter. *f.* Greater compactness ; there being no necessity to move more than 3 ft., whilst handling an almost unlimited number of prills. *g.* Less liability of interchange. *h.* Less handling and therefore less chance of accidents. *j.* The turning of a lever obviates the necessity of muffle work.

Breaking During Parting.—When silver does not exceed 10 to 1 of fine gold present, breaking does not take place. When prills do break there is no more danger of loss than in any other method of parting known to the writer. It has been found, after exclusive use of the dishes for a period of 8 years, that systematic composition checking, points to efficient parting and that, by experiment, prills parted in this way do not retain more than 0·2% of silver.

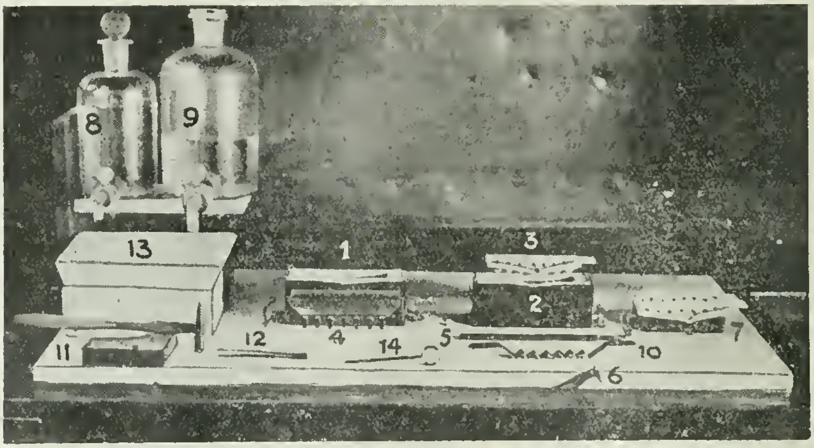
Capacity of Depression.—One cc. of 50% acid is used in each depression—this is sufficient to part prills containing 20 mgm. of silver and 5 mgm. of gold.

Rapidity.—28 prills can be cleaned, decanted, washed four times, dried and annealed in 10 min. as against (i) 28 in 30 min. and (ii) 28 in 25 min.

Heating Elements.—Two are needed, each consisting of about 7 ft. of resistance wire which is wound on sheet asbestos : they are run in series on the Village Deep where the current is 115 volts alternating. By increasing or decreasing length of wire any voltage whether alternating or direct can be used.

Life of Elements.—No deterioration is apparent ; after 100 hr. the wire becomes coated with a thin layer of oxide which appears to protect the metal from any further action of the heated air.

Cost of Current Consumed.—About ¼d. per hr.



Electric Parting Apparatus.

THE SAND-FILLING OF MINES.

(Read at September Meeting, 1913.)

By W. A. Caldecott (Past-President) and
O. P. Powell (Member).

DISCUSSION.

Mr. E. E. Hardach (*Member*) : Sand-filling was started on the Wit Deep over two years ago, principally for the purpose of relieving the pressure on partly worked out areas and of supporting the roof while reclaiming the pillars and other reef left in the old workings, and is being extended to follow up more closely to the current stope faces. It has materially lessened the number of air or pressure blasts in the mine, and enabled us to recover nearly all the pillars in the old areas so far worked over.

The surface plant, which, including the bore hole, cost £7,620, consists of a system of belt conveyors under the sand treatment vats (so arranged as not to interfere with the track haul-

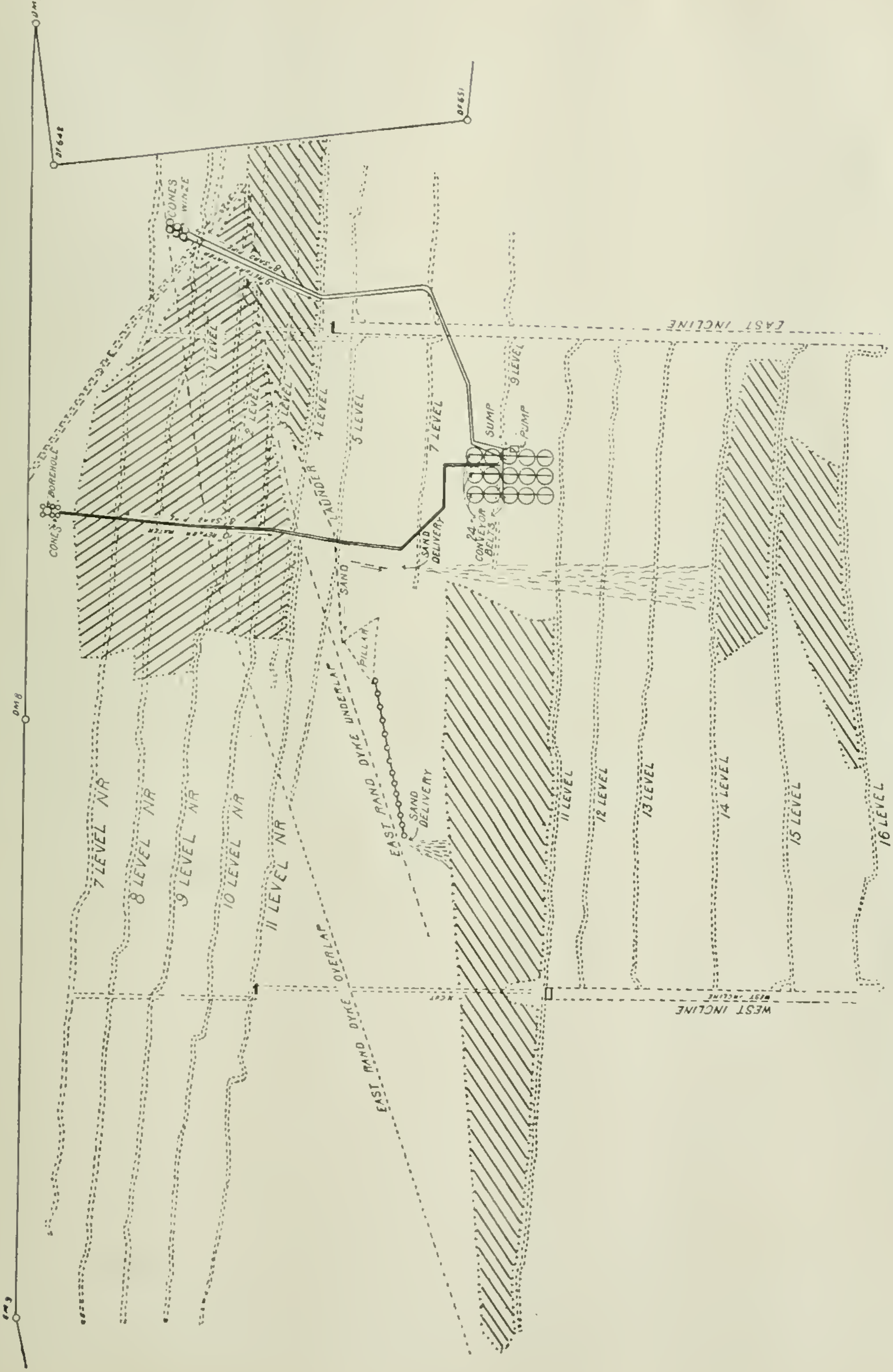


Fig. "A."—Showing the General Surface Arrangement of Sand-Filling Plant with the Relative Position of the Sand Packs Underground shown Hatched.

age to the dump) which delivers the sand into a sluice box, with a grade of 20%, to the suctions of two 6 in. Robeson sand pumps. One of these pumps delivers the sand to a steeply inclined winze for filling in the south section of the mine, and the other to a 6 in. bore hole 780 ft. deep for filling the north section of the mine. 8 in. screwed piping is used for deliveries, and 9 in. and 10 in. cheaper quality pipes for the return water. The pipes are laid out 1 ft. below the surface, and at a grade of about 1½% towards the pumps. The deliveries are 1,200 ft and 1,400 ft. long respectively, and they show no signs of wear, and have given no trouble from choking, except on one occasion when one of them had been several months out of use. Water is circulated for about half an hour after we stop pumping sand. There are 6 dewatering cones at the winze and 6 at the bore hole. To convey the sand through the deliveries, from 2 to 3 times its weight of water is used, and the underflow from the cones contains about 30% moisture. Each cone will handle about 10 tons of dry sand per hour.

Only current sand has been used; and to 31st January, 1914, 328,645 tons had been put in the mine at a surface cost of 3·82d., and an underground cost of 3·74d., making a total cost of 7·56d. per ton of sand filled. The surface costs include discharging the sand from the treatment vats, running and maintenance of conveyor belts, pumps, cones, and water lost at 7½d. per 1,000 gallons. The underground costs include all timber, matting, launders, and labour required for putting in the stulls to retain the sand and for supervising the filling.

The sand from the winze is conveyed underground in launders with a minimum grade of 15°, when it is necessary to deflect it a considerable distance laterally; when this is not required it is allowed to run on the footwall. Deflecting stulls made of 4 in. uprights, wedged in place and lagged and covered with a strip of jute matting, have been found cheap and serviceable for deflecting the sand for two or three weeks. The upper portion of the mine has a dip of 55°, and when the sand is run in launders at this grade they wear rapidly, but at 15° they do not show much sign of wear.

In the steep portions of the mine, and where it is desired to keep a level open permanently, the pillars above the level are thinned down to 5 ft. or 6 ft., and the box holes closed with stulls, lagging, cocoa-nut matting, and the sand filled in above. Where a level is not required to be kept open more than two years, or where the dip is less than 40°, we take out all pillars and run a line of stulls 15 ft. or 20 ft. above the level, lag, mat, and fill with sand. We have tried jute matting, but have not found it satisfactory as it does not allow the water to filter through freely, and is liable to impound water behind the stull, which, when sufficient has collected, will break through and perhaps cause a serious accident.

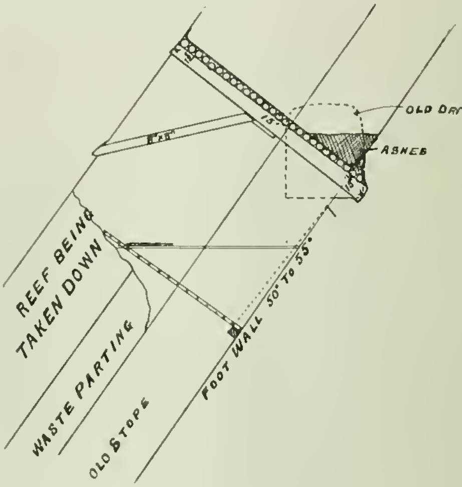


Fig. B.

We have no difficulty in retaining the sand in the steep portions of the mine as it seems to set very firmly, but we have always taken the precaution of putting in substantial stulls in the steep wide stopes where it was desired to work below the sand-filling. Fig. "B" shows one of our most difficult places to work. Here a large reef was stripped down from the hanging-wall of a steep stope, the pillars in the old stope being all badly crushed. We have had in some places to put in sand packs and then reclaim the reef above them, but generally we find it more economical to take out the reef down the old stopes and follow up with the sand packs from the sides; in a few cases we have had to put in sand packs above, as shown in Fig. "B."

Mr. J. H. Veasey (*Member*): It is to be regretted that this paper, so full of sound practical information, has not received greater attention. It was disappointing to find the problem of sand-filling the deeper mines—where some little time ago so-called "air blasts" were creating doubts and fears—somewhat neglected in the discussion.

I think the merits of the Cinderella system deserve more than the passing notice they received. We are here to practice the truest mining economy, and are not justified in using a method or installing a process only for the pleasure of seeing it work satisfactorily. So far as I can recollect the description of the Cinderella process there were three points in its favour, and one against.

First in favour.—It was found that the loose, dry sand from the edge of the dump had been so far neutralised by atmospheric influences as to require no further treatment for the destruction of cyanide.

Second.—The water used for transport of sand when drained was only pumped back to the foot of the vertical shaft, *not* to the surface, thus saving a 4,000 ft. lift of pumping.

Third.—The discovery that dry sand would fall in a vertical flume without choking, eliminated any necessity for a 4,000 ft. borehole, and saved *(a)* the difference in cost between the flume and a pipe, *(b)* the heavy wear of wet sand on a pipe, and, *(c)* the usual troubles with pipes through choking. Against the system it is said that it "can be operated only in fine weather"—a doubtful compliment. I have not heard anyone state definitely that the system *cannot* be operated in wet weather, but the originators do not claim otherwise, and on a mine there should be no difficulty in finding other employment for the sand filling gang on wet days.

In a country where we get a maximum of fine days—and farmers have not seen many wet ones in the last six months—it is to be doubted whether ability to operate in all weathers, as against ability to operate in fine weather only, would justify the difference in cost between a part wet, part dry, system, such as the Cinderella, and an all wet system necessitating preferably a bore hole, and an increased amount of pumping and arrangements for destroying the cyanide present in current residues.

As an underground man — and I hope a practical man—I am grateful to the authors for the store of information given in their paper.

THE EFFECT OF CHARCOAL IN GOLD BEARING CYANIDE SOLUTIONS WITH REFERENCE TO THE PRECIPITATION OF GOLD.

By Morris Green (Associate).

DISCUSSION.

Mr. Andrew F. Crosse *(Past-President)*: The author brings up a very interesting point, that is, the occlusion of gases in the charcoal. I think he is quite right in saying that the gases when they are condensed in this have a far more active chemical effect than in the ordinary free state. He also shows that he has extracted oxygen and carbon monoxide from charcoal. The question I want to ask is: If oxygen and carbon monoxide are present together in this active condition, should they not act on one another and form carbon dioxide?

Mr. F. J. Pooler *(Associate)*: This paper gives us some useful additional information to our knowledge of the power of carbon as an absorbent of gases. The author is evidently of the opinion that charcoal itself has no chemical action in the cases studied. This is, to some extent, in line with van 't Hoff's remark that* "the absorption of gases and dissolved bodies by animal charcoal, of dyes by fibres, and so on are rather to be treated as surface phenomena." The author states that "the supposition that carbon *per se* participates in the reaction is so obviously incorrect that it requires no further comment." Nevertheless, he gives us a fairly elaborate example in support of his statement, from some of his previous work with graphite, in which he says 'graphite' is regarded as 'carbon.' Now the structures of graphite and carbon are so different that the explanation does not seem quite conclusive. The author assumes that if it were possible for carbon to act chemically here, it could only do so in an electro-chemical way. This, however, does not take into account the possibility of a catalytic action of the "fine" carbon which is deposited in the charcoal cells. Since carbon monoxide "passed into an ordinary gold-bearing cyanide solution would not precipitate gold," why should it in charcoal—unless the charcoal itself has some action? Some text-books suggest the possibility of absorbed gases being liquefied in the pores, and that† "the 'condensed' gas is usually more chemically active than the ordinary gas." This seems like a contradiction since liquefied gases are, on the whole, less chemically active than the gases themselves. Thus one is thrown back on the idea of the carbon acting catalytically. The catalysis however does not necessarily mean that the action takes place with or without the aid of the carbon monoxide, and the author's work advances our knowledge at least one step in the clearing up of the mystery that attaches to occluded gases in general and *absorption* by charcoal in particular. In a claim to have proved the transmutation of elements made recently, stress has been laid on the power of glass, silica, etc in occluding gases and the possibility and probability of errors arising from this source; which all goes to show that our knowledge of these things is not too profound. Whence it behoves us, before accepting any conclusions arrived at on these lines, to make sure that all possible sources of error have been removed. So far as I can see, the author does not tell us what precautions he took to assure himself that none of the gases mentioned in Table V. came from the silica tube. This seems to be rather a serious omission. Before dismissing the question of the action of charcoal, I should like to ask if the precipitating power of charcoal has been tried on other gold or silver solutions than aurocyanide solutions? If so this might help in clearing up the point as to the feasibility, or otherwise, of a catalytic action of "fine" carbon, for it is quite possible that the fine carbon functions both in the absorption of gases phenomena and in this action. Potassium cyanide is so unstable that the structure of the deposited carbon may have some effect on the solution, so far unsuspected.

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* Lectures in Theoretical and Physical Chemistry, Pt. II., p. 77.
† Mellor, "Modern Inorganic Chemistry," p. 721.

In discussing the results of Table V., the author refers to the decomposition of CO into CO_2 and C at 450° C. Now, is the inference drawn necessarily correct, since under the conditions of his experiments in the silica tube, the concentration of C particles would be practically infinite? Would not this then tend to prevent the decomposition of CO into CO_2 and C? Again, it is known that below red heat carbon does not burn in very dry oxygen with formation of CO_2 but of CO. Since the author had a temperature of 500° C. and a high vacuum in his silica tube, it is probable that the carbon was sufficiently dry to cause some action of this kind. It is also not very clear why the action of oxygen upon carbon should not be very appreciable at 500°, since finely divided carbon ignites at quite low temperatures. It is therefore not apparent why the percentage of CO need be probably higher than that found in Table V.

When one considers the volumes of gases found in the various experiments there is apparently no connection between the different results obtained. This is curious and might repay further study. The large percentages of nitrogen found in Tables IV. and V. do not confirm the usual text-book statements that oxygen is more easily absorbed than nitrogen, the absorption coefficients for one volume of charcoal being given by Mellor for the gases in the tables as follows:—

CO_2	68.
CO	21
O_2	18.
N_2	15.
C_2H_4	75 (after Hunter).

The author's numbers do not show any relation apparently to these quantities, and it is probable either that there are great variations with temperature and conditions of experiment, or that some chemical action took place at 500° C. This point I will elaborate later. It is evident that the numbers given in Tables II., III. and IV. are of little value in this research as the carbon lost little, if any, of its precipitating power. In the table given for gases from graphite, one wonders why the results are so different, again, from the charcoal results. They seem to bear out my remark above, that the comparison of the actions of the two forms of carbon is not conclusive.

Referring again to the volumes of gases obtained, a curious relation appears. Taking 250 cc. as the volume of all gases in Table V., we have:—

$$CO_2 = 31 \text{ cc.}$$
$$O_2 = 1{\cdot}75 \text{ cc.}$$
$$CO = 65{\cdot}5 \text{ cc.}$$
$$\therefore \text{ Total oxygen} = 31 + 1{\cdot}75 + 32{\cdot}75 = 65{\cdot}5 \text{ cc.}$$

Now referring to Table III., with 82 cc. of gases, we have:

$$CO_2 = 0{\cdot}16 \text{ cc.}$$
$$O_2 = 20{\cdot}5 \text{ cc.}$$
$$CO = 0{\cdot}9 \text{ cc.}$$
$$\therefore \text{ Total oxygen} = 0{\cdot}16 + 20{\cdot}5 + 0{\cdot}45 = 21{\cdot}11 \text{ cc.}$$

If there had been 250 cc. here, then

$$\text{Total oxygen} = \frac{21{\cdot}11 \times 250}{82} = 64{\cdot}4 \text{ cc.}$$

This is sufficiently near to 65·5 cc. to make one wonder if the proportion of oxygen to total gas was fairly constant, but that during the rise in temperature most of the remaining oxygen combined with carbon.

I hope that these inconclusive desultory remarks will not, in any sense, be taken as a depreciation of the author's work. I have had the pleasure of seeing so much of it in active operation, that I am fully alive to its value, as a contribution to our knowledge of the points he has covered and its economic worth to the gold industry. But one feels that his conclusions are founded on insufficient experimental data. The final experiment with CO passed over sugar charcoal, which then became a precipitant, seems to support the author's conclusions, but it would have been more satisfactory had we been told how the CO was prepared and purified. We must know more of the properties of carbon as an absorber before final conclusions are arrived at.

It was pleasing to see that the author would have nothing to do with the suggestion of "osmotic pressure," which was apparently put forward as an example of loose thinking through a confusion of osmotic and gas pressures.

It should be noted that I have not cast doubts upon the fact that CO is the effective gas in the reactions, but that I have an idea that the condition of the carbon has a lot to do with the reaction. The results of Table VI seem to support this idea. It may also account for the wide divergence between the author's results and those of Hunter taken above from Dr. Mellor's recent text-book.

In conclusion, I think we should thank the author for his paper, so full of interest and value, and I should like to take this opportunity of congratulating the Society and the School of Mines on the possession of so essentially a "local product" with a desire to undertake original work and the training and mental equipment to carry it out.

A METHOD OF ASSAYING CONCENTRATES AND BATTERY CHIPS FOR GOLD AND PLATINUM METALS.

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

I believe that I am justified in stating that not very much has been published on the methods used for estimating the platinum metals in ores and concentrates, but a great deal of work has been carried out on their separation in alluvial platinum, the results of which have been published. I have carried out a few experiments lately and the results may be of interest to some of our members.

A great deal more work would have to be done before the very best methods could be adopted, but this class of work must be paid for by those who would derive benefit from it. As the value of the platinum metals has risen so much of late years, I believe in many cases it would pay to recover these metals, and there is no reason why it should not be done here. In this connection I would draw your attention to a very interesting paper on the "Detection of Platinum Metals in Cupellation Beads" in Bulletin No. 111, of the Institution of Mining and Metallurgy of Dec. 11th, 1913, by Messrs. C. O. Bannister and G. Patchin.

The method which I have evolved for determining the metals mentioned, namely, gold, platinum, and osmiridium, in the ores, concentrates, etc., is as follows :—

Having obtained buttons of lead, according to the ordinary pot assay method, of sufficient size to have absorbed all the precious metals in the portion melted, cupel at a fairly high temperature, and add 2 to 4 gm. of silver (standard silver can be used) to each lead button. I use this large quantity of silver, as by experiment I have found it advisable, when much iridium is present.

Having obtained a bead of silver containing the metals mentioned, proceed as follows :—

Place a porcelain crucible $1\frac{3}{4}$ in. in dia. over a Bunsen burner (or a spirit or petrol lamp), and add several lumps of fairly pure cyanide of potassium so that when melted the crucible is about half full Then add several small pieces of cadmium to the melted cyanide, the weight of the cadmium being about three times that of the silver bead. At a low red heat the melted cadmium is clearly visible through the cyanide of potassium, and when this is the case, the bead of silver is now dropped into the melted cadmium and quickly alloys with it. As it is very important that the silver, etc. is well mixed into a homogeneous alloy, give the crucible a shake round by taking it up with small tongs. Then pour the melt into a warm button mould and after cooling, dissolve off the cyanide in warm water and the bead of metal left is then digested in dilute nitric acid (1 : 3). The cadmium and most of the silver are dissolved and the other metals are left as a black powder. I catch the powder on a small best quality filter paper and wash well, then transfer the filter paper to a platinum dish and burn away the paper at a low temperature. I then put in about 10 gm. of pure acid sulphate of potassium and melt in the crucible at a low red heat. When cold the melt is dissolved in warm water and the clean precious metals are caught on a filter paper, and well washed. The filtrate may contain rhodium and palladium, but I have never found any in solution. The filter paper containing the gold, platinum, etc., is then burnt and the gold and platinum dissolved in cold aqua regia, giving plenty of time. The residue left is osmiridium. The filtrate containing gold and platinum is carefully washed into a small porcelain dish and evaporated to dryness on a water bath, this operation being repeated twice, adding a few drops of hydrochloric acid each time, and finally dissolving in water. The solution of gold and platinum is transferred to a very small conical shaped flask and sulphurous acid passed through slowly. The gold is precipitated as metal, and after standing for some time is collected on a small best quality filter paper and washed. It can be calcined and weighed, but it is better to wrap it up in a little lead foil and cupel it. It is perfectly absurd and unnecessary to take refined gold as this is, and alloy it with silver and part it over again. The filtrate contains the platinum, which can be easily precipitated by adding a few drops of hydrochloric acid and a little magnesium wire or ribbon. This precipitated platinum must be collected on a small ashless filter paper, washed well, calcined and weighed. The osmiridium left after dissolving the platinum and gold is collected on a small ashless filter, well washed, calcined and weighed.

I have described the process in detail, but I cannot claim that it is all original, though some of it is, and the combination is new.

Examples :—1. I obtained a small sample of "Battery Chips," which is a very nasty material to assay, weighed up 100 gm. and digested for two days in dilute hydrochloric acid, using fresh acid when required. The undissolved residue was mostly iron pyrite, and this was collected on a filter, washed, and then roasted with 50 gm. of quartz powder ; it was run down and treated as described. The residue after removing the platinum by aqua regia gave 10·6 oz. osmiridium per ton (2,000 lb.). Gold obtained as described 31·26 oz. per ton, and platinum 1·25 oz. per ton.

2. A small sample of "concentrated black sand" gave the following results per ton of 2,000 lb. : Gold, 36·6 oz. ; platinum 2·9oz. ; osmiridium, 26·2 oz.

I made several synthetical tests with gold and platinum and obtained satisfactory results, but more experiments should be made to ensure certainty.

Prof. G. H. Stanley (*Vice-President*) : I should like to say that there are two points which seem to me to be weak. It may be that the method of parting described by Mr. Crosse corrects them, but that I should like to prove. In my own experience, using the ordinary methods, I found that the ordinary fusion assay conducted as usual does not always collect all the iridium ; it may catch all the gold and platinum, but not the iridium. Some of it is left in the slag. I am doubtful whether it really alloys with lead, but it can be wetted with it in the same way as gold is wetted by mercury. If the slag is recrushed and re-fused, generally a further quantity of iridium is obtained in the second button and some possibly in a third. At any rate, after parting, some of the iridium is very finely powdered. I have often filtered with a close paper, and I have had cases where the filtrate has apparently been perfectly clear, but on standing and just tapping the containing beaker the tiny iridium particles can be observed beginning to coagulate at the bottom. In that way, it may be that some error is introduced. Apparently the method is quite satisfactory for gold and platinum, since Mr. Crosse says it has been checked ; but I should like to know whether any experiment has been made, trying a known amount of iridium finely divided in the first place.

Prof. J. A. Wilkinson (*Member of Council*): As there was no discussion on the two previous papers on this subject, a brief survey of the three methods may tend to stimulate some interest amongst those who are more directly concerned, the mine assayers, some of whom to my knowledge have done work on this subject.

METHOD OF ASSAYING BLACK SANDS, ETC., FOR PLATINUM GROUP METALS

Proposed by

J. Gray.		C. Toombs.		A. F. Crosse.	
Obtain a lead button and cupel with Ag Residue ↓	To get rid of Pb	Obtain a lead silver button, cupel, Residue ↓	To get rid of Pb	Obtain Pb button, cupel	To get rid of Pb
				Add Ag, obtain Ag button and add to melted Cd under fused NaCN giving	
Treat with hot conc. H_2SO_4	Ag	Treat with HNO_3Aq (1 : 2·4)	Ag	Alloy of Cd, Ag, Au, Pt, etc.,	Cd, Ag
Dissolve Au,Pt in aq.reg. ↓ $HAuCl_4 + H_2PtCl_6$	Insoluble Pt groupmetals	Weigh the residue		Treat with HNO_3Aq Residue Treat with $KHSO_4$	Rh, Pd
Make alkaline with NaOHAq Add H_2O_2 filter ↓	Au	Dissolve out Au with 10% aq.reg Residue Pt metals ↓	Au	Residue Au and Pt Dissolve in aq.reg	Os, Ir
H_2PtCl_6 Pass H_2S	Pt	Wash, decant, ignite, weigh		Solution $HAuCl_4$ and H_2PtCl_6 Pass SO_2	Au
				Treat filtrate with Mg and HCl	Pt

Mr. H. Rusden (*Associate*): With regard to the parting of the cadmium and silver button with nitric acid, has Mr. Crosse examined the filtrate for platinum, because I am under the impression that he will find platinum there ; a certain amount of the platinum would dissolve in the presence of a large amount of silver.

Mr. Andrew F. Crosse (*Past-President*): I think Mr. Rusden is wrong, because I think cadmium is one of those metals that precipitate platinum. As I said, these points require a lot more experimenting on, so I do not wish to appear dogmatic.

Mr. H. Meyer (*Member of Council*): May I ask what result would have been returned if this assay (for example with battery chips) had gone through without special care to separate the platinum and iridium. Would it have been still 31 oz. or more?

Mr. Andrew F. Crosse (*Past-President*): Do you mean, if there had been an ordinary gold assay done whether the platinum metals would have been weighed with the gold?

Mr. H. Meyer (*Member of Council*): What would have been the gold return if done by the ordinary method?

Mr. Andrew F. Crosse (*Past-President*): It could not have been assayed correctly in the ordinary way; some of the platinum metals would have been weighed with the gold, but platinum itself would partly be lost.

Mr. H. Meyer (*Member of Council*): You may be right about that; but I have never yet seen a special return of platinum or iridium from an ordinary assay.

Mr. Andrew F. Crosse (*Past-President*): Those may have been quite exceptional battery chips; I do not know where they came from. The reason I bring it up for discussion is because valuable metals are being lost.

Mr. H. Meyer (*Member of Council*): To put the thing very commercially, have buyers been buying platinum and osmiridium in the shape of gold?

Mr. Andrew F. Crosse (*Past-President*): If they have treated this material in the ordinary way by cyanide, all the osmiridium and platinum would remain undissolved and this valuable material would be lost.

Mr. H. Meyer (*Member of Council*): I just want to find out whether it is the seller or the buyer that loses?

Mr. W. Cullen (*Past-President*): The point raised by Mr. Meyer is quite an important one. Of course I do not know whether the amounts of platinum and osmiridium involved are large, but seeing that platinum is now in the neighbourhood of £9 10s. per ounce, and osmiridium somewhat higher, it appears to me that it would pay to look into the matter. What Mr. Meyer, no doubt, has in his mind is that platinum and osmiridium are really being sold as gold, and that the mines on the Witwatersrand are losing to this extent. Although what I have said has very little to do with Mr. Crosse's paper, I think that the question of the presence of these two rare metals requires investigation, and if the quantities are anything like those which I have been told about unofficially, it certainly would pay to go into the question of extraction and separation and ship them home to Europe as platinum and osmiridium. It may not be generally known that dealings in the platinum group are controlled by a very few persons and naturally it is their business to keep their processes as secret as they possibly can.

Prof. J. A. Wilkinson (*Member of Council*: Do I understand from the remarks Mr. Cullen has made that he advocates a policy of refining gold before it is sent to Europe?

Mr. W. Cullen (*Past-President*): Decidedly.

ELECTRIC BLASTING.

(Read at October Meeting, 1913.)

By Wm. Cullen, (Past-President), Thos. Donaldson (Member of Council), and W. Waters (Associate).

DISCUSSION.

Mr. J. Chilton (*Member*): The authors of this treatise deserve commendation for their timely and interesting paper, and though the subject is not new, the impetus given by it in the direction of industrial safety will tend to make every mining man their debtor. Electric blasting was first practised as far back as the year 1851, when Dumas was sinking through heavily watered strata at Veyras, Ardeche, France, and since that time the method has slowly spread over the mining world. After suffering from neglect for years the system is regaining its rightful position in the forefront of mining methods. In many of the American mines troubled with dangerous gases, it has been installed, and men are hoisted to the surface, so that if a premature or accidental explosion does occur at blasting time, all the men are safe, firing taking place from the bank. In the North of England, the practice is rapidly extending, though as yet the portable battery is mainly used and shots are fired singly. The English Commission on Accidents in Mines 1886 in reporting "consider that firing by electricity should receive the serious attention of mine managers." The new German Mining Regulations promulgated in 1912 are based on the belief that the system of firing by electricity is safer than that of ignition by ordinary safety fuse, and in certain mines of Dortmund district electric shot firing has been made compulsory. The primary object of the system is to insure the health and safety of the underground workers, and if by it dust and fumes and the other dangers of the mine can be avoided, surely the method merits a trial in the mines of the Transvaal.

The condition of some of our mines after blasting operations can only be described as deadly. The Miners' Phthisis Prevention Committee in their report tell us that in place of the 3 mgm. which should be considered the standard allowable, the air of some of our mines suddenly becomes charged with as much as 286 mgm. and sometimes even rises to 540 mgm. of dust to the cubic metre of air. It is little use looking for any diminution of our industrial scourge while miners are asked to remain in such an atmosphere even for a few minutes per day. Hauling men at downcast shafts only will tend to lessen the evil, but this cannot be done in many cases. Water blasts, sprays and sprinklers help to lay a large amount of dust, but they are powerless to eliminate entirely the dust in the air immediately after blasting. Until some device is invented that will entirely do away with the dust, the better way is to get the miners to the surface before firing starts. This aspect of our conditions has not been overlooked by our industrial captains, and in most well managed mines not more than 8% of the workers are underground at lighting up time. Yet this number is compelled to breath the dust-laden atmosphere, and as long as the present system remains, no substantial reduction of our industrial plague can be expected. Under the system described by the authors the necessity of exposure to dust and fumes is avoided, and though the method may not be applicable to every mine, yet its extension to single shift mines is desirable and reasonable and could be made without any great sacrifice of efficiency. The question of cost will have no place in the counsels of the industry if it can be shown that the electric firing method will sensibly reduce our phthisis bill.

Our present system of blasting must stand condemned by all thoughtful students of our mining conditions. Surely it is possible to find some method less costly in life and health than the present one. In looking over the painful pages of the Government Mining Engineer's Report for 1912 one is startled at the tribute that was paid in that year for our present method of breaking ground. Through the courtesy of the Mines Department the following data were supplied. In 1912 in the Union of South Africa no less than 21·3% of the fatal accidents in mines was due to our present method of preparing charges and using explosives.

Analysis according to causes. Union of South Africa:—

	Accidents.	Deaths.	Injuries.
Drilling into unexploded holes	137	130	193
During blasting operations	61	31	37
Due to fumes	24	36	2
Sundry and unknown ...	17	4	19
Whilst charging ...	12	7	12
Handling detonators ...	30	2	29
	281	210	292

On the Witwatersrand gold mines 10·6 of all accidents were caused by explosives, analysis showing the following:—

	Accidents.	Deaths.	Injuries.
While charging ...	12	7	12
Drilling into unexploded holes	126	120	176
During blasting operations	43	22	27
Due to fumes	23	35	2
Sundry and unknown ...	12	4	14
Handling detonators ...	23	...	23
	239	188	254

In a word, our present methods of dealing with explosives in charging and blasting operations cost the Witwatersrand gold mines no less than 188 lives and caused serious injury to 254 persons in the year 1912. While such a price in life and limb is being paid, he will be a bold man who will defend a system that means such a waste of life; and the point it is desired to make clear is this, that with electric blasting a marked reduction in these accidents would occur and some items would be entirely swept away.

The introduction of electric shot firing will undoubtedly lead to the introduction of specialism in underground work. The rockdriller of the near future will be a driller of holes, purely and simply, and need know little of blasting and explosives. The breaking of rock will be in the hands of a few carefully chosen miners on each mine, who will commence to load up the holes after the drilling shift is done. The rush and hurry of the present system will be abolished and accidents whilst charging up will be sensibly reduced.

It is contended by some mining men that the suggested system will increase the number of misfires, and therefore the accidents due to drilling into unexploded holes. The authors of the paper make no extravagant claims concerning this matter and admit 0·6% of misfires at the

Meyer and Charlton Gold Mines where electric blasting is in force; but that percentage is far below that of safety fuse. Careful observations carried over four months with the best brands of safety fuse give a percentage of 7·46 misfires. With greater practice in electric blasting the number of misfires should be reduced, as 0·6% is very high compared with the best European practice, for Mr. J. Belfitt in his paper on "Detonators," read before the Association of Colliery Managers, states that out of 43,161 shots fired in 2½ years he had only three misfires.[*] Perhaps the greatest critic of electric firing is C. Dobblestein, who writing in *Glückauf*, Jan. 1913, contends that electric firing is not safer than fuse firing, but admits that many of the electric firing accidents are due to faulty detonators, bad firing apparatus, and the ignorance of the average miner. Probably no system of ignition can be invented than will be absolutely fool-proof, but electric blasting seems to point the way to a reduction of blasting accidents due to drilling into misfired holes.

In 1912, 43 accidents and 22 deaths occurred on the Witwatersrand due to blasting operations. These were mainly due to persons entering the zone of blasting operations deliberately, or owing to entrances not being safeguarded; hiding away and sleeping in stopes; and holing through without proper warning. As all persons will be out of the mine before blasting starts, under the suggested method, this item would be entirely eliminated. Many of the deaths due to fumes are caused by the burning of explosives through faulty safety fuses, and by persons being compelled to pass through fumes on their way to the shaft after blasting. As these causes would cease to operate with electric ignition, deaths due to fumes would be reduced.

Accidents due to running fuses would become a thing of the past under the electric system. The Chief Inspector of Explosives in his report for 1910 concerning safety fuse says that "safety fuse does not run." Without questioning the good faith of this statement, one may doubt its accuracy. Accidents recorded as "premature explosions" convey to the mind of the mining man the idea of a running fuse. Lighting up accidents numbered 9 with 6 deaths and 4 injuries. With firing by electricity, both running fuse and lighting up accidents would be entirely swept away.

Electric blasting will no doubt have its own peculiar dangers, and the effect of exploding a large number of holes simultaneously would be disastrous in some of our older mines, but the introduction of delay action fuse would avoid this. The accidental closing of the electrical circuit is seemingly impossible with the safeguards of the Meyer and Charlton system, and looking at the system generally it seems likely that electricity will do for the safety of the mine what it has done for the safety of the sea.

From a rock-breaking point of view, electric firing would not make an appreciable difference except in the blasting of cuts where real simultaneity would be achieved. The question of cut and round firing can safely be left to the miner, who would soon adapt himself to the altered conditions. By a system of four hole cuts instead of the usual three he can increase his chances of successful blasting by 50%. In double drives and shafts double cuts could be used with delay action fuse for the side holes.

It is encouraging to note the interest that this paper has aroused in mining circles, and to note that safety and health problems are receiving their due share of attention even in times of industrial unrest and distraction.

THE NATURAL SODA DEPOSITS OF AFRICA; WITH SOME NOTES ON THE ALKALI TRADE.

(Read at November Meeting, 1913).

By John Watson, F.I.C. (Member of Council).

DISCUSSION.

Prof. J. A. Wilkinson *(Member of Council)*: After reading this paper through carefully I must confess to a feeling of disappointment caused chiefly by the fact that the expectations raised by its title were by no means realised. At the outset brief notes are given on four natural soda deposits in Africa and in only one case are the data for which a chemist chiefly looks, given. Concerning the German East African deposit I have not been able to get any data, but in the case of the lakes at Fezzan full details can be found in Lunge's book, Vol. 2, p. 61, 2nd edition. A pictorial description of Lake Magadi was given in *The State*, February 1911, p. 242, and the following analysis of native soda from Magadi was given by A. Gordon Salamon in his report to the Company at its formation:—

Na_2CO_3	43·55
$NaHCO_3$	40·41
NaCl	0·36
CaO and MgO	Faint traces

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[*] *Coal and Iron Trades Review*, Aug. 26, 1904.

SiO_2	0·07
Fe_2O_3 and Al_2O_3	0·04
NH_3	Nil.
SO_3	Trace
P_2O_5	Nil
Matter insoluble in boiling water	Trace
Water (by difference)... ...	15·57
	100·00

whence Total Na_2O in natural product = 40·38%
Do. calcined do. = 58·08%
Equivalent Na_2CO_3 do. do. = 99·3%

The main portion of the paper deals with the Zoutpan (which being translated means salt- and not soda-pan), north of Pretoria. This is an impure deposit of soda about 27 ft. thick, the datum concerning the thickness of the dark coloured impregnated clay above the main salt deposit, 7 ft 2 in, having been overlooked. The analyses given are incomplete and Hoffmann's analyses (also incomplete) given by Cohen, who first described the deposit, show that very little sulphate is present. As given by Clarke they read Cl 43·47%, SO_4 0·03%, CO_3 15·17%, Na 41·33%. Under the heading South African Alkali, Ltd., the author gives his own analyses, which, like the above, are purely technical and hence designed for a special purpose. The opportunity has been lost of rendering a distinct service to science by carrying out a series of exact and full analyses of the various layers and this is a fact to be lamented. In the analyses given under soda crystals, how is the fact explained that the local company's product is purer by about 3% than the pure substance!

The rest of the paper is beyond criticism and much of it is unworthy of one who, like the author, has had a considerable experience of alkali work. A large portion can be found in text-books, even those of an elementary character, and it is obvious that the author has not kept abreast of the later developments of alkali production. Concerning the possible uses of the company's product, its value in soap manufacture, which lies in its conversion to caustic soda, one of the most valuable commercial sodium compounds, is barely mentioned, and at present I see no valid reasons for its introduction into reduction works practice. As a water softener it has distinct value, but in this case purity is essential. Mr. Whitby's euphemism, that the paper bespeaks an earnest advocacy of a new industry, is true in many ways, but it should also be borne in mind that economic principles lie at the foundation of industrial success.

THE SOLUBILITY OF IODINE IN SODIUM IODIDE SOLUTION.

(Read at December Meeting, 1913.)

By Harold W. Gill, B.Sc.(Lond.), A.I.C. (Member).

DISCUSSION.

Dr. J. Moir *(Past-President)*: After what Dr. McCrae has written in criticism of this paper, there remains little for me to do but go over the same ground. The paper consists of very good experimental material — on which the author must have spent much time—yet absolutely spoilt and rendered useless by his careless presentation of the facts and apparent inability to think clearly. A man who for example deduces a chemical formula from the mere weights of the materials used, deserves severe criticism. His formula NaI_2 is merely ridiculous, for, by the same method of reasoning, water would be HO_8, and hydrochloric acid HCl_{35}! It is easy to show, moreover, that not even as regards *mere weights* of material do his tabulated results support the ratio 1 : 1 between NaI and I: his actual figures, if obtained in the manner given in the text, agree with the ratio 4 : 5; and this ratio when reduced to molecular ratios in the orthodox way, becomes nearly 2 : 3. Thus the author indicates that he took out 10 gm. of the solution of iodine in so called 20% NaI made at 25° C. and found in it 2·002 gm. of free iodine. This leaves 7·998 gm. for the water and NaI in the specimen taken, and the amount of NaI would be about 20% of this, namely, 1·6 gm. (since the contraction on dissolving iodine in NaI is probably quite small): the ratio of weights is therefore 4 : 5, and the ratio of molecular* weights is $\frac{1·6}{150} : \frac{2·002}{127}$ or 2 : 3 nearly; and the supposed chemical compound, if it existed, would be $2NaI + I_3$ or Na_2I_5. The solution in reality contains, of course, a mixture of NaI and NaI_3 (with a little NaI_5 at the higher concentrations), and the author might as well write a chemical formula for a plum-pudding! It may be noted that I get a different result for the "formula" from that deduced by Dr. McCrae from the same *data*. This is mainly due to the fact that it is impossible to say, from the paper as printed, which of two processes the author has really used, and that Dr. McCrae has followed the statement of the tables on p. 292, whereas I have believed the (contradictory) statements on p. 291. In regard to this self-contradiction of the paper, I think it will be quite impossible for

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* Making iodine monatomic for the moment.

Drs. Comey or Seidell to make any use of the author's figures. Thus, in Table I, we are told that 100 gm. of "5% sodium iodide solution" *dissolve* 4·99 gm. of iodine at 25° C., whereas on p. 291 it appears that what the author really did was to determine the amount of iodine *contained in* 100 gm. of the finished polyiodide solution, which is a totally different thing, because it leaves only about 95 gm. of sodium iodide solution to dissolve the 4·99 gm. of iodine. Now, when we add to this ambiguity the fact that the author never tells us whether his "5% NaI" is 5 gm. in 100 cc., or 5 gm. in 100 gm. finished solution, or 5 gm. plus 100 cc. water (all of which are different), we see that there are no less than six different interpretations of his experimental material! In addition, the author heads all his tables with the phrase "concentration of NaI," whereas the actual figures are not concentrations at all. "Concentration" is a technical term meaning *gram-molecules* per *litre* of solution, yet the author uses it of 5 gm. in 100 gm. (not cc.), and sometimes even of 5 gm. in 100 gm. of a solution which has changed in *volume* by having iodine added and in which consequently the exact content of NaI is, as far as I can see, unknown. This kind of thing is unpardonably careless: in fact the author seems to treat the grave and definite technical terms of chemistry in a very cavalier fashion.

The author will now probably think I have gone through his paper with a microscope: at any rate I have also succeeded in finding a flaw even in his experimental results, for in Table I. he gives the iodine dissolved (at 25° C.) by "20% NaI" as 20·02, whilst in Table IVb. the quantity dissolved at 25·75° C. is given as 19·69. This is impossible: it is obvious from the other results that a *larger* amount must dissolve at the higher temperature. It is no doubt an experimental error; but if so, this and the other results should not have been stated to two places of decimals if they are in doubt by three units in the *first* place.

The first paragraph of the author's "Summary" is really too wonderful for words. He says: "The amount of iodine dissolved (at all temperatures) by a solution of sodium iodide is directly proportional to concentration of the sodium iodide." Take the two extreme cases: (1) if the concentration of NaI is zero, then according to the author, water dissolves no iodine! Obviously, in any theoretical deductions from his experiments, the solubility of iodine in plain water should have been deducted from the total solubility before any attempt was made to assess the influence of NaI in dissolving more iodine: (2) taking the other extreme, it might appear that a solution of 50 gm. NaI in 50 gm. water would *contain* 50 gm. of dissolved iodine!! Seriously, the author ought to tell us what the above "half-and-half" solution can do in the way of dissolving iodine, and also tell us what volume is occupied by 100 cc. or 100 gm. of this "50% solution" after it has been saturated by iodine.

The *pièce de resistance*, however, is the bit about $NaI_3.2H_2O$ on p. 291 (repeated in the summary as if it were *the real thing*, or the blessed word Mesopotamia). This is really lovely: not only is the supposed compound as gratuitous as the formula N_4O for air, but it has water of crystallisation even in solution! I take it from the author's patronymic that he is a Celtic person like myself, which may explain the mystery: he may have used methods of divination or second-sight to solve a problem for which there is no chemical method available.

In conclusion, I wish to call your attention to the fact that this is evidently a difficult subject, since I find that Dr. Seidell's book (referred to by Prof. Wilkinson) is in error in regard to the cognate subject of iodine in *potassium* iodide: the third column of Seidell's table, giving *weights* of KI, is 10 times too large all through. This is a curious mistake, since the first two columns, giving the value in *molecular* ratios, are quite correct, and also since Seidell himself calls attention to a similar mistake on the same page.

It may be noted that if one takes the precaution of subtracting the solubility of iodine in plain water from the figures given by Seidell (*i.e.*, corrected), the ratio between iodine dissolved and KI present becomes nearly constant at all *concentrations* (not temperatures as the author says of sodium iodide) and is quite near to 1 : 1, so that if there were a chemical compound in these solutions it would be KI_2 (K_2I_4 by the artiad-perissad law).

This analogy, if the author noticed it, seems to have entrapped him rather badly: and perhaps he will not take it amiss if I tell him that in *theoretical* chemistry the race is not to the swift (*i.e.*, the conclusion-jumper), but to the suspicious and critical.

THE UNION PATENTS BILL.

(Read at December Meeting, 1913.)

A. L. SPOOR and W. E. JOHN (Members).

DISCUSSION.

Mr. H. A White *(Vice-President)*: The thanks of the Society are certainly due to the authors for the able account of the new Patents Bill. There may be some little trace of "special pleading" on behalf of the inventor and his agent so that it becomes necessary to emphasize the fact that there are two parties to the bargain between the public and the inventor. Seeing that the case for the inventor has been put so well before you. I propose to devote most of my remarks to the other side of the question.

I must first call attention to the curious fact that the authors find the present fees moderate, for the tax on invention they amount to may in many cases be a serious consideration. It has often being computed that the total royalties derived by the united inventors of the world will not pay the total fees demanded by governments; the basal reason for such a state of affairs probably being the essentially evolutionary nature of the progress of invention. The birth of an idea is rarely a sudden process, and its commercial utilization is most usually heralded by a succession of partial failures in which many take part. There is hardly a single modern process in actual use at this time which is not covered all over with patents "as thick as leaves on Vallombrosa."

In actual practice therefore a new patent is frequently a mere circumscribing of more or less floating ideas common to many for the benefit of one—if necessity is the mother of invention, adaptation is the father. It is thus possible in some cases that the granting of a patent may put a bar in the way of progress and real working improvements may be prescribed because their path lies through a jungle of patents whose width of claims may easily be stretched to embrace their scope. The fact that there is nothing new under the sun will be driven home to many an aspiring inventor after a brief interview with his agent, but it should easily be within the reach of all who are contemplating improvements to know what has already been claimed in the matter he proposes to deal with. A properly arranged and indexed abstract of all patent specifications is therefore most essential: the preparation of this should be undertaken at once.

Patent Office.—I see no reason why a sub-office should not be provided for Johannesburg, as the business to be expected there is probably greater than in any other town or even some provinces in the Union.

Commissioner of Patents.—I think it would be well in this age of specialization to confine the requirements to high legal attainments as the position is principally a judicial one. The combination suggested is perhaps not too rare, but patents deal much more with technology than high science. Section 13 provides for the appointment of as many examiners of patents as may be necessary and these of course should be chosen for their acquaintance with the various branches of applied science.

Definition of Invention.—One of the most futile of amusements is the multiplication of definitions of things that have no exact limits. The authors, for example, object to the phrase "known or used by others," but in their own suggested substitute for the definition of the Act (under *a*) the word *others* is used in precisely the same way. Their criticism as to the words "any foreign country" does not appear to be sound. How could a patent be defeated by its use among the head-hunters of Borneo unless a knowledge of the fact was obtainable in this country, and in that case the ban pronounced by the authors under the heading "applicant" must be applicable. The real fact is that civilization tends to form one country of all nations in matters of science and technics and in our own South Africa the dissemination of technical literature speedily makes common knowledge of all real advances made throughout the whole world. The mere introduction of devices in use in other parts of the globe cannot logically be called a patentable invention. I consider that the definition in the proposed Bill is sufficiently good though "Section 61" should be added to the sections mentioned therein.

Opposition to Patents.—In practice the "armoury of weapons" feared by our authors includes many which are useful only in exceptional cases. The most objected to by the authors (*e*) is undoubtedly by far the most important. "That the invention is not novel, or is not capable of being patented" is the base of most oppositions and it would be a serious blot upon the law if it were eliminated. A patent creates a monopoly and this dangerous but valuable right can only be given in exchange for some real and new advance. The average patent may be said to refer more to variation than advance and in very many cases it is interesting to contrast the magnitude of the claim made with the minimum of novelty shewn. No really new and useful invention can be affected by the paragraph referred to. Paragraph (*d*) should have the words "by the opponent" deleted, for other parties may be and frequently are interested directly or indirectly. With reference to paragraph (*j*) it may be observed that it is only reasonable that the title of the invention should indicate the subject matter sufficiently, for all whose business may be involved have to study those patents which may effect their interests. As the total number of patents is very large a falsely entitled patent may easily escape scrutiny and injustice might readily be caused.

Claims.—I think it would be well to fix some limit to the number of claims permitted for each patent. I have examined patents containing from 70 to 100 claims and in such cases the difference between one claim and the next is little more than a matter of words. Perhaps if the ordinary patent fee were reduced and an additional fee for every claim after the first six were charged a great deal of wasted time and paper might be saved. Section 36 (3) says that the specification must end with a distinct statement of the invention claimed, but I am of opinion that the confused jumble of repetition and minute variation so frequently met with requires a great stretch of courtesy to receive such an appellation.

Patent Agent.—I am in sympathy with the authors in this matter. The necessity for permitting solicitors to act as patent agents is of course more obvious in the smaller towns than in the larger centres.

Compulsory Working.—I cannot agree with the authors that a validly patentable invention is something the public had no right to use and the ideas contained in the paragraph referring to the matter appear to me to be based upon a one-sided view of the facts of the case. An inventor must always owe much to his predecessors; he is entitled to a large share but not to all the advantages of his invention. If he claims *all* the profits derived why should anyone use his invention at all? If as is so frequently the case he is merely the formulator and expropriator of current ideas to which he barely adds a finishing touch, why should that entitle him to block the stream of progress? More than half the patents applied for are for improvements and many of these are such as would suggest themselves to many using the matters referred to. Such patents are frequently applied for only to protect their users against rival claims and in this way a vicious circle is created the escape from which is provided for by this principle of compulsory working. It may readily be imagined that a fair proportion of oversea patents applied for in this country, referring to manufactures, are for the purpose of preventing competition here with the foreign factory and it is possible that the establishment of effective local industries might be seriously hindered if this remedy were not provided, and it is considered that its abuse is adequately guarded against by the conditions imposed.

Minor Alterations.—The second paragraph of Section 60 (3) is a definition of foreign country and should be placed with the other definitions in Chapter 1. Section 16 should be altered so as to leave the way open for sub-offices in every province. Section 23 requires amendment to make it clear that the register of patents includes the specifications, which are unaccountably omitted. Section 46 (2) should be deleted and (1) amended by the addition of the words "together with the provisional specification if any." If provisional protection has been given the public in return must have the information given in the specification in the same way and for the same reason as they demand the publication of the completed application. The 12 months secrecy is allowable, but after that time if the application is not proceeded with the abandoned idea must be assumed to be public property.

In conclusion I must endorse the view of the authors that the Bill on the whole seems to be a good one.

The meeting then terminated.

Review.

"Pocket Book for Miners and Metallurgists," by F. Danvers Power. 3rd Edition, 1914. Small 8vo. (Crosby Lockwood & Co., 371 pp. 6s.)

This is a useful compilation of notes, rules, formulæ, tables, etc., of value more particularly to the professions mentioned.

There is such a tremendous amount of data required in these professions, as in others, that the problem resolves itself into making a useful selection while still keeping the volume to note-book size. Naturally certain fundamental matter is the same for all professions and therefore figures herein as usual in such works. In addition there are sections on chemistry and assaying, mineralogy and geology, ore-dressing, toxicology, photography, etc. It is obviously very difficult to decide what to admit and what to reject in such a book and somewhat doubtful whether extreme condensation is always advisable

Much of the general information is necessarily very condensed but may serve a useful purpose in giving a lead in a brief consideration of a subject in the field, fuller consideration being reserved for the office, where presumably more extensive literature will be available. But in the section on ore-dressing it would probably have been more useful to confine the scope to certain definite data, rather than to attempt a survey of the subject as a whole.

Still the book will undoubtedly prove useful to engineers in the field, for it contains a wealth of valuable information in small compass and readily accessible.

It may be noted that in this 3rd edition the rock classifications have been altered, the section on slags extended, and ore-dressing brought up to date; while additional matter deals with slide rules, mine sampling and knots and splices. (G. H. S.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ELECTRICAL METHOD OF PREPARING AQUEOUS COLLOIDAL SOLUTIONS OF METALS.—"Colloidal solutions of gold, silver, platinum, palladium, copper, lead, iron, zinc, tin, nickel, aluminium, magnesium, bismuth, antimony, and cadmium, respectively, have been prepared in a few seconds by the electrical disintegration method, using a high frequency alternating arc, the leads to which were taken from two points on the inductance of the oscillatory circuit of a Poulson arc as used in wireless telegraphy. By varying the conditions it was possible to obtain currents of from 0 14 to 15 ampères and an E.M.F. of 480–4080 volts, and colloidal solutions showing a wide range of colours were thus obtained from several metals, especially gold, silver, and copper (see also following abstract)."—H. MORRIS-AIREY and S. H. LONG (*Proc. Univ. Durham Phil. Soc.*, 1912-1913, *5*, 68-70). *Journal of the Society of Chemical Industry*, Nov 15, 1913, p. 1,015. (J. W.)

COLOUR CHANGES IN COLLOIDAL GOLD.—"Electrical migration and other tests were made on red, blue and purple colloidal gold solutions prepared by electrical disintegration by means of a high frequency alternating arc (see preceding abstract). The results show that the red colloid moves to the cathode and is probably associated with hydroxyl ions: the particles are very small. The blue colloid moves to the anode and is probably associated with hydrogen ions. The red colloid is less stable than the blue, into which it is converted by the action of an electrolyte or of an electric field. The purple colloid is a mixture of the red and blue forms in variable proportions."—S. H. LONG, (*Proc. Univ. Durham, Phil. Soc*, 1912–1913, *5*, 113-118), *Journal of the Society of Chemical Industry*, November 15, 1913, p. 1,015. (J. W.)

CARE IN LABORATORY WORK.—"'If cleanliness is next to godliness, soap must be considered as a means of grace,' said Henry Ward Beecher, and certainly cleanliness is of great importance in a chemical laboratory. Yet many analytical chemists and assayers are slipshod and slovenly in their work, the accuracy of which is thereby impaired. Drops, for example, cling to the sides of dirty burettes: substances are contaminated with dust before being weighed; assay fluxes are more or less exposed to salting; grinding machinery is not thoroughly cleaned; solutions are injuriously affected by fumes; beakers are wiped with filthy rags; samples and balances are not cared for as they should be; bottles are not properly washed before use; precipitates are only partly washed; blank determinations are not made; in fact some laboratories are about as clean and orderly as a pig sty, and much time in the aggregate is lost in searching for various articles. 'My brethren, these things ought not so to be.'

As, at best, great care and attention to details are required for even fairly accurate results in many determinations, surely the more obvious precautions should not be neglected. If drops stick to the sides of burettes and other measuring vessels, the grease and dirt causing this should be removed with some cleaning fluid, a strong solution of commercial chromic acid in water being excellent for the purpose. This solution can be left overnight in contact with the dirty surfaces and may be repeatedly used. Some advocate putting vaseline on the taps of burettes, but I find that the vaseline may work up into the burettes and cause drops of solution to adhere All glassware should be thoroughly washed with tap water as soon as possible after use and allowed to drain before being put away. Flasks can be set upside down in holes in a board, while a grating of thin wooden slats on edge answers well for draining beakers. A burette should always be washed out immediately before use with a little of the solution with which it is to be filled, and time should be given for bubbles of air to escape before beginning to titrate. When the solution is run out quickly it is well to also wait a few moments before reading a burette to allow any liquid on the sides to join the main body. Burettes ought to be covered to keep out dust and solutions should not be allowed to stand in them for days, or even for hours. Errors are also caused by poor supports in which the burettes are lopsided and often not twice at the same angle with the perpendicular.

Where there is danger of contamination from the ordinary water, beakers and other vessels should be washed out (before using) with distilled water; in fact the safe plan is to take it for granted that everything is dirty or salted unless positively known to be otherwise. If distilled water and other reagents are kept in bottles exposed to dust, with necks unprotected by caps, some dust, unless previously removed, will surely enter the portions poured out. Reagents ought to be in cupboards; but dark curtains in front of the shelves are better than no protection at all from the dust and light. Solutions particularly sensitive to light should be kept in dark bottles, and ordinary bottles, painted outside with two coats of black varnish, serve admirably. I find, for example, that a solution of potassium permanganate remains unchanged for months in a bottle thus treated.

A convenient way of protecting assay fluxes from possible salting is to keep them in deep granite-ware oblong pans, side by side, in a long box provided with a hinged cover, which box can be on a table or shelf of suitable height, with space in front for crucibles. Old, used fireclay crucibles are excellent to run through crushers and grind on bucking-boards, after pulverizing rich ore, and the ground crucibles form a sort of by-product and should be saved for mixing with fireclay in patching up or making assay furnaces. Pulverized crucibles, put in dry, make good furnace bottoms, easily removed and renewed in the event of slag running out of cracked crucibles.

If the balances can be kept in a room free from fumes and dust, so much the better; but, if not, card-board boxes are far more satisfactory covers than are the close-fitting, cloth affairs sometimes seen, such boxes being serviceable, tight and little trouble to make when adhesive tape is used for joining the edges.

Some appear to think that reagents marked 'C. P.' are absolutely chemically pure, which is practically never the case. To remove the last traces of impurities may be exceedingly difficult, as in getting rid of one substance traces of others are often introduced. Hence the importance of blank tests in particular work. I find, for instance, from 1 to nearly 1½ mgm. of nitrogen in the ordinary quantities of C. P. reagents, and distilled water employed in the Gunning method for nitrogen, using a little sugar in the blank to take the place of the organic matter present, in a regular analysis. Occasionally 'things are not what they seem.' Obviously, in nitrogen, and in a certain stage of potash determinations, the absorption of ammonia fumes should be rigidly guarded against, such fumes being very common in laboratories.

Errors, which no subsequent care can remove, are also due to analyzing portions not truly representative of the samples, and the incomplete washing and purifying of precipitates are frequent sources of inaccuracy. What with impurities in the chemicals and the many possibilities of influencing the results, it is small wonder that a complete analysis rarely adds up to an even 100%. In fact, should this happen, losses and gains may have accidently exactly counterbalanced, or some constituents have been determined by difference, or the analysis is open to suspicion.

The analyst must therefore be content with doing his best and, instead of despising 'the day of small things,' must take all reasonable precautions if he hopes for accuracy." — Gordon Surr, *Mining and Engineering World*, Nov. 1, 1913, p. 750. (H. A. W.)

Temperature of Decomposition of Dolomite. —"Dolomite begins to decompose at about 500° C. into calcium and magnesium carbonates. This is also the approximate temperature at which magnesite begins to evolve carbon dioxide (this *J.*, 1913, 361). The decomposition of dolomite reaches a maximum between 710° C. and 730° C., when the magnesium carbonate formed is immediately decomposed. The calcium carbonate begins to decompose at 875° C., and does so most rapidly at 900—915° C."—O. Kallauner (*Chem. Zeit.*, 1913, *37*, 1,317), *Journal of the Society of Chemical Industry*, November 29, 1913, p. 1,069. (J. W.)

Determination of Radium in Radio-active Substances.—"A small fragment (0·01—0·0001 gm.) of the mineral or salt is placed in a hollow in a thin rod of arc carbon clamped between terminals connected with a source of electric current. The rod is disposed inside a closer chamber provided with inlet and outlet tubes. It is heated by passage of a current of about 10 ampères per sq. mm. of cross-section to 2000°—3000° C. for 10—30 seconds and the emanation is transferred, by way of tubes containing calcium chloride, soda-lime, phosphorus pentoxide and cotton-wool, to an electroscope which has been exhausted, re-filled, observed, again exhausted, and then placed in communication with the heating chamber. Concordant results are obtained. In the case of solutions, an accurately measured small quantity of the relatively concentrated solution is placed on the carbon rod, and the determination made after one month. Experiments with pitchblende showed that the emanation was evolved only very slowly up to about 750° C., after which the amount liberated increased rapidly with rise of temperature."—A. L. Fletcher (*Phil. Mag.*, 1913, *26*, 674-677), *Journal of the Society of Chemical Industry*, October 31, 1913, p. 992. (J. W.)

The Dissolution of Gold and Silver in Cyanide Solutions.—"According to I. I. Andreev in a recent number of *Zeitschrift fur Elektrochemie*, the solubility velocity, AG, for gold divided by the percentage of KCN concentration is fairly constant at very low KCN concentration and reaches a maximum at 0·25% KCN concentration. In 1 to 50% KCN solutions the solubility velocity is proportional to the concentration of the O_2 present. Weakly ozonized O_2 has an accelerating effect on the solubility velocity for Au. If the O_3 content exceeds 4%, only a temporary acceleration occurs; the surface of Au becomes brick red, due to oxide formation, and a definite retardation in the solubility velocity follows. H_2O_2 has an identical effect, very slight amounts accelerate, increasing quantity retarding. Some visible oxide formation with accompanying retardation takes place in air under certain conditions even in KCN solutions of 0·01 to 0·005% concentration. This phase is being further investigated: PbO_2 and MnO_2 individually and alone have no effect on the solubility velocity of Au in KCN, but in the presence of H_2O_2 or O_3, O_2 is given off and marked acceleration of the solubility velocity takes place. No Au oxide layer was observed. Analogous experiments were carried out with silver. Increasing amounts of H_2O_2 or O_2 give accelerating effects in much greater degree than with gold and the solubility velocity diminishes only with comparatively high O content. Combination of O_3 and H_2O_2 increases the solubility velocity 25-fold."—I. I. Andreev, December 6, 1913, p. 889. (H. A. W.)

Solvent for Precious Metals. — "A new method of generating halogen cyanides in connection with an oxidizing agent has been patented by Dr. Hans Foersterling, of the Roessler and Hasslacher Company, of Perth Amboy, N.J. The inventor melts sodium bromide and sodium cyanide in the proportion of one molecule of each, and allows the mixture to solidify. No decomposition takes place in this process. In order to convert this into bromocyanide, it is treated with an acid solution of hydrogen peroxide, and the reaction proceeds as follows:

$$NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O.$$

If an acid was not used, sodium hydrate would form, and destroy the bromocyanide. The rapidity of formation of bromocyanide according to the above equation depends on the concentration of the solution. The formation of the corresponding chlorine and iodine cyanides according to the same formula, is much more rapid than the formation of bromocyanide." H. Foersterling, *Metallurgical and Chemical Engineering*, December, 1913, p. 715. (J. A. W.)

The Estimation of Traces of Chlorides in Water.—"In estimating traces of chlorides in natural water by titration with $\frac{N}{100}$ $AgNO_3$,Aq. $\frac{N}{100}$ $AgNO_3$Aq, using K_2CrO_4 as an indicator, it is necessary to take certain precautions.

A known volume of water (150 to 200 cc.) must be boiled until all the calcium bicarbonate is decomposed, then it is allowed to cool, made up to the original volume, left to settle, decanted, and 100 cc. of the clear liquid titrated with the silver nitrate.

The end point is determined by colorimetric test against the end point made with 100 cc. of a standard solution containing 5 mg. of NaCl per litre. H_2S in water is destroyed by boiling with a slight excess of HNO_3, which is finally neutralised by the addition of $CaCO_3$. Ammonia can be similarly neutralised."—Maurice Lombard (*Bull. Soc. Chim.*, 1913 [iv.] *13*, p. 1,006-1,011), *Journal Chemical Society Abstracts*, December, 1913, ii., p. 1,068. (J. A. W.)

METALLURGY.

THE ELECTROLYSIS OF CYANIDE SOLUTIONS.—"The experiments upon which this article is based were conducted by E. H. Koenig and S. E. Woodworth in the Metallurgical Laboratory of the School of Mines of Columbia University. The work was undertaken with the idea of determining the validity of some of the conflicting statements which were found in reviewing the literature on the electrolysis of cyanide solutions

The electrolysis of cyanide solutions which are used for the treatment of gold and silver has caused much speculation, not only on account of the application of electrolysis as a means of recovering the gold and silver from leaching solutions, but also as a means of regeneration of the cyanide. Two of the main items of expense of the modern cyanide process are the consumption of cyanide and the cost of recovering the gold and silver in a fairly pure state.

Electrometallurgists and electrochemists, having in mind the reduction in cost of treatment of gold and silver ores, have developed a number of processes in which electrolysis has been applied for the recovery of metals from leaching solutions and for the regeneration of the cyanide, which during the treat ment has been converted into cyanide compounds whose solutions do not dissolve gold and silver.

Conclusions from the Literature on the Subject.—The conclusions, which may be drawn by making a digest of the literature on the electrolysis of cyanide solutions, are as follows: 1. Electrolysis has, under favourable conditions, been utilised as a means of precipitating and recovering gold and silver from cyanide leach solutions by electrolysing the solution with suitable electrodes and with a current density of 0·01 to 0·6 ampere per square foot (0·11 to 6·5 amp. per square meter). The metal was deposited as a dense film when the current density was properly regulated with regard to the composition, the concentration, and the circulation of the solution. The deposit formed is pulverulent and non-adherent if the solution is dilute, the circulation sluggish, and the current density high. 2. Iron anodes dissolve in cyanide electrolytes, causing consumption of cyanide and contamination of the solution by precipitates of iron compounds 3. Carbon and graphite anodes are satisfactory in cyanide electrolytes when used at low current density, but they disintegrate at high current density, causing fouling of the solution. 4 Peroxidised lead anodes are not dissolved during electrolysis of cyanide solutions, and the consumption of cyanide by their use is less than when metallic iron anodes are employed. 5. Fused iron oxide anodes are permanent in cyanide electrolytes, and the cyanide consumption by their use is small. 6. Oxygen, supplied to cyanide leaching solutions either by additions of chemicals or by electrolysis, is beneficial for the treatment of certain refractory ores, in that the extraction is thereby accomplished more rapidly and more efficiently. Electrolysis is claimed to be a cheaper and more efficient method of supplying oxygen to cyanide leaching solutions than by the addition of chemical oxidizers. 7. Regeneration of cyanide works solutions, which contain cyanogen compounds, is possible by electrolysis. There is controversy as to whether alternating current can be used for this purpose. 8. Electrolysis of solutions containing sulphocyanide and cyanimide by means of direct current at high current density (50 amperes per square foot, 540 amperes per square meter) produced solutions of cyanide. Fused iron oxide anodes were found to be the most satisfactory for this purpose. 9. Electrolysis of solutions containing only cyanide produces cyanates, which will not dissolve gold and silver. 10. Telluride gold ores were effectively treated by use of solutions which contained an halogen-cyanide compound, which was generated by electrolysis of solutions containing an halide salt and cyanogen compounds, employing direct current at high current density for producing the active halogen-cyanide compound.

Experiments.—The object in view, when the experiments were undertaken, was to determine: (1) which kind of anodes are the most permanent, and which kind give the highest efficiency as regards the regeneration of cyanide solutions: (2) whether the electrolysis of cyanide leaching solutions is beneficial in increasing the extraction of gold and silver from ores, and in reducing the cyanide consumption; (3) whether high or low current density produces the greatest amount of active solvents in cyanide leaching solutions, the active solvents being those which are active in dissolving gold and silver; (4) whether a relative difference in current density at the anode and the cathode affects the consumption of cyanide, and what effect this relative difference in current density has upon the production of active solvents in cyanide solutions; (5) what effect the alkalinity of cyanide leaching solutions has upon the results of electrolysis; (6) whether electrolysis of cyanide leaching solutions is beneficial in reducing the time required for the efficient treatment of gold and silver ores; and (7) what current density at the anode and at the cathode is most suitable for the regeneration of cyanide leaching solutions which contain ferrocyanide and sulphocyanide.

Conclusions.—A summary of the results of the experiments on the electrolysis of cyanide solutions is as follows:

1. The electrolysis of cyanide solutions by direct current and by means of insoluble anodes is accompanied by progressive consumption of cyanide, which is the result of oxidation of the cyanide to oxycyanide compounds.

2 Anodes of metallic iron, nickel, and lead are dissolved during the electrolysis of cyanide solutions; these metals on entering the solution are immediately precipitated, the lead as hydroxide and cyanogen compounds. The consumption of cyanide by means of these anodes was relatively greater the lower the current density, as at high current density oxygen was simultaneously liberated, and consequently less metal was dissolved. The consumption of cyanide was less in case of the lead anodes than with either the iron or the nickel.

3. Peroxidised lead and 'passive' iron anodes were found to be more permanent than metallic anodes of either iron, nickel or lead. There was no apparent corrosion of either the peroxidised lead or the passive iron anodes so long as they kept continuously in use, but exposure to the atmosphere for several days destroyed their permanency, due to spontaneous oxidation. The passive iron anodes were found to be more satisfactory than peroxidized lead anodes for electrolysing cyanide solutions in that the cyanide consumption was much less and the voltage was slightly less in most cases.

4. Electrolysis of cyanide solutions which were used for leaching a special refractory gold and silver ore containing sulphide minerals was found not to be beneficial either in reducing the cyanide consumption or increasing the percentage of extraction.

5. Low current density at both the anode and the cathode gave less consumption of cyanide, and in several cases seemed to have produced a slight

amount of active solvents, but not enough to compensate for the loss of cyanide which resulted by the electrolysis.

6 The cyanide consumption during electrolysis of cyanide solutions was greater when the current density at the cathode was run higher than at the anode, which suggested that the loss of cyanide is the result of oxidation, with the formation of oxycyanogen compounds. The lower the current density at the anode and the higher it is at the cathode, the relatively greater is the cyanide consumption.

7. When cyanide solutions containing sulphocyanides or ferrocyanides are electrolysed, the cyanide consumption was much less than that which occurred in pure cyanide solutions, which indicates that sulphocyanide and ferrocyanide act as protective agents during the electrolysis of cyanide solutions.

8. The increased alkalinity of cyanide solutions reduced the cyanide consumption during electrolysis, due to increasing the conductivity of the solution. The higher the voltage the greater the cyanide consumption, and *vice versa*.

9. The electrolysis of cyanide leaching solutions had no apparent effect upon reducing the time required for treating a refractory ore nor in increasing the percentage of extraction.

10. Peroxidized lead anodes and passive iron anodes are corroded in cyanide electrolytes which contain sulphocyanide, if the electrolysis is conducted at high current density. No apparent corrosion occurred when a low current density (below 3 amperes per square foot, or 32 amperes per square metre) was used.

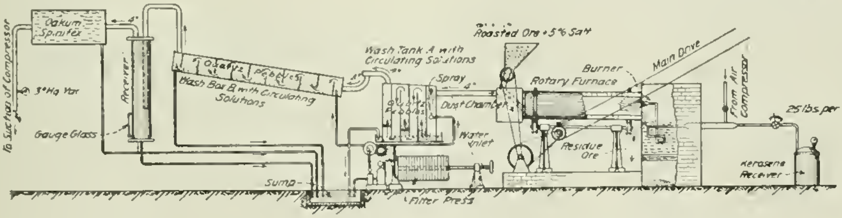

Diagram Showing the Arrangement of Apparatus for the Howe Process.

11. Passive iron anodes are more satisfactory than peroxidised lead anodes for the electrolysis of cyanide solutions which contain either sulphocyanides or ferrocyanides, in that by their use the cyanide consumption is very much less and the voltage slightly less. The lower the current density the smaller the consumption.

12. The regeneration of cyanide solutions which contain sulphocyanide and ferrocyanide does not occur by electrolysis with direct current, whether the conditions of electrolysis be made oxidizing or reducing by varying the relative current densities at the anode and the cathode.

The suggestions which were stimulated by the foregoing experiments will form the basis of a series of future experiments, which it is hoped will contribute something more to the knowledge of what occurs during the electrolysis of cyanide solutions."—E. F. KERN, *Mining and Scientific Press*, Oct. 11, 1913, p. 577. (H.A.W.)

THE HOWE VOLATILIZATION PROCESS.*—"The main features of this process were described in this *Journal* of March 29, 1913, and further results on a large scale by Ben Howe, at the Gwalia Consolidated mine, Western Australia, should be of interest. This property is one of the big mines in regard to ore deposits in this State, and the lode has been traced for about 3,900 ft, and over 100 ft. width averages $9.60 per ton. Hitherto the ore, containing antimony, even in the upper levels, has resisted the best known metallurgical methods to give a reasonable extraction. The mine has produced about $1,200,000 from above the 100 ft. level, and diamond-drilling to 500 ft. has shown $12.48 ore. In consequence of the success achieved with the experimental furnace, a plant capable of treating 30 tons of ore per day was installed at the mine, and early in July the first clean-up from this plant had seemed to show the success of the process on a commercial basis. It was then proposed to erect a 100 ton unit at once, consisting of a No. 8 Krupp ball-mill, rotary roasting furnace, condensing chambers, and gold recovery plant.

Well known metallurgists at Kalgoorlie have concluded that there are immense possibilities ahead of the process for the treatment of refractory, especially antimonial and arsenical ores, and those that will not yield a satisfactory extraction to cyaniding after roasting. In the case of the Gwalia Consolidated, the best results from roasting, sliming, and cyaniding were 75%, while from the volatilization process, where the ore is roasted with salt, the recovery is 92%. Another important point is the fact that in the new process there is considerably less expenditure on equipment.

Details of the Process.—The details of the process as at present running are quite simple. The ore is crushed dry, in a ball-mill, with 30 mesh screens, 5% of salt being added to the broken ore as it feeds into the mill. The mixture of crushed ore and salt is fed direct into a rotary brick-lined furnace, 27 ft. long × 5 ft. dia., making 2 r.p.m. The raw ore under these conditions roasts in the first 9 ft. of the furnace, and volatilization takes place in the next 18 ft. When the ore gets to the end of the furnace it has lost its gold and goes direct to the tailing dump.

The temperature required in the furnace is somewhat higher than that usually attained in a roasting furnace, and is obtained by gas firing from a wood generator. This generator is worked by pressure, not suction. Air at about 6 to 8 in. water-gauge is blown into the generator from a small Roots blower, and the same blower also supplies hot air for mixing with the gas before burning. The arrangement is simple, and well under control of the man in charge of the furnace. Large furnaces require less attention than smaller ones, and Mr. Howe believes that in the future a furnace about 9 ft. outside dia. × 90 ft. long, treating 100 tons per day, will be a good size. Such furnaces, built of boiler plate, rivetted together, would not be very expensive. Fuel consumption at present is about 16% of the ore treated, but this will be considerably reduced with a longer furnace. The regulation of temperature is an important item in the process.

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* See also this *Journal*, Vol. XIII., May, 1913, pp. 569-570; also Vol. XIV., July, 1913, pp. 14-15.

Course of Gases Through the Plant.—The gases are hot as they come from the furnace, and carry a good deal of dust, about 5% of the total ore. These gases pass through a dust-chamber, and then into a wooden box, 5 × 5 × 18 ft. The first 6 ft. of this box is brick lined to withstand the heat, and the whole of the top, 18 ft. long, is covered by a perforated tray which is kept full of water. This water runs through the perforations into the box as a constant shower. It is picked up again at the bottom of the box by a centrifugal pump and returned to the upper tray, thus keeping it in constant circulation. It should be mentioned that the perforated tray is divided into two parts, one for the dust portion of the box and the larger one for the gold-collecting portion. Each portion has its own separate pump. The 'dust' pump takes the water from the bottom of the box, as already mentioned, and passes it over two spitzkasten and a wooden table, like an amalgamating table, the former to remove the dust and the latter to cool the water somewhat before returning it to the top perforated tray. The 'gold' pump simply circulates the water from bottom to top continuously, but a small proportion, say 15% passes through a gravity filter, similar to those used in clarifying cyanide solutions. From these filters the gold is removed as a black sludge, similar to that from zinc-boxes, and is melted in a similar manner.

The shower of water in the gold-collecting area falls on cocoanut matting filters of special design, and it is through these wet filters that the fumes are made to pass. At present it is passed through one and then another of these coir filters, and so recover about 85% of the gold. During the next few weeks a third and perhaps even a fourth filter will be tried. As a final filtration the gases pass through a chamber filled with oakum and spinifex, the latter being a a tough native grass.

In the case of copper ores, more salt, even up to 10%, is required, but to compensate for this increased cost it is found that 80% to 90% of the copper content is also recovered from the ore by merely passing the gold-collecting solutions over scrap iron.

Mr. Howe states that the total cost, from the mine shaft to recovery of bullion, will not exceed $1.80 per ton. At a recent meeting of the Company in London it is reported that the results of this process have not been up to expectations, although a number of Australian metallurgists regard the process with favour and consider it an important step toward the successful treatment of refractory ores."—B. Howe, *Mining and Scientific Press*, Oct. 1913, p. 535. (H. A. W.)

MINING.

The Safe Handling of Missed Holes.—"The great cause of mining accidents is falls of ground, and it will probably always so remain. The next most important source, as abundantly proved by figures for all countries, is explosions of powder. There is this important difference between these two hazards, however, namely, that the first is extremely hard to guard against, the governing conditions being infinitely variable, whereas the second can be attacked with much greater success and a standard code of rules prescribed. Among explosive accidents it is probable that those due to missed holes predominate, and the detection and removal of these is a subject of the greatest importance. A missed hole once found, the question of how to render it harmless arises. In general, three lines of procedure are open, namely, to remove the powder by picking it out, to explode it with a new primer, to drill a new hole near it, but at a safe distance and in a safe direction, and explode the old hole with the new one. The object of our enquiry was to ascertain the opinion of those competent to have one, as to which of these methods was preferable. In general, it may be stated that the best practice favours shooting the old charge with a new primer, although exceptions to this are to be found. L. F. S. Holland states that the Smuggler-Union Company uses the new primer and does not permit the powder to be picked out. Robert E. Hanley, of the Mammoth Copper Mining Co., favours the use of the new primer in most cases. This is also the practice at the Calumet & Arizona. Mr. Gorhing, Superintendent of Mines, writes that in his opinion the only method of treating a missed hole is to insert a new primer, and that he knows of no circumstances under which it would be safe to drill a hole near by, or to clean out the old powder. Frank Juliff, foreman of the same Company, concurs and adds that he would advise that no drilling in the vicinity be done in drifts, raises or stopes while any powder remains in the holes. M. W. Mitchell, the other Calumet & Arizona foreman, agrees in general, but would permit the cleaning out of the powder in exceptional cases. W. H. Staver and E. B. Miller, of the Liberty Bell, unite in recommending shooting the old charge. In the event of this method failing, Mr. Staver favours cleaning out the powder or possibly drilling a new hole, if this can be safely done. Mr. Miller remarks upon the fact the collar of the missed hole may be shot off, leaving no room for a new primer. In such case, he would plaster the new primer over the hole and shoot it. He would never dig out the powder but would permit drilling a new hole when necessary. The rules of the Hecla Mining Co. governing the subject of misfires are quoted by James F. M'Carthy, manager, as follows:—'Miners who have blasted, on coming off shift, must notify the shift coming on if there have been any missed holes and how many, so that they may have full knowledge of the matter. In such case also the miners coming off shift must write plainly on the blackboard in the shaft house the places where they have left any missed holes and the number of such missed holes. Miners going on shift must examine said blackboard for the record of any missed holes before going to work. Every employee, after spitting a fuse, must remain within hearing distance and count the number of shots he hears and carefully note the number of missed holes, if any. No employee shall under any circumstances approach a missed hole within 20 min. after spitting the fuse. Employees must carefully examine the workings before setting up machines, for loose or scaling rock or any element of danger, and all missed holes must be fired before drilling is commenced. Picking giant powder out of a hole is strictly forbidden. No tamping of powder with steel or iron bars is allowed.' It will be noted that the rules prohibit the cleaning of the holes. A copy of these Hecla rules is given every man on the day he begins to work, and a copy is kept posted in the shaft house. Percy E. Barbour, manager of the Unwarra Mining Co., writes as follows:—'I am not in favour of drilling a hole adjacent to a hole which is full of unexploded powder. In our present work we invariably shoot the hole over, first carefully drawing out the tamping and old primer and inserting a new one. Our primers are made by inserting a fuse and fulminate cap in the end of a stick of powder, the top of which is tied around a fuse. Therefore after the tamping is withdrawn this primer can be pulled out by a gentle tug on the fuse. The old fuse is cut off and the primer and old cap put back in the hole and a new primer made and inserted. If, as sometimes happens, the primer explodes and leaves unexploded powder in the hole we handle it in one or two ways. If there is little powder in the hole and shooting will not seem to break enough rock to pay for smoke delay we take out the powder, but we first fill the hole with water to prevent any chance of explosion. If the contents seem to promise sufficient results we put in a new primer and shoot the powder that is left. As to the detection of missed holes: in our drift work it is easy to tell by the looks of the face whether the hole has failed to go or not. If the hole has been shot, but leaves a boot-leg, these are all tried with a tamping stick to discover if there is any unexploded powder left in the hole.' To pull out the old primer is equivalent in danger to cleaning the hole. Presumably, this primer is on the top of the hole. Mr. Bilharz, general manager of the Doe Run Lead Co., follows the same plan and recommends that the hole be cleaned and that no new hole be drilled. It is evident that neither extraction of the unexploded charge, nor drilling a new hole, is considered safe by most operators, although Mr. Barbour and Mr. Bilharz advise extracting the primer or the powder, and several others would permit it when necessary. This would seem to represent about the sentiment existing in the industry, and agrees with our own view that leaving the old charge undisturbed and exploding it with a new primer is by far the safest course to pursue."—*Engineering and Mining Journal*, Nov. 29, 1913, p. 1030. (A. R.)

MISTAKES MADE IN MINE MANAGEMENT.—"The following table is a carefully figured statement of the efficiency of air drills, averaged through many shifts, with many drill-runners. The drill-runners, and the blacksmiths who sharpened the steel, disputed the figures at first, but when they tried to disprove them they found that the figures were correct.

What the Drill-Runner Does for 480 min. of an 8-hr. Shift.

Work.	Time. min.	Cost.	Per cent. of time.
Setting up machines ...	82	$0.82	17
Shifting machines ...	66	.66	14
Changing steel	75	.75	15
Morning and noon starting	24	.24	5
Oiling and cleaning ..	15	.15	3
Getting steel and water ...	15	.15	3
Tearing down and blasting	45	.45	9
Actual time drilling ...	158	1.58	34
Total, 8 hr., or ...	480	$4.80	100

The foregoing figures are based on a wage scale of $4.80 a day or 1 ct. a minute, assuming that 'blowing' the smoke can be done between shifts.

This table of figures may seem unimportant at first reading, but it is interlocked with so many other items of expense that the results are really amazing.

The blacksmith shop being located on the surface is closely observed by the bosses, and generally the shop is crowded full of steel to be sharpened, which rushes the blacksmith to his limit. To make a showing he sharpens all he can, and gains a reputation for being a fast blacksmith, and the 'super' thinks he has a 'jewel of a blacksmith,' because he turns out a big bunch of steel every shift. A little error in 'gauging' the steel so that the longer bits will not 'follow,' would not be noticed by the 'super' on top—but the drill-runner notices it and may easily lose an extra hour of his time struggling to get the steel to follow. If there are 20 to 30 drill-runners losing an hour a day each 'changing steel,' and struggling with it on account of poor gauging by the blacksmith, it might be realised that a blacksmith might do better work if he wasn't crowded quite so hard, and that it would pay to give him a helper in the shop. Again, the temper and the general shape of the steel and its condition all have a bearing on the efficiency of the drill. If the drill-runner has to change steel oftener than necessary, it is taking time from the cutting machines, and reducing the efficiency of the drill. Do not blame the drill-runner for bad steel. It is not always the number of steel bits sharpened that makes the best blacksmith. I can compare (without naming either) two mines in which the conditions are almost identical; one of which has no less than 3 tons of steel in the rack, and the blacksmith is required to sharpen between 400 and 600 pieces of steel a day; while the other using the same make of drills, same air pressure and in the same rock, uses less than 33% as much steel. Both mines use machine sharpeners. The one using the less steel gets a round of holes drilled in just about one-half the time that the other requires to get in its rounds.

It makes a difference in progress and expense whether the drill-runner is cranking his machine while it is cutting, or whether he is 'fighting' with poor steel. If the drill-runners are not getting results, may be the pressure in the blacksmith shop is too high. If the miners are waiting for steel, and robbing one another of steel, or using battered or dull steel, the drills are not cutting rock, then the blacksmith force had better be increased. Twenty drill-runners losing an hour a day each from their work on account of poor steel, or lack of steel, figures 50 ct. an hour most any place, and that's $10 a day, $300 a month, or $3,600 a year lost.

Figuring lost time of men on pay, how many times will men go to the station for a drink during a shift, when in reality what they want is fresh air, and don't know it. How many times will men leave their work and go to the end of an air line or back into a well-ventilated passageway and stand and fill their lungs, generally thinking it is cooler where the air is fresh, when in reality, the fresh air, carrying less CO_2 and H_2O (humidity) may be but little cooler. It's air they want. Ventilation will save much time.

The average cost of good mining candles is 1¼ ct. each; it would take 16 candles to give the same light that one electric light globe would give. (I wonder how the superintendent would like to do his work with a candle on the desk.) The candles are more expensive, by the hour, than electric lights, considering the number burning at a time, the number carried in the hip pocket until broken, the number laid up on timbers and forgotten to roll behind or fall off and be broken, the number left when blasting, and the pieces of candles thrown away by the men. Thirty per cent. of the miner's time is lost 'fooling with bad lights.'

Having ascertained that the efficiency of the drill is but 34% of the possible or total efficiency, I wish to call attention to the fact that the average machine drill used in the mines of the United States, if rated as an engine, would be 1¼ to 1½ h.p. Not to discuss the difference between piston and hammer drills, we must acknowledge that a portion of this 1½ h.p. is used up in mechanical friction and the friction of long steel in deep holes—greater in piston drills than in hammer drills—but on an average we can say that from 33% to 50% of this 1½ h.p. is lost in friction and 'muddling,' which quite safely might be figured as leaving '1 h.p. of work.'

To get this 1 h.p. of work, we have wasted a great amount of energy. Taking the engine horse-power running the compressor, or the meter reading, and we find that from 25 to 35 h.p. on the surface is required to deliver 1 h.p. of work for 34% of the working shift, with the odds laid that there are more drills requiring over 35 h.p. than under."—LETSON BALLIET, *Mining and Engineering Journal*, Dec. 6, 1913, p. 1023. (A. McA. J.)

NOTES ON THE EFFECT OF TEMPERATURE IN MINES IN GREAT BRITAIN.—"One of the problems to be solved in deep mining is regulation of temperature, to enable the miner to work under healthy conditions and maintain efficiency. The temperature increases with depth; the rate of increase varies. Table 1 shows the geothermic gradient in different parts of the globe:—

Table 1. Geothermic Gradient at Different Localties.

Place.	Geothermic gradient, in feet per degree Fah.	Place.	Geothermic gradient, in feet per degree Fah.
Produits, No. 28	49·2	Sneyd ...	68·6
Rosebridge ...	54·5	Searle ...	69·0
Sainte Henriette ...	55·9	Ashton Moss	72·8
Abram ...	58·7	Simplon ...	72·0
Dolcoath ...	60	Wheeling ..	74·0
Paruschowitz	62·1	Harris's Navigation ...	76·0
Agrappe, No. 3	63·9	Port Jackson (N.S Wales)	80
Pendleton ...	66	Mont Cenis Tunnel ...	80·4
Rams ...	66·5	St. Gothard Tunnel ...	84·7
Agrappe, No. 10	66·8	Marcinelle, No. 1 ...	88·6
Sperenberg ...	67	Marcinelle, No. 2 ...	92·9
Schladebach	67·1	Dowlais ...	94·5
Kingsword ...	68	Tamarack ...	100
Agecroft ...	68·4	Hamstead ...	110
Dukinfield ...	68·4	Transvaal ...	220
Florence, Staffordshire ...	68·4		

Further variations occur, depending on the conductivity of different rocks, proximity to volcanic zones, the presence of thermal springs, and the percolation of water. It has been pointed out by Hoefer that heat is emitted from certain coal-seams as the result of decomposition and metamorphism. Variations in temperature are also met with, due chiefly to direct oxidation.

The temperature which affected the working capacity of the miner is shown in another table from which it will be seen that the average atmospheric temperature-gradient of mines in Great Britain is approximately 100 ft. per degree Fahrenheit. The effect of ventilating currents on mines would be to cool them down gradually, were it merely a case of heat absorption. The actual atmosphere is influenced by the ventilating current; but oxidation is also an important factor. Whether the rock is cooled by conduction and evaporation, or warmed by oxidation, depends on the volume of air circulating, and on the capacity of the mine walls for oxidation. The effect produced will depend upon the rate of oxidation as compared with the rate of ventilation. The loss in oxygen between intake air and return corresponds to the production of a certain amount of heat; the heat evolved per cubic centimetre of oxygen absorbed is between 3 and 3·8 calories, and if this figure is more than sufficient to account for the change in temperature and moisture between intake and return, the mine is increasing in temperature, and heating up beyond the temperature of the surrounding rocks. If the heat equivalent is less than indicated by the change, the mine is being cooled by the air-current. Both effects are seen in most coal-mines; the intake and part of the workings are cooled by the air-current, whilst the workings and return are warmed up.

The factors, which influence rise or fall in temperature of mine air depend on (1) temperature: (2) rate of oxidation; (3) increase due to compression of the air in the shafts (1° F. per 180 ft.); and (4) cooling effect due to taking up moisture (every grain added to a cubic foot of air equals a cooling effect of 80° F.)

It is now fairly established that susceptibility to temperature is influenced by the temperature of the wet-bulb, and not that shown by the dry-bulb, nor by the quantity of moisture in the air, nor the humidity. Capacity for work is seriously impaired when the wet-bulb exceeds 82° F., although this figure may be exceeded if a strong breeze blows, but 85° F. wet-bulb, may be taken as the limit. Inconvenience is experienced at 72° F. wet-bulb. To obtain an efficient return for labour, the temperature must be below 82° wet-bulb, a current of air should blow over the workers, and little clothing should be worn.

By adopting these precautions in British coal-mines, there would be no difficulty in working at depths of 5,000 or 6,000 feet; and it will be possible to mine at greater depths by means of artifical methods of drying and cooling the air, and by taking precautions against the heating effects of oxidation."—PROF. JOHN CADMAN, D.Sc., *Transactions of Institution of Mining Engineers.* (E. J. M.)

MISCELLANEOUS.

GRIPPING POWER OF PORTLAND CEMENT.—"The setting of Portland cement is assumed to be due to the formation of a gel, which ultimately stiffens to a lime-aluminate-silicate mass and forms a close-fitting network around any embedded inert material. With concrete, such a network surrounding angular gravel or sand would be stronger than one enclosing rounded grains, and the strength attained with washed sand would be greater than with unwashed, since, in the latter case, the intervening layer of soft clay prevents the surrounding network from directly gripping the grains of sand. Similarly with ferroconcrete, rust is detrimental as preventing close contact between the iron and surrounding gel."—H. LUFTSCHITZ (*Tonind.-Zeit.*, 1913, *37*, 1,491), *Journal of the Society of Chemical Industry*, October 31, 1913, p. 978. (J. W.)

ACID PROOF LINING FOR TANKS "can be prepared from the following formula: 10% litharge, 20% short-fibre asbestos and 70% sand, mixed to a plaster with a 40° Bé. solution of silicate of soda. The plaster can be applied as thick as desired. An application of acid after drying will 'set' the cement and render it acid-proof."—*Metallurgical and Chemical Engineering*, Dec. 1913, p. 705. (A. McA. J.)

IGNITION OF WOOD.—"The substance the most employed to prevent the ignition of wood is silicate of potassium in solution; the wood is soaked in it, which protects it for a long time from all danger of inflammability. But several successive layers are necessary in order to arrive at a really efficacious immunisation. The following is a good recipe:—35% of silicate of potassium, 35% of sulphate of baryta, from 1% to 2% of zinc white, and lastly 28% of water; but it is not the only recipe. The greater number, however, are trade secrets. M. Wolff considers of great importance the impregnation of the wood with alum in order to render it incombustible; this process, however, does not appear to be in use."—*Chemical News*, Sept. 26, 1913, p. 159. (C. T.)

ANTISEPTICS AND DISINFECTANTS.—"A general discussion of the problems of infection and disinfection. It is concluded that in the process of disinfection one or more of three types of activity may be engaged: (1) ionisation and diffusion, (2) adsorption, (3) chemical action. The last type of activity is preceded by the first and second when the disinfectant is an electrolyte and by the second when the disinfectant is a colloid. The importance of adsorption in disinfectant is emphasised, especially in regard to the advantages possessed in many cases by emulsions. Colloidal metals are powerful disinfectants. In the case of phenolic substances the germicidal power frequently increases with the boiling point. For example a fraction from blast-furnace phenols boiling at 248° C. had a Rideal-Walker coefficient 3 points above that of a fraction boiling at 220° C., and five points above that of a third fraction boiling at 207° C. In practical disinfection it is necessary to vary the nature and concentration of the disinfectants according to the species of microbe to be destroyed and other conditions and no method of standardisation can give information concerning the strength of disinfectant, time of application, etc., necessary for a given case in actual practice. The use of the Rideal-Walker test, which compares the strength of a disinfectant with that of a pure phenol solution required to kill *B. typhosus* in a definite time under standardised conditions, has, however, led to a marked improvement in the germicidal value of higher grade phenolic disinfectant emulsions. Full details of the method of carrying out the test are given and it is stated that variations in results obtained by two workers have always proved to be due not to some defect of the method but to neglect of one or more of the prescribed conditions."—D. SOMMERVILLE (*J. Roy. Soc. Arts*, 1913, *61*. 913–919, 927–937, 945–952), *Journal of the Society of Chemical Industry*, November 15, 1913, p. 1,027. (J. W.)

Reviews and New Books.

MINERAL DEPOSITS. By WALDEMAR LINDGREN. Cloth, octavo, 902 pages, illustrated. New York; McGraw-Hill Book Co. Price 25s. net.

The author is well-known as one of the foremost authorities on economic geology; he has long been a distinguished officer of the United States Geological Survey, and he is now professor of economic geology in the Massachusetts Institute of Technology, at Boston. By using the title 'Mineral Deposits,' the author widens his scope, and extends it beyond the merely economic limitations indicated by 'ore' deposits. The intimate knowledge of American mining regions that was obtained by Mr. Lindgren while on the Geological Survey has been supplemented by professional journeys to Mexico, Canada, Australia, and Europe. The geographical scope of his examples of ore deposition is wide therefore, and to it he has added a wealth of references, rendering his foot-notes an invaluable bibliography of the literature on the subject. In the early chapters he outlines the conditions governing the flow of water underground, elucidating the origin, chemical composition, and work of such mineralizing solutions. Then four chapters are devoted to the structural conditions that determine the shape and texture, the continuity and discontinuity, and the relative persistence of ore-bodies. Then comes the classification. The divisions include (1) Deposits formed by mechanical processes. (2) Those produced by chemical processes of concentration in bodies of surface water. (3) Those formed by evaporation. (4) Those resulting from rock decay. (5) And those concentrated by circulating waters. Next come those that originate from the activity of thermal waters and intrusive rocks, culminating in magmatic differentiation. Metamorphism and oxidation, as modifying causes, are separately discussed. Deposits formed 'near the surface' are distinguished from those formed 'at intermediate depth.'"—T. A. RICKARD, *The Mining Magazine*, Jan. 1914. (A. R.)

CHEMICAL GERMAN. By FRANCIS C. PHILLIPS. The Chemical Publishing Co., Easton, Pa., 241 pages (14×22 cm.). Price $2.

"Most courses in chemistry and chemical engineering require German for admission and include it in their curricula. Nevertheless, graduates of such courses are always unable to read German chemical journals intelligently. This is merely another instance of the futile character of the instruction usually given in foreign languages." . . .

"All in all, the book is a good one. The student who masters it will be able to read the German journals without much difficulty. Without access to the German literature it is impossible for a chemist to inform himself thoroughly upon any topic."—*Journal of the Franklin Institute*, Dec., 1913. (J. W.)

INDUSTRIAL CHEMISTRY: A MANUAL FOR THE STUDENT AND MANUFACTURER. Edited by ALLEN ROGERS (in charge of Industrial Chemistry, Pratt Institute, Brooklyn, N.Y.) and ALFRED B. AUBERT (formerly Professor of Chemistry, University of Maine). Pp. xiv + 854, with 340 illustrations. London: Constable & Co., Ltd., 1912. Net 24s.

". . . The sections on sulphuric acid, nitric acid, chlorine and allied products, fertilizers, the petroleum industry and lubricating oil, are specially interesting as affording a good deal of insight into American works practice. The chapters on these subjects are well-written and well-illustrated, and are particularly valuable from the fact that they supply details of working costs, etc., which are not always given in manuals of this kind. . . . The entire range of subjects has been well handled, and as we have already indicated, it will be found extremely useful to the English technical chemist as a reliable guide to the best American practice." *Chemical Engineering and The Works Chemist*, Nov. 1913, pp. 326-327. (J. W.)

A NEW ERA IN CHEMISTRY. Some of the more Important Developments in General Chemistry during the Last Quarter of a Century. By HARRY C. JONES, Professor of Physical Chemistry in Johns Hopkins University. New York: D. Van Nostrand Co., pp. xii + 326. Price $2.

"After an introductory chapter on the condition of chemistry at the dawn of the 'New Era' in 1887, the author proceeds to outline the development of the law of mass action, the hypotheses of Van 't Hoff, Le Bel and Guye, the phase rule of Gibbs, the views of Van 't Hoff and Le Chatelier on chemical equilibrium, and the modern theory of solution, including the electrolytic dissociation theory of Arrhenius." "The recent experimental work of Thomson, Rutherford and Mme Curie is briefly described, and the bearing of this work upon chemical phenomena is discussed. The chapters devoted to the work of Ostwald and his pupils in the old Leipzig laboratory, in what is known as 'Die gute Zeit,' are most interesting."—*Journal of the Franklin Institute*, Dec., 1913. (J. W.)

CYANIDE PRACTICE, 1910-1913. Edited by M. W. von Bernewitz. Published by the *Mining and Scientific Press*, San Francisco and the *Mining Magazine*, London. 732 pp.

"This is the third of a series of books that began with 'Recent Cyanide Practice,' edited by Mr. T. A. Rickard and printed in 1907, and which was continued by 'More Recent Cyanide Practice,' edited by Mr. H. Foster Bain and printed in 1910. They include articles on all phases of current cyanide practice based on experience in all parts of the world. In the main the papers here reprinted were first published in the *Mining and Scientific Press*. The foregoing description is given by the editor in the preface to the work.

Summaries of the progress in treatment of gold and silver ores during 1910-11-12 are given by Mr. Alfred James, yet a single article giving a critical summary of the miscellaneous subject matter in this work would have enhanced its value. As it is, some of the information given appears to be contradictory. For instance, dealing with the chemistry of cyaniding, certain well-known authorities precipitate copper, gold and silver cyanides with sulphuric acid. On the other hand, we have the Gitsham process described, which seems to indicate that solution of gold takes place with an acid solution.

The intelligent metallurgist will find the present work informative and stimulative. The subjects dealt with include the chemistry of cyaniding special problems, crushing, concentration and treatment of concentrate, roasting, agitation, decantation, filtration, precipitation, disposal of residues, estimation of tonnages, recent cyanide practice, recent notable mills."—*Australian Mining Standard*, Dec. 24, 1913, p. 566. (H. A. W.)

Allen's Commercial Organic Analysis. Vol. 8. 4th Edition. Entirely re-written. Royal 8vo., pp. 706. *Churchill.* Net 21s.

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Nagel, Ph. D., Oskar. Producer Gas—Fired Furnaces. 200 p., 8vo. 200 illustrations. *Vacher & Sons, Ltd.*, London, S.W. Net 8s. 6d.

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Poynting, J. H., and Thomson, Sir J. J. A Text Book of Physics. Electricity and Magnetism. Parts I. and II. Statics, Electricity and Magnetism. 8vo., pp. 360. *C. Griffin.* 10s. 6d.

Price, T. Slater, and Twiss, D. F. A Course of Practical Organic Chemistry. 2nd Edition. 8vo., pp. 254. *Longmans.* 3s. 6d.

Shone, I. The Evolution of Greater Britain's Antiseptic and Town Sewage-Drainage Systems of the Twentieth Century. 8vo. *Spon.* Net 25s.

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The Coal Resources of the World. An enquiry made upon the initiative of the Executive Committee of the 12th International Congress, Canada, 1913. 3 vol. and atlas. 4to. *Dulau*, London, 1913. Net £5 5s.

Watkins, C. A. Chemistry for the Engineer, Electrician and the Practical Man. Cr. 8vo. *Jl. Soc. Chem. Industry*, 15th Nov., 1913. *Spon*, London, 1913. Net 7s.

Westcott, H. P. Handbook of Natural Gas. Cr. 8vo. *Spon.* Net 17s.

Wright, H. B. Transmission Line Formulae for Electrical Engineers and Engineering Students. Cr. 8vo. *Constable.* Net 8/6.

Abstracts of Patent Applications.

(C.) 615/12. Alfred Argles. Improved amalgamation process. 25.11.12.

This application relates to an amalgamation process for the treatment of ores containing sulphides, tellurides or selenides, which consists in suitably oxidising to sulphate, etc., and grinding with mercury which is thus converted to sulphate: ferrocyanide of potash is then added to convert the mercuric sulphate to cyanide of mercury, and amalgamation of the gold, etc. is thus more efficiently attained.

(C.) 111/13. John Hawthorne Wilson (1), Robert Craib (2). Improvements in sand pumps. 5.3.13.

This application covers a centrifugal sand pump, comprising a rotating blade impeller, characterized by the inlet side of the impeller being projected into the casing inlet passage as an annular extension of substantial length such as to impart substantial rotary motion to the incoming pulp and thereby minimise the abrasive impact of the pulp on the blades of the impeller; also with a replaceable ring arranged between the end of the rotating impeller and the inlet pipe connection.

(C.) 125/13. Edgar Arthur Ashcroft. Improvements in, and apparatus for, the electrolytic manufacture of light metals, caustic alkalies in pure anhydrous condition, and alloys of light metals with heavier metals or the continuous treatment of such alloys for obtaining final products. 8.3.13.

The above application is for improvements, and an improved apparatus for the electrolytic manufacture of light metals, caustic alkalies and alloys of light metals with heavier metals. The chief point is an arrangement for continuous treatment of such alloys for obtaining final products and preventing reversible reactions at the place of contact of the electrolyte with the alloy in the cells, for which purpose, amongst other suggestions, the applicant employs an exposed cathode area, restricted within certain limits. He also employs beside the two cells another intermediate reservoir to keep up the equilibrium.

(C.) 270/13. Charles Washington Merrill. Process for precipitating and recovering valuable metal bearing materials from solutions and precipitants for use in connection therewith. 13.6.13.

This application refers to improvements in the use of zinc dust for precipitation of gold-bearing solutions by exclusion of the atmosphere from the press used which is always kept full of solution for that purpose.

(C.) 505/13. Gabriel Jacobus Luyt Wannenburg. A dust preventer. 22.10.13.

The above application deals with an apparatus for preventing the escape of dust in rock drilling and the like operations.

This consists of a metal tube with rubber washers, and is so adjusted with a spiral spring that it seals hermetically, as the applicant claims, the hole in the rock.

This tube is connected with the rock drilling apparatus. The said apparatus has only one outlet through which the dust is allowed to escape, and that outlet is let into a receptacle of water.

(C.) 547/13. Lamertine Cavaignac Trent. Apparatus for the treatment of ores or other materials for recovery of the metals therein. 17.11.12.

This application refers to an apparatus which is intended for the purpose of continuous thickening of slime pulp, while at the same time replacing the valuable solution by means of barren solution or a water wash.

A continuous overflow of settled and clear solution is taken from a rim launder on the periphery of the collecting vessel: at an intermediate depth a thick pulp is withdrawn by means of a circular perforated suction pipe, passed through a pump and returned to the vessel still lower down to a revolving distributor to which are fixed scrapers which move the completely settled slime to two outlets at the very bottom of the vessel.

The revolving distributor may be either partly or wholly supplied with barren solution for the purpose of replacing the solution in contact with the incoming slime pulp.

(C) 548/13. Alfred Argles. Improvements in apparatus for the treatment of ores or other materials for recovery of the metals therein. 17.11.13.

The application describes a complex apparatus, illustrated by 75 figures, and having 23 claims for the treatment of ores in general and battery tailing in particular. It comprises distributing and screening hoppers, a cradle grinding mill, a concentrator, leaching vat, precipitating box, an amalgamating plate, and an amalgamator and magnetic separator.

(C.) 562/13. John Emery Bucher. Method of producing metals. 20.11.13.

This application refers to improvements in the manufacture of the alkali metals, particularly potassium. By the action of heat upon the cyanides in the presence of iron or aluminium the pure metal may be distilled from the re-action mixture at temperatures from 500 to 1,200° C. As no oxygen need be present the formation of dangerous explosive compounds such as $K_6C_6O_6$ is entirely avoided.

(C.) 579/13. Douglas William Stacey. An improved process and apparatus for the separating of a solution containing gold or other metals or material from finely crushed insoluble substances. 3.12.13.

The inventor claims a patent (1st) on an improved process and (2nd) on an apparatus for separating solution containing gold or other metals or material from finely crushed insoluble substances with more particular reference to the cyanide process for extracting gold.

(C.) 582/13. Hans Andrew Hansen (1), Joseph Milton Schmueser (2). Improvements in linings for ball or pebble mills. 4.12.13.

This application relates to improvements in tube mill linings, which consist in providing segments or units with a curved rib on one edge and capable of being held in position by the one wedge shaped unit which is bolted to the casing.

The drawings show its adaption to a conical shape tube mill.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P). 595/13. Charlie Beatson McMullin. A hand lashing rake. 13.12.13.

(P.) 596/13. Josephus Davey. Improvements in and relating to dust allayers for use with rock drills and the like. 15.12.18.

(P.) 597/13. Harold Stockman. Improvements in apparatus for concentrating ore and the like. 15.12.13.

(P.) 598/13. George Henry Frederick Bazier. An automatic relief valve as applied to leaves used in the vacuum filtration of gold ore slimes. 17.12.13.

(P.) 599/13. Cruffydd Vaughan Trevor. Hose clamp for drilling wet up or down holes. 17.12.13.

(P.) 600/13. Henry Marrs. An invention to catch and confine the dust which results from dry rock-drilling or boring in mines and other places. 18.12.13.

(P.) 601/13. Frederick Burnette. The transportable bunker or muckbox. 18.12.13.

(P.) 602/13. John Ernest Roux Adendorff. Improvements in or relating,to fuse lighters, torches, and the like. 19.12.13.

(C.) 603/13. The Knowles Oxygen Co., Ltd. (1), Robert William Grant (2). Improvements relating to the production of hydrogen and oxygen by electrolytic processes. 19.12.13.

(C.) 604/13. Ester & Company, Limited (1), Arthur Percy Strohmenger (2). Improvements in the preparation of metals and alloys. 19.12.13.

(C.) 605/13. David Corrie (1), George Alfred Ashcroft (2). Improvements in the manufacture of detonators. 19.12.13.

(C.) 606/13. Edward Clinton Smith. An improved bottle mouth. 19.12.13.

(C.) 607/13. Joseph Heinrich Reineke. Improvements in the method of and means for the electric transmission of signals and speech in mines. 19.12.13.

(C.) 608/13. George James Coles. Improvements in and relating to automatic couplings for railway and similar vehicles. 19.12.13.

(P.) 609/13. The Benoni Tshisa Stick Syndicate, Limited. An improvement in the means for lighting fuses for use in blasting or for analagous purposes. 20.12.13.

(P.) 610/13. John Pountney Udal. Improvements in the manufacture of nitro-glycol. 20.12.13.

(P.) 612/13. Armour Hall. Pipe coupling. 24.12.13.

(C). 613/13. Emanuel Rosenburg. Improvements relating to synchronous dynamo electric machines. 27.12.13.

(C.) 614/13. Thomas Kemplay Irwin. Improved apparatus for the aeration and gasification of liquids. 27.12.13.

(P.) 616/13. Frederick Joseph Trump (1), Frederick Wartenweiler (2). Improvements in binding dust and the like in and about mines. 27.12.13.

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THE JOURNAL

OF THE

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Vol. XIV. MARCH, 1914. No. 9.

Proceedings

AT

Ordinary General Meeting, March 21st, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, March 21st, 1914, Prof. G. H. Stanley (Vice-President) in the chair. There were also present:

37 Members: Messrs. J. E. Thomas, J. Gray, C. Toombs, John Watson, A. Whitby, Dr. W. A. Caldecott, A. F. Crosse, W. R. Dowling, Dr. J. Moir (Members of Council), A. S. Allam, H. C. Boydell, J. M. Campbell, Prof. J. Cellier, R. A. Cooper, W. M. Coulter, E. H. Croghan, J. G. Dealy, T. N. Dewar, J. M. Dixon, H. N. Dungey, R. Dures, H. W. Gill, C. J. Gray, W. E. John, W. A. Ledingham, E. A. Osterloh, F. D. Phillips, W S. V. Price, E. Roberts, D. F. Scott, C. B. Simpson, A. L. Spoor, A. R. Stacpoole, W. E. Thorpe, G. A. Watermeyer, E. M. Weston and H. B. Williamson.

8 Associates and Students:—Messrs. G. H. Blenkinsop, O. A. Gerber, R. W. Irwin, F. J. Pooler, P. Scatterty, H. Stadler, H. Ward and J. A. Woodburn.

9 Visitors, and Fred. Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General meeting were confirmed.

NEW MEMBERS.

Dr. James Moir and Mr. A. Whitby were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman declared that the candidates for membership had been elected, as follows:—

GUINEY, JAMES THOMAS, East Rand Proprietary Mines, Ltd., P. O. Box 185, East Rand. Mill Manager.

WATERMEYER, GOTTFRIED ANDREAS, A.R.S.M., P. O. Box 1242, Johannesburg. Surveyor.

GENERAL BUSINESS.

SIXTH INTERNATIONAL CONGRESS OF MINING, METALLURGY, ENGINEERING AND PRACTICAL GEOLOGY.

The Secretary: The Council has received an invitation from the President, Vice-Presidents and Executive Committee of the Sixth International Congress of Mining, Metallurgy, Engineering and Practical Geology, which will be held in London from July 12th to July 17th, 1915. It is under the distinguished patronage and with the co-operation and support of the University of London, the Imperial College of Science and Technology, and the principal Engineering and Mining Societies in England. A cordial invitation is extended to this Society to take part officially in the Proceedings and to attend the Meetings. The Council has accepted the invitation on behalf of the Society and hopes that any members going Home in 1915 may be able to attend. I shall be glad to send them further particulars as they come to hand.

ANNUAL MINING EXHIBITION.

The Secretary: With regard to the Mining Exhibition, it will have been noticed that the prospectus has been printed in the *Journal*, copies of which have been widely distributed. Since it was issued, the Rand Mutual Assurance Company, through their Prevention of Accidents Committee, has offered three prizes of, £25, £15, and £5 for the best "Safety First" device or schemes of procedure, which may be exhibited at the Mining Exhibition in May, and, further, a prize of £10 for the best type of enclosed lamp exhibited, suitable for use by miners whilst handling explosives underground; the cheapness and simplicity of construction of which will be material considerations. The conditions will be published shortly. It is also intended to give medals in addition to the money prizes, but it will be at the discretion of the Committee to reduce or withdraw any prizes or medals should the exhibits, in the opinion of the Judges, not be deemed sufficiently good. I hope members will make the competition as widely known as possible, and on receipt of application I shall be pleased to issue the conditions to anyone requiring them.

FINE SCREENING.

Mr. A. McA. Johnston (*Past-President*): The use of fine screening for grading of crushed ore is so universal that I need offer no apology for bringing to your notice some screening which we have imported mainly for testing slime charges and residues. Up till the present time the finest screen in use here has been the 200 mesh (approximate aperture 0·0030 in.). This I show you now has 260 mesh (approximate aperture 0·0023 in.).

The dimensions of the two screens are as follows:—

	260 Mesh.	200 Mesh.
Thickness of screen or cloth, inches ...	0·0035	0·0045
Diameter of wire, inches	0·0014	0·0018
No. of threads per linear inch	260	200
No. of holes per square inch	67,600	40,000
Diameter of aperture (calculated), inches ...	0·0024	0·0032

The diameter of the wire was measured with a micrometer, and this result checked by a formula similar to that published in the June supplement on the Standardisation of Battery Screening, 1906, p. 394. The formula for copper screening is:—

Diameter $= 0{\cdot}00296\sqrt{W}$, where W is the weight in milligrams per linear inch of the wire. In each case absolute accuracy is impossible owing to the twilling of the wire rendering the micrometer reading slightly inaccurate, whilst the same reason applies to the measurement of an inch of straight wire. Instead, therefore, of weighing one linear inch of the wire, one square inch of the screen was weighed and the result divided by 520. This gives a calculated aperture diameter of 0·0023 in., which may be taken as more approximately correct than that based on the micrometer reading.

An examination of the cloth shows that the apertures are not all equal—a condition which is found also in the 200 mesh. The difficulty of obtaining divisions equal in size and also square is extremely great, but strides seem to be being made in this direction, and we hope soon to see the perfect screen even in such a texture as this. The figures obtained from this screening are valuable in that a closer approximation of the quantity of fine sand in any charge is obtained, thus whereas 1·5% of +200 mesh in a slime differs little from 3·5% +200, when the same charge is tested on the finer screen 9% and 19% respectively may be obtained. Such figures usually help materially in determining the reason for a higher or lower assay value. With the use of this screen we are further able to check results already shown by several of the coarser screens.* One of these is that the coarser sand in a slime charge is poor in value and does not carry much of the gold. Thus, some figures obtained were:—

Slime Original.

Grading	Percentage	Assay Value dwt.	Total Gold Per cent.
+260 (aper. = 0·0023″)	14·4	0·38	4·8
−260 ,,	85·6	1·27	95·2

Slime Residue.

Grading			
+260 (aper. = 0·0023″)	13·8	0·16	12·5
−260 ,,	86·2	0·18	87·5

Dr. W. A. Caldecott (*Past President*): Mr. McArthur Johnston has explained the utility of this screening, and I may add that Dr. Moir told me that he hopes screening of this nature may be of use in the process of determining the dust in mine air,† as the 200-mesh screen is found to be some coarse.

Mr. Andrew F. Crosse (*Past-President*): I am inclined to think that my friend Mr. Stadler's way of elutriation is more accurate for separating fine particles when beyond a certain degree of fineness. I should prefer that method to the use of a screen.

The Chairman: I think the thanks of the meeting are due to Mr. Johnston for bringing forward this note. Personally, I remember it is not so long since that 200-mesh screening was thought to be unattainable, and this 260-mesh to day marks a very big advance.

PERSONAL ALLUSIONS IN PAPERS.

Mr. F J. Pooler (*Associate*) drew attention to what he considered to be the personal nature of certain contributions to discussions. He thought such personal allusions should be avoided in future.

The Chairman said he thought the Society always prided itself on the fact that it gave free leave to anyone to express his opinions, and did not think that the remarks to which Mr. Pooler drew attention were meant to be personal attacks.

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* cf. Rand Metallurgical Practice, pp. 374-5.

† See this *Journal*, Nov. 1902, p. 186.

RELATION OF FALLS OF ROOF IN COLLIERIES ON THE MIDDELBURG COALFIELD TO WEATHER CHANGES.

By CHAS. J. GRAY, A.R.S.M., F.G.S., M.I.M.E. (Member).

It is well known that falls of roof kill more men than any other form of colliery accident. They can be prevented by a lavish use of timber but their causes must be fully realised before such falls can be guarded against most economically. It is therefore hoped that the following investigation will prove useful.

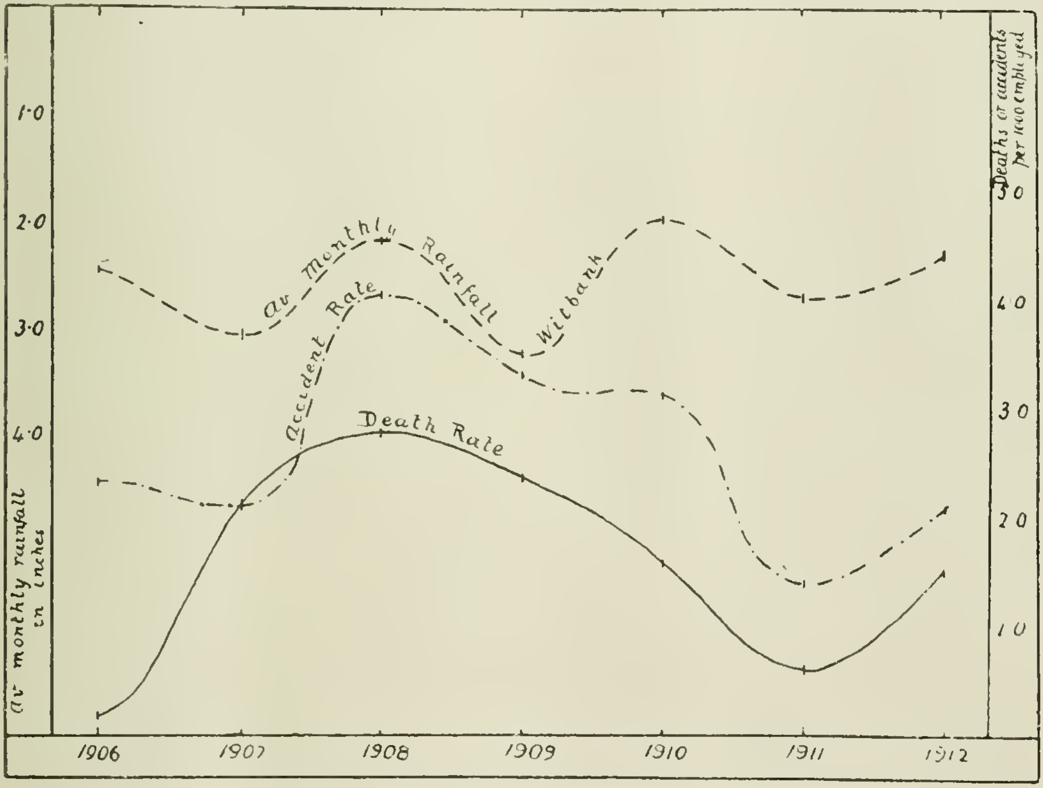

FIG. 1.—Relation of Accident and Death Rate to Rainfall.

In the year 1912 the accident rate from falls of roof in the Middelburg coalfield increased, despite apparent greater care in working. The writer noticed a concentration of fatalities and accidents in the five months June to October, 9 out of 11 fatalities and 10 out of 15 accidents occurring in those months. That the explanation was not to be found in fluctuations either in the numbers employed or in output per employee was shown by investigation of the labour and output returns for the collieries at which the accidents occurred. The question then arose as to whether there was any connection between the increased number of accidents, concentrated in the dry months, and the unusually prolonged dry season. In order to decide that question two lines of investigation were followed:—

(1) Comparison between the annual accident rates and the rainfall.

(2) Comparison between the number of accidents in particular months and the rainfall in those months.

1.—Following the first line it was found that for Witbank, the most central position on the coalfield, rainfall records were available for the seven years 1906 to 1912 only, and that comparison must therefore be limited to that period. The comparison which works out as follows is shown graphically in Diagram I.

It will be noticed that the rainfall rose and fell in alternate years, rising three times and falling three times. The death rate also rose three times and fell three times, but four out of the six changes were in a contrary direction to the changes of rainfall. The accident rate rose twice and fell four times and five out of the six changes were contrary to changes of rainfall.

	1906.	1907.	1908.	1909.	1910.	1911.	1912.
Falls of Colliery Roofs.							
*Death rate per 1,000	0·19	2·11	2·78	2·38	1·58	0·62	1·54
*Accident rate	2·34	2·11	4·07	3·30	3·17	1·39	2·10
Rainfall.							
†Monthly mean inches	2·11	3·02	2·15	3·19	1·92	2·63	2·25

The changes from year to year were :

	1906-7.	1907-8.	1908-9.	1909-10.	1910-11.	1911-12.
Death Rate	+1·92	+0·67	−0·40	−0·80	−0·96	+0·92
Accident Rate	−0·23	+1·96	−0·77	−0·13	−1·78	+0·71
Rainfall	+0·61	−0·87	+1·04	−1·27	+0·71	−0·38

Taking the years in which the death rate and accident rate rose, and summing the changes, we find that +3·51 death rate corresponds with −0·64 rainfall, and +2·67 accident rate with −1·25 rainfall.

Similarly for the years in which the rates fell, we find that −2·16 death rate corresponds with +0·48 rainfall, and −2·91 accident rate with +1·09 rainfall.

In considering the statistics more carefully it must be remembered that changes in the amount of precaution taken, affect accident and death rates. Clearly there has been increased precaution in recent years, but how it has affected the accident rate for particular years is doubtful. To partly eliminate the precaution factor it seems however useful, and not unreasonable as a crude approximation, to assume that uniform increase in precaution since 1908, when both the accident and death rates reached their maximum, accounts for the difference between the 1908 and 1912 rates. That assumes an improvement of 0·31 per annum in the death rates and 0·49 per annum in the accident rates, so that the rates apart from increased precaution would have been :—

	1906.	1907.	1908.	1909.	1910.	1911.	1912.
Death Rate	0·19	2·11	2·78	2·69	2·20	1·55	2·78
Accident Rate	2·34	2·11	4·07	3·79	4·15	2·86	4·07
Rainfall							
and the changes	2·41	3·02	2·15	3·19	1·92	2·63	2·25

	1906-7.	1907-8.	1908-9.	1909-10.	1910-11.	1911-12.
Death Rate	+1·92	+0·67	−0·09	−0·49	−0·65	+1·23
Accident Rate	−0·23	+1·96	−0·28	+0·36	−1·29	+1·21
Rainfall	+0·61	−0·87	+1·04	−1·27	+0·71	−0·38

Now, in every case, the accident rates change in a direction contrary to the rainfall. The chances against that being a fortuitous coincidence are great. The same result could be obtained by crediting a much smaller effect to increased precaution.

Summing the changes as was done with the unadjusted rates, we find

+3·82 death rate corresponds with −0·64 rainfall
−1·23 ,, ,, ,, +0·48 ,,
+3·53 accident ,, ,, ,, −2·52 ,,
−1·80 ,, ,, ,, ,, +2·36 ,,

The crude correction has made the correspondence between a change in the accident and death rates and a change in the reverse direction in the rainfall, still more marked than it was.

From the foregoing it appears very highly probable that a decrease in the annual rainfall at Witbank, corresponds with an increase in liability to accident from falls of roof in the collieries, and *vice versa*.

2.—Proceeding now to compare the numbers of accidents from falls of roof in particular months with the rainfall in those months, we find that for the seven years 1906 to 1912 the figures were as follows :—

Month.	*Deaths.*	*Accidents.*	*Average rainfall.*
January ...	5	11	6·37 inches.
February ...	3	4	4·52 ,,
March ...	5	6	3·15 ,,
April ...	7	11	1·62 ,,
May ...	3	9	0·53 ,,
June ...	8	10	0·13 ,,
July ...	6	14	0·05 ,,
August ...	7	10	0·45 ,,
September ...	4	6	0·65 ,,
October ...	11	11	3·01 ,,
November ...	0	4	5·04 ,,
December ...	5	9	4·59 ,,

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* The figures given being averages of monthly rates, are not exactly accurate for the years, but they are sufficiently close for all practical purposes.

† The rainfall temperature and humidity statistics used in this paper were kindly supplied by the Meteorological branch of the Irrigation Department.

Diagram 2 shows the relation graphically.

As would be expected with such small numbers of accidents, there is irregular fluctuation from month to month. A general tendency for the numbers to be high in the dry months is however noticeable. Thus of the five months in which the number of deaths was above the average 5·3, four are months in which the rainfall was below the average 2·51, and of the eight months in which the number of accidents was above the average 8·75, five are months in which the rainfall was below the average.

The effect of fortuitous fluctuations can be lessened by taking longer periods than one month for comparison. The figures taken quarterly become :—

	Deaths.	*Accidents.*	*Average monthly rainfall.*
January to March ...	12	21	4·68 inches.
April to June	18	30	0·76 ,,
July to September ...	18	30	0·38 ,,
October to December	16	24	4·21 ,,

Taken half yearly and putting the six dry months together we have :—

	Deaths.	*Accidents.*	*Average monthly rainfall.*
Oct. to March	28	45	4·44 inches.
April to Sept.	36	60	0·57 ,,

A more direct comparison between accident and rainfall statistics is given in the following figures for the years 1906 to 1912.

Number of months in which rainfall was		*Accidents.*	*Accidents per month.*
Less than 1 inch	38	58	1·52
More than 1 inch	46	47	1·02

Accident statistics are available for a longer period than rainfall statistics and they have been examined from 1903 to see whether the concentration of accidents in the dry months is consistent from year to year. The figures are :—

Year.	*Deaths.* Jan. to March. Oct. to Dec.	April to Sept.	*Accidents.* Jan. to March. Oct. to Dec.	April to Sept.
1903	2	5	2	5
1904	0	0	1	2
1905	5	2	5	4
1906	1	0	5	7
1907	8	4	8	4
1908	6	7	9	10
1909	3	10	6	12
1910	5	5	10	10
1911	1	3	2	7
1912	4	7	3	12
	35	43	51	73

Deaths were most numerous in the dry months in five years and most numerous in the wet onths in three years.

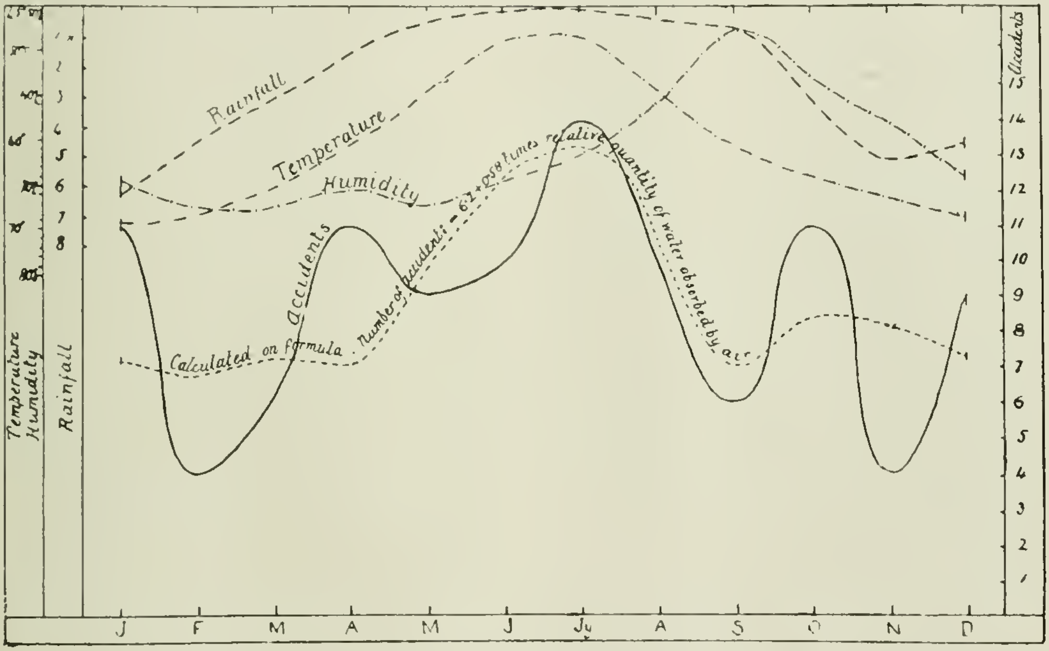


FIG. II.—Relation of Accidents in Particular Months to Rainfall, Temperature and Humidity—(1906-1912 inclusive).

Accidents were most numerous in the dry months in seven years and in the wet months in two years.

The foregoing comparisons show that liability to accident from falls of roof is almost certainly much greater in the dry winter months than in the summer months. Apparently it is nearly 50% greater.

Both lines of investigation have now been followed, and it has been found that accidents from falls of roof on the coal field have increased and decreased both annually and seasonally as the rainfall decreased and increased. That being so, it is morally certain that accidents and rainfall are connected. Either rainfall affects accident statistics, (for it is unimaginable that accidents affect rainfall), or the same seasonal phenomena affect both accidents and rainfall. On consideration it seems that the latter must be the case, for percolation through the strata, which is the only known way in which the rainfall can intervene, would increase and not reduce the accident rate in the wet summer months. Though percolation is known to cause a few accidents, and may partly account for the number in December and January being higher than in November and February its influence must be slight.

Rejecting rainfall as the direct cause of the fluctuation in accident rates, we proceed to consider changes in temperature and air humidity which are closely connected with rainfall. As complete records of those changes at Witbank are not available, the Middelburg records have to be used.

Taking temperature first we can make the following comparisons :—

	1906.	1907.	1908.	1909.	1910	1911.	1912.
Accident rate ...	2·34	2·11	1·07	3·30	3·17	1·39	2·10
Mean Temperature ...	59·74	61·07	61·69	61·04	60·62	60·97	60·43 deg. F.

	1906-7.	1907-8.	1908-9.	1909-10.	1910-11.	1911-12.
Accident rate ...	−0·23	+1·96	0·77	−0·13	−1·78	+0·71
Mean Temperature ...	+1·33	+0·62	−0·65	−0·42	+0·35	−0·54

Month.	*Accidents.*	*Mean Temperature.*
January ...	11	69·2 deg. F.
February ...	4	68·7 ,,
March ...	6	65·6 ,,
April ...	11	60·2 ,,
May ...	9	54·5 ,,
June ...	10	48·9 ,,
July ...	14	48·3 ,,
August ...	10	54·1 ,,
September...	6	60·6 ,,
October ...	11	64·1 ,,
November...	4	66·6 ,,
December ...	9	68·6 ,,

The monthly comparison is shown graphically in Diagram 2.

It will be seen that the annual changes of mean temperature are very slight, and that there is no marked correspondence between them and changes in accident rates. Three times accident rates and mean temperatures changed in the same direction, and three times in a reverse direction. A total decrease of 2·91 in accident rates corresponded with an increase of 0·61 in mean temperature, and a total increase of 2·67 in accident rates with an increase of 0·08 in mean temperature.

In the monthly comparison we see a tendency to correspondence between concentration of accidents and low temperature, which is more clearly shown by the following figures :—

	Accidents.	*Mean Temperature.*
October to March	45	67·1
April to September	60	55·3

Now, taking humidity as the basis of comparison we find the figures to be (see p. 399) :—

Month.		*Accidents.*	*Mean Humidity.*
January	...	11	69·6
February	...	1	72·4 ,,
March ...	...	6	72·1 ,,
April ...	...	11	70·6 ,,
May ...	...	9	72·2 ,,
June ...	...	10	69·7 ,,
July ...	...	14	66·9 ,,
August	...	10	60·9 ,,
September	...	6	52·3 ,,
October	...	11	57·6 ,,
November	...	1	62·9 ,,
December	...	9	68·9 ,,

The monthly comparison is shown graphically in Diagram 2.

In the annual figures, correspondence between increase of accidents and decrease of humidity, and *vice versa*, is almost as marked as the correspondence in the case of rainfall. Four times the changes were in reverse directions. A decrease of 2·91 in accident rates coincided with an increase of 13·6 in mean humidity and an increase of 2·67 in accident rates with a decrease of 13·0 in humidity. If allowance is made for improvement of accident rates owing to increased precaution, as was done in examination

	1906.	1907.	1908.	1909.	1910.	1911.	1912.
Accident rate	2·34	2·11	4·07	3·30	3·17	1·39	2·10
Mean Humidity	62·6	72·0	61·5	71·4	68·2	65·7	63·2%

	1906-7.	1907-8.	1908-9.	1909-10	1910-11.	1911-12.
Accident rate	−0·23	+1·96	−0·77	−0·13	−1·78	+0·71
Mean Humidity	+9·4	−10·5	+9·9	−3·2	−2·5	−2·5

of rainfall comparisons, correspondence becomes still more marked.

The monthly comparison shows that the maximum concentration of accidents comes before the time of lowest humidity.

Taken half-yearly in accordance with seasons the comparison is :—

	Accidents.	*Mean Humidity.*
October to March ...	45	67·3
April to September ...	60	65·4

If the six months of lowest humidity are put together the comparison becomes :—

	Accidents.	*Mean Humidity.*
January to June ...	51	71·1
July to December ...	54	61·6

It will be seen that the correspondence between the distribution of accidents in a year and the changes of humidity in that year is not very marked.

We have now found (a) that both annually and seasonally the accident rates change in a reverse direction to the rainfall, (b) that seasonally they change in a reverse direction to the temperature and (c) that annually they change in a reverse direction to the humidity of the atmosphere.

The rainfall is directly connected with the interaction of temperature and humidity and it remains to be seen whether by combining temperature and humidity in one consideration we get correspondence with accident rates both annually and seasonally.

The only way in which temperature and humidity can affect colliery roofs is by drying or moistening. If the air entering the mine is cold and saturated it absorbs moisture to keep it saturated as it rises in temperature. If it is hot and saturated it deposits moisture as it falls in temperature. If unsaturated when it enters the mine it absorbs or deposits moisture according to the exact conditions with regard to moisture and temperature. The effect of air on a mine roof depends not only on the temperature and percentage of water vapour in the air, but also on the quantity of air circulating and that is affected by the difference of temperatures within and without the mine. As mean annual temperatures have varied little, temperature has only slightly influenced the relative drying effects of the air in different years. Changes in that effect have depended almost wholly on humidity. The relation of high annual accident rates to low mean humidity has already been observed. Though temperature varied little from year to year, it varied considerably from month to month and owing to their double effect such variations influenced the drying or moistening of the roof more than variations in humidity.

It is impossible to calculate accurately the effect of the air, but some further progress with our investigation may be made by assuming as crude approximations that the air, when in the mine, reaches the mean temperature (60·8° F.) and becomes saturated and that (as nearly all colliery ventilation in the district is natural), the quantity of air circulating varies as the square root of the difference of mine and surface temperatures. On those assumptions we arrive at the following figures (see p. 400) :—

The results are plotted as a curve in Diagram 2.

It will be seen that the last column shows a drying action in each month with maximum in July which is the month in which accidents reach their maximum.

Making a quarterly comparison we have :—

	Deaths.	*Accidents.*	*Average relative drying.*
January to March ...	12	21	1·42
April to June ...	18	30	6·03
July to September	18	30	7·03
October to December	16	24	3·03

The correspondence is marked.

We have now seen that considering temperature and humidity together in their influence on the drying of mine roofs, we get correspondence both annually and seasonally, with accidents from falls of roof just as we did between rainfall and such accidents. The correspondence is however much more likely to be one of cause and effect. Indeed it is what would be expected, for if we remember that in the collieries under consideration, the roofs under which men work are nearly always either shale or impure coal, and that such materials contract when dried, we shall expect an increased absorption of water by the air-current to increase the number of roof falls. If the roofs were of sandstone drying should have relatively little effect, and for Natal collieries, where that is generally the case, readily available statistics for the years 1905 to 1909 show 58

Month.	Accidents.	Difference of temperature from mean temperature.	Relative Quantity of air circulating (A.)	Weight of vapour per cu. ft. air at mean temperature and humidity for month. Grs.	Weight of water required to bring air to saturation at mean temperature 60°s (B). Grs.	Relative Quantity of water absorbed, or relative drying effect (A B).
January ...	11	+ 8·4	2·90	5·3	+0·6	1·7
February...	4	+ 7·9	2·82	5·6	0·3	0·8
March ...	6	+ 4·8	2·20	5·1	0·8	1·8
April ...	11	– 0·6	0·78	4·1	1·8	1·4
May ...	9	– 6·3	2·51	3·5	2·4	6·0
June ...	10	–11·9	3·45	2·8	3·1	10·7
July ...	14	–12·5	3·54	2·5	3·4	12·0
August ...	10	– 6·7	2·59	2·9	3·0	7·8
September	6	– 0·2	0·45	3·1	2·8	1·3
October ...	11	+ 3·3	1·82	3·8	2·1	3·8
November	4	+ 5·8	2·41	4·5	1·4	3·4
December	9	+ 7·8	2·79	5·2	0·7	1·9

accidents from falls of roof in the months April to September and 54 in the other months of the year.

To the writer it seems that the foregoing investigation fairly leads to the conclusion that though many things affect accident rates from falls of roof on the Middelburg coalfield, the influence of the weather runs through those rates like the tide through disturbances of the sea.

It is even possible to form some crude idea of the extent of that influence for, if we take the last preceding table, and assume that the number of accidents in any period is made up of a constant plus a multiple of the relative drying figures, calculation indicates that approximately eight in the summer months and 23 in the winter months, making in all 31 out of the 105 accidents, or roughly 30 %, were due to weather conditions. Similarly, dealing with the previous table, a formula, number of accidents equals 6·2 + 0·58 times relative drying figure, is deduced as giving a monthly approximation, and the results are plotted in Diagram 2.

That drying causes certain roof falls is well-known, but whether or not changes in drying effect, due to atmospheric changes, have been recognised as having important influence on accidents is doubtful. The effect of seasonal deposit of moisture has however been remarked upon. For instance in A. R. Sawyer's "Accidents in Mines," page 86, the following passage occurs "in the same way atmospheric vapour condensed by coming in contact with strata colder than the air which conveys it may have a loosening effect on them. This is more likely to be the case in summer with shallow mines and unextended workings."

In South Africa atmospheric conditions can seldom cause deposit of moisture on the roof near the working face, as that can occur only in hot moist weather and when the face is near the mine entrance. Generally such condensation as there may be, will be on the intake airways. On the other hand the air current will rarely reach the working face so warm and charged with water vapour as to be saturated at the temperature of the roof and therefore unable to exert a drying influence. That being so it is natural that occasionally a sheet of roof shale or a block of roof coal, which has been sounded and examined without detection of any open crack, should fall not long afterwards. Such things will occur most frequently when the drying effect is greatest which is in cold dry weather and more particularly on winter nights.

If managers and men recognise the weather effect they will apply their own safeguards, but to the writer the practical lessons particularly for winter guidance, seem to be (a) sound and examine newly exposed roof frequently, or prop it systematically, (b) prop the under side of every slip which if opened may cause a fall and (c) when practicable course the air so that areas of bad roof shall be on the return side. If those things are done, a future investigator may fail to find any indication of weather influence on accidents from falls of roof in the Middelburg coalfield.

Prof. J. Cellier *(Member)*: In rising to propose a vote of thanks to Mr. Gray for his paper, I agree with him entirely, especially where he draws attention to the scarcity of timber in this country. Probably it is largely for that reason that a system of pillar and stall is adopted here, also owing to the size of our seams; however, his analysis has been most instructive. On the Rand in the winter months of 1909 we all thought because it had rained a lot in the early part of the year that that was why we had such a lot of accidents from fall of roof; however, it appears now to be quite the contrary. I was on a very bad mine at that time, and am glad now to find a reasonable solution of the matter.

The Chairman: In these days when safety in mining operations is engaging so much attention, I think this is really an important contribution on the subject. I therefore have pleasure in seconding Prof. Cellier.

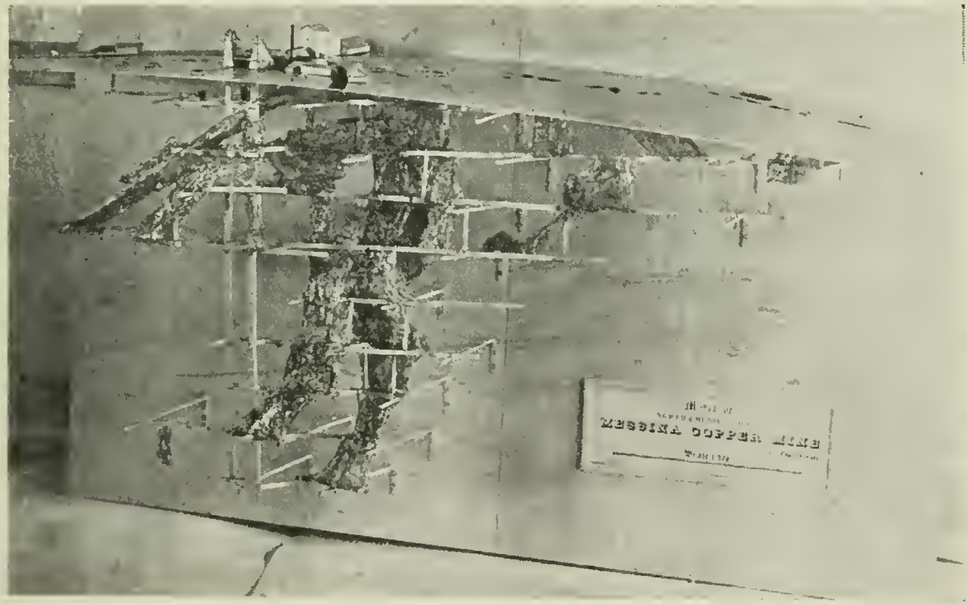
MINING COPPER ORES AT MESSINA.

(Read at August Meeting, 1913.)

By J. ALLAN WOODBURN, M.Inst.M.M. (Associate).

REPLY TO DISCUSSION.

Mr. J. Allan Woodburn *(Associate)*: I have not written out a formal reply to this paper, because the discussion hardly warrants it. I have to thank Mr. Chambers for a very valuable addition to the paper in connection with the ancient workings and the geological features of the Northern Transvaal; and I have also to thank Mr. Veasey very much indeed for the appreciative remarks which he has made with regard to it. I certainly thought that some remarks might have been made with regard to the advisability or otherwise of sinking two shafts so close together as was shown in the diagram; but, as no remark has been made either for or against and no criticism has been given on it before the Society, it does not call for any further remarks from me. I have promised to send a model I have made of the Mine to the forthcoming Exhibition. I am doing this in the hope that models of mines will be made more frequently than they have been in the past. I have had some experience of trying to make models of peculiar, vertical, erratic reefs, and I have had great difficulty indeed in getting them to represent the actual conditions so that anyone could recognise them; but I think this time I have been able to make a model which, at least, is an improvement on many of the models I have yet seen. A photograph of the model is reproduced here. I hope when it is at the Exhibition that it will receive some attention, so that further improvements can still be made, because we at Messina found it invaluable to the surveyor, the mine captain and myself in following up the development of the mine and in laying off future development; and if further improvements, which I feel sure can be made, are made, that in many mines, especially where the ore-bodies are faulted, or lying at varying inclinations, valuable assistance will be got by having a model made, in addition to keeping plans and sections.



I hope those who are interested in mining will criticise this model, and suggest any improvements which they think may be helpful.

Mr. E. M. Weston *(Member)*: I should like to thank Mr. Woodburn for his paper. I may say that the President asked me during my holidays if I could not write some criticisms on the paper, but it seemed to me such an example of wise and prudent mining practice that I did not find anything on which to base criticisms. However, with regard to twin shaft sinking, I understood that as the main ore shoots on the mine are concentrated near this particular shaft, and as it was not advisable to increase its size, that it was quite legitimate to sink another shaft near it to enable the mine output to be handled better. I know, however, that Mr. Woodburn is a great believer in the twin shaft idea for developing mining areas on the Witwatersrand, especially on the East Rand. Personally I do not agree with him I think at some future time some member might promote quite an interesting discussion by giving a paper on such a proposed method for development. Then Mr. Woodburn could defend his theory that twin shafts, sunk close together, are the most economical way of developing such Rand areas.

The Chairman : I am sure we are very much indebted to Mr. Woodburn for his promise to show this very interesting model at our Exhibition, and I think, at the same time, that it will serve as a lead to many others of our members to come forward with models of the mines with which they may be connected, or with appliances employed thereon. At the same time, I hope Mr. Woodburn will make us to a still greater extent his debtors by letting us hear something in the future as to the metallurgical side of the Messina operations, which, from rumours we hear, are extremely interesting.

VENTILATION OF THE MINES OF THE RAND : THE PROBLEM OF OBTAINING HEALTHIER CONDITIONS.

(Read at September Meeting, 1913.)

G. H. Blenkinsop, M.Inst.M.M. (Associate).

REPLY TO DISCUSSION.

Mr. G H. Blenkinsop *(Associate)* : The discussion on my paper has dealt largely with the temperatures in the mines and with the method which I used in calculating underground temperatures. Attention was called by some of the speakers to a paper read before this Society at the May meeting 1911, by Messrs. J. Whitehouse and W. L. Wotherspoon, and it appears to be the opinion of the said speakers that this paper settled the question of mine temperatures once for all. The temperatures given in Messrs. Whitehouse and Wotherspoon's paper are however lower than those returned to the Mines Department by the different mines on the Rand (see Table No. 30, in the annual report of the Mines Department for 1912). Some of the speakers evidently thought that as I had taken lateral distance into account in my calculations that I held the opinion that temperatures increased with lateral distance as well as with depth. I am surprised at this, for it proves that the correct method of taking temperatures underground is not generally understood.

I will here state a fact that must not be lost sight of in ascertaining the temperature of underground workings. A current of air passing over a moderately heated surface such as the rock face of underground workings absorbs heat at the rate of 5° for every 1,000 ft. traversed. This is when the velocity of the current is at the rate of 16 ft. per sec., and this rate is practically the velocity at which the air currents produced by fan ventilation travel. I wish to call attention to the fact that I mention moderately heated surfaces for air passed over highly heated surfaces, such as the stove of a blast furnace absorbs heat very rapidly. The figure of 5° for every 1,000 ft. has been established as the result of the careful tests made in England, Germany, France, Belgium and New Zealand. This being so, if the temperature of the atmosphere at the top of a down-cast shaft is 50° the temperature of the air current at the bottom of the said downcast shaft will be 67·5°, assuming that the shaft is 3,500 ft. deep, 3,500 at 5° per 1,000 equals 17·5°. This is the increase in the temperature of the current of air caused by its passage down a shaft 3,500 ft. deep. The rock temperature at the bottom of the said shaft may be 80° or 85°. Now if we trace the current of air from the bottom of the shaft along the levels to the working faces we shall find that it increases in temperature 5° for every 1,000 ft. traversed until the temperature of the air current and the rock temperature are approximately the same.

I trust it will soon be understood why I took the lateral distance into account when calculating the temperatures at the working faces.

Some few meetings ago a member read a paper in which he severely condemned the practical man. The practical man is sometimes useful. For example, practical experience would have shown Messrs. Whitehouse and Wotherspoon that temperatures taken at or near the bottom of a downcast shaft are very much lower than the actual mine temperature, *i.e.*, the temperatures at the working faces. Consequently their paper is misleading as far as mine temperatures are concerned. This is confirmed by the difference between the temperatures returned to the Mines Department by the Village Deep Mine and those returned by other mines of about equal depth which also have mechanical ventilation.

Mr. Walton stated in his remarks that the temperature at his mine, the Rose Deep, is 73°: that being so, one is curious to know why he has made a declaration to the Mines Department (see Table 30, already quoted), that the average temperature in the Rose Deep is 83°.

Before I leave this question of temperatures I would mention that Mr. Moynihan read a paper before the December meeting of this Society as his contribution to the discussion of my paper, in which he says "The author's main object is to lower the temperature in the working places of the Rand." I do not think there is anything in my paper from beginning to end which points to this conclusion, and I shall not, therefore, refer to his remarks.

It is stated in my paper that the object is to supply pure air to the working faces. Nothing is said about supplying air ot lower temperature. The reason for referring in my paper to the temperature in the mines is that medical experts are unanimous in saying that persons working in a temperature of 75° F. and above, have a lowered physical resistance to disease and are therefore particularly liable to be affected by small quantities of dust in the air and any other impurities which the air may contain. One speaker stated that fan ventilation removes dust laden air from the mine; that is so, but not until after it has been breathed by the workers.

It has been said that the system of ventilation which I advocate will stir up dust: this however does not happen, for the pipes conveying the air to a working face will have a diameter of say 2½ in., or say an area of 4·8 in. At the end of this pipe is attached a bell-shaped mouth piece having an outlet 9 in. in dia. or an area of say 63·6 in., consequently the pressure of say 8 to 12 oz. at the outlet of the 2½ in. pipe is so reduced at the outlet of the bell-shaped mouth piece that it is not sufficient to blow out a candle. The fresh air diffuses itself quietly over the working face, and there is no draught or rush of air. I state this from actual experience, for I have used this system of ventilation at two mines.

At the November meeting of this Society, Mr. H. C. Boydell read a paper, in which he stated that ventilation by blowers had been tried in Australia and had been proved to be a failure. He gave an instance where the quantity of air actually delivered at the faces amounted to only 13% of the capacity of the blowers. It is evident that this loss of air between the blowers and the faces must have been owing to perforated air pipes or faulty connections, and Mr. Boydell's account of this particular Australian experience is simply a record of incompetence. What would happen at a Rand mine if only 13% of the air from the compressors reached the working faces? As you all know, the pressure of air supplied to the machines on the Rand mines is 70 lb. and upwards, as against 8 to 12 oz. from a Roots' or similar blower.

Other points which have been mentioned during the discussion are the cost of the installation and the horse-power required. In my paper I state that the cost of an installation for a mine employing a total of 2,000 workers underground would be less than £10,000. This figure is based on actual experience of this class of work. For obvious reasons one cannot be expected to publicly furnish a complete specification. With reference to the horse-power required, some alarming figures have been mentioned during the discussion. The horse-power required for Roots' blowers capable of supplying 100,000 cu. ft. of air per min. at a pressure of 12 oz. + the h.p. required to overcome friction in the pipes is 600 to 650 as determined by the length of pipes.

It has been stated recently that spraying has removed the dust peril. Spraying has undoubtedly improved underground conditions, for, as stated in my paper, a large percentage of the dust produced by drilling, blasting and shovelling has been allayed by the carrying out of the regulations for the allaying of dust. The very fine dust, that is, the impalpable dust, cannot be allayed by spraying, and it is this impalpable dust which has such an injurious effect on persons whose physical resistance is lowered by the temperature of the air in which they work. In a current of air such as that produced by fan ventilation travelling at the rate of say 16 ft. per sec., it is impossible to allay the very fine or impalpable dust by spraying. I wish to call particular attention to this statement that spraying will not allay the very fine particles of dust whether the spraying is done by water alone or a mixture of water and molasses, even when the current of air is only travelling slowly, and I think a very little consideration will convince anyone that fine dust will not settle when the current of air is travelling at the rate of say 16 ft. per sec. I trust that no one will gather the impression that I advocate the erection of dust chambers underground I have given this information for the purpose of illustrating how difficult it is to bring about the settlement or deposition of very fine dust. The use of water drills will reduce the quantity of very fine dust produced, but fine dust will always be largely produced by blasting.

A feature of the system of ventilation which I advocate is that it ventilates the whole of the mine and not simply the main road ways. The fresh air liberated at the working faces diffuses itself throughout the whole mine and replaces an equal quantity of air laden with impalpable dust and other injurious impurities. If say, a total of 100,000 cu. ft. of fresh air per min. is liberated at the working faces an equal quantity of foul air must be forced out of the mine. Each shaft will act as an upcast shaft or outlet for the impure air. It is proposed to work the blowers night and day and also during the week end stoppage. The whole of the mine workings, stopes as well as drives, should in a very short time be as sweet as the proverbial nut if this system of ventilation is adopted.

In conclusion, I beg to thank those gentlemen who have taken part in the discussion, and I think we are all agreed that the subject of improved ventilation of the mines of the Rand is one of great importance, for improved ventilation means not only healthier conditions for the workers, it also means a substantial reduction in working costs. I think it will be found that the mines which have the lowest working costs are also the best ventilated.

THE DETERMINATION OF THE ACIDITY OR ALKALINITY OF WATERS—A STUDY IN INDICATORS.

(*Read at September Meeting, 1913.*)

By Jas. Moir, M.A., D.Sc. (Past-President)

REPLY TO DISCUSSION.

Dr. Jas. Moir (*Past-President*): I am rather disappointed that I have only induced one of our analysts to discuss the very important and fascinating subject of indicators. Mr. John Watson's discussion covers a lot of ground, so I shall select from it certain matters which seem of importance, and this time will refrain from treating Mr. Watson, who is (compared with myself) a sort of chemical Nestor, to any of those thistle-pricks* to which he so delicately refers in his introduction.

I fear that organic chemistry is not appreciated as it ought to be. I sent a copy of my paper to a literary friend in London, and his comment was: "What struck me as good about your paper was the note at the end—'the meeting then broke up'—that exactly expressed my feelings."

The most important point raised by Mr. Watson is: how are the results of titrations for acid to be stated? I have myself to confess to some slipshodness in having stated that the New Goch water contained 65 to 70 parts of *free* sulphuric acid, for part of this acid is not completely free, being bound to alumina and $Fe(OH)_3$. These two substances however are not bases in the strict sense, though they form salts: for when the salts are dissolved in water they are partially decomposed by it and re-act acid. I give some experimental data on this point later, but meanwhile am rather puzzled for a correct phraseology to express the facts. Perhaps I should have said that the Goch water requires 53 to 57 parts of caustic soda per 100,000 to make it truly neutral or equivalent to distilled water (53 soda = 65 sulphuric acid): or I might have said that it contained 65 to 70 parts of sulphuric acid, free and potential.

I am ready to admit that Mr. Watson and other practitioners who use methyl-orange (or litmus boiling) where corrosion of iron by "free" H_2SO_4 is in question, are, as a rule, correct within 5% to 10% of the truth, *i.e.*, if they add 10% to the methyl-orange results after titrating to first yellow (not salmon), their figures will be about right. Nevertheless, I strongly recommend the indicator butter-yellow (dimethylaminoazobenzene) for this corrosion-titration, as it is more correct theoretically and also considerably sharper than methyl-orange in presence of aluminium sulphate. Its yellow end-point corresponds to the H_2SO_4 uncombined with bases plus about 15% of the H_2SO_4 in $Al_2(SO_4)_3$, and occurs just about the stage where basic aluminium and ferric sulphates begin to appear. Even then the water is not absolutely neutral, but it is doubtful if the residual acidity is sufficient to affect iron. Direct experiments to compare the corrosive effect of equivalent quantities of ferric and aluminium sulphates with that of, say, $N/100$ sulphuric acid ought to be done, but I think that this lies strictly in the province of the School of Mines or the mining-group laboratories. I think it is safe to assume that the other salts present, viz., calcium, magnesium, ferrous, and sodium sulphates are all without action on iron.

Another point I wish to make is that the common practice of deducing the "free" sulphuric acid by calculating the bases found into sulphates and subtracting from the total sulphuric acid, is quite wrong, and gives results which are far too low. Such methods would be all right if $Fe_2(SO_4)_3$ and $Al_2(SO_4)_3$ were neutral

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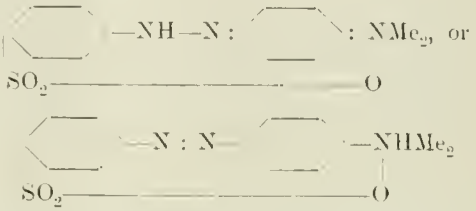
*See also this *Journal*, 1910, Presidential Address, last par.

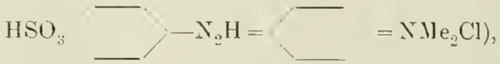
but in point of fact in such waters this method is worse than the methyl-orange titration.

It should be noted that my observations all refer to *cold* solutions, and that if boiling solutions are used, as is customary with litmus, all the indicators then resemble methyl-orange, because CO_2 is eliminated and basic salts are hydrolysed. Litmus may be the "beatified" indicator, but it is a nuisance that it has also to be boiled, and I personally never use it except for rough tests.

You will remember that I stated that neutral methylorange is really yellow, not salmon. I have since made an interesting experiment on the point. By treating a weighed quantity of methylorange (which is a sodium salt) with the calculated quantity of boiling acid to combine with the sodium, digesting, and cooling, the free compound was obtained (so-called dimethylaminoazobenzenesulphonic acid). It is deep red with a bright violet lustre (like coerulignone) and is quite sparingly soluble in water (about 1 in 4,500), giving a *yellow* colour—contrary to the statements of Ostwald and the orthodox ionists. It is not really an acid, but an internal salt like glycocoll, thus:



The salmon colour is only obtained if water containing acid equal to at least $N/5000$ is used to dissolve it (formation of



and this salmon coloured solution is *acid* to

TABLE A.

Indicator.	Original colour.	cc. NaOH to first change		cc. NaOH to definite alkalinity		Alkaline colour
		(*a*) Amm. alum.	(*b*) $Al_2(SO_4)_3$	(*a*) Amm. alum.	(*b*) $Al_2(SO_4)_3$	
1 Congo-red ...	purple brown	0·1	—	2 to 2·5 (lakes)	—	red
2 Butter-yellow ...	pinkish	0·20	0·20	0·5	0·6	yellow
3 Methyl-orange ...	,,	0·5(?)	0·30(?)	1·5 to 2 5	2 to 2·5 (useless)	,,
4 Litmus	red	2·5 (violet)	—	2·8 to 3·0	—	blue
5 Rosolic acid ...	yellow	2·6 after going back	—	2·70	—	pink
6 Alizarine ...	orange	lakes even while acid		2·7	—	purplish-red
7 Gallein	yellow	,,	(red)	no end point till excess		olive to purple
8 Methyl-red ...	pink	gradual fadi'g	about 2·0	2·45	2·35	(sharp) yellow
9 Lacmoid... ...	red	2·6 (violet)	—	2·70	—	,, blue
10 Nitrobenzeneazo-guaiacol	yellow	2·65	2·50	2 70	2·65	,, crimson
11 Nitro-oxyazobenzene	,,	2·75	—	2·80	—	,, pink
12 Orange I. ...	,,	2·80	—	2·90	—	,, ,,
13 Naphtholphthalein...	pale straw	2·75	2·65	2·85	2·75	,, blue-green
14 Phenacetoline ...	dark straw	lakes (useless)		—	—	pink
15 Meldola's dye* ...	orange	,,	,,	no end point		purple then blue lake
16 Hewitt's indicator†	,,	2·6	2·5	2·8	2·7	,,
17 Phenolphthalein ...	*nil*	2·95	2·75	3·0	2·75	(sharp) pink
18 Oxydiphenylphthalide	,,	2·8 (exc. dye)	—	uncertain		red-violet
19 Tetrabromphenolphthalein ...	,,	3 0 ,,	—	3·3 vague	—	violet
20 Thymolphthalein ...	,,	3·7	2 9	4·0	3·1	blue
21 Benzaurine ...	orange	3·2	—	4 to 5	—	purple
22 Orange II. ...	,,	3·0	—	5·0	—	pink

* 4-nitrobenzeneazo-α-naphthol

† Na-sulphonate of Meldola's

litmus, methyl-red, and all the other "weak" indicators, whilst it is alkaline to fluorescein and Tropaeoline OO, etc. (See Noyes, *Am. Chem. Soc.*, 1910, 815).

I have also made a special investigation of the different indicators with aluminium salts and caustic soda. A solution of 9·07 gm. of ammonia-alum per litre (*M*/50) and a solution of 6·307 gm. of hydrated aluminium sulphate* per litre (*M*/100 or *N*/50 as regards aluminium) were made, both being 0·06*N* as regards sodium if one neglects the ammonium in the alum.

10 cc. was titrated with *N*/5 NaOH, using different indicators, the theoretical quantity for complete precipitation of $Al(OH)_3$ being 3·0 cc., although precipitation of $AlOHSO_4$ occurs much earlier, generally at 0·3 cc. soda for alum and 0·8 for aluminium sulphate. Both aluminium solutions were acid to all the indicators except Tropaeoline OO.

Some valuable conclusions can be drawn from this work. Firstly, methyl-orange and Congo-red are nearly useless for aluminous mine-waters, as there is no real end-point: Butter-yellow is much sharper and gives an end-point corresponding to a *small* hydrolysis of aluminium sulphate, which is probably correct as regards corrosion of iron.

Secondly, for determination of *true* neutrality, the five indicators numbered (9) to (13) are the best, as indicated in the original paper from other evidence. In practice I titrate first with Orange I (or Methylred) and then with Naphtholphthalein, and average the results of the two titrations. If Mr. Watson is unwilling to experiment with these new indicators, he can get a fair approximation to the same result by averaging the results of titrations with lacmoid (to blue) and phenolphthalein, but as they will be comparatively wide apart, the average is less accurate. Results of titrations for *true* neutrality should be stated in terms of caustic soda or as "free and potential" sulphuric acid as I have already suggested. At the stage of true neutrality these aluminium solutions have about 95% of the alumina precipitated: this agrees with the statements of various workers on aluminates in caustic soda. Thus R. T. Thomson (*Chem. News*, 1883, 123 and 184) states that alumina, with boiling litmus or rosolic acid, counts as a base as only $\frac{1}{27}$ of what it should be, *i.e.*, 1 gm. is equivalent to $\frac{40 \times 6}{27 \times 102}$ gm. of NaOH; as if 96·3% of it were inert.

Thirdly, the end-point with phenolphthalein does not occur until all alumina has been precipitated and ammonia begins to be liberated from ammonia-alum; but this may be accidental, as the $Al_2(SO_4)_3$ behaves differently: I confess I cannot explain this result. On the other hand, thymolphthalein does not change definitely until some free NaOH is present, being almost unaffected by NH_3: the difference between the two aluminium solutions, due to ammonia, is very marked in this case. Finally, Orange II does not change until $NaAlO_2$ has been formed, $Al(OH)_3$ being an *acid* towards this substance. A separate experiment with Orange II in pure water showed that a concentration of *N*/800 caustic soda is necessary to change it.

I have also made some rough observations on the phthalein indicators with alkali, as I found that R. T. Thomson's figures (*l.c.*) make them out to be less sensitive than they really are:

TABLE B.

Indicator.	Concentration of NaOH giving first faint colour.	Concentration of Na_2CO_3 giving first faint colour.
Phenol-phthalein	*N*/25000	*M*/10000
Orthocresol- ,,	*N*/20000	*M*/8000
Thymol- ,,	*N*/12000	*M*/4500
Carvacrol- ,,	*N*/8000	*M*/3500

In titrating very dilute Na_2CO_3 with acids (as in air-tests), thymolphthalein was found to give results about 8% lower than phenolphthalein; it was also found that sodium bicarbonate is intermediate in reaction between these two, being acid to the former indicator and alkaline to the latter (*M*/500 strength $NaHCO_3$ used) but only about 0·30 cc. *N*/5 alkali or acid respectively per litre of $NaHCO_3$ solution is required to reach neutrality.

Cresolphthalein is an excellent indicator; its colour, a rather bluish pink, is visible in greater dilution than that of phenolphthalein, and its change is very sharp. Its absorption-band is in the yellow-green and is comparatively narrow. Carvacrol-phthalein is rather superior to thymolphthalein: its blue alkaline colour is free from the red dichroism which is so troublesome in the case of thymolphthalein in artificial light: these two indicators constitute an interesting case of isomers with very nearly the same properties.

The next point is water-softening by means of sodium carbonate. I have not much to say except that there are a lot of rather cryptic practical-man's processes in existence for doing the necessary estimations. I think, however, I am right in saying that if enough soda is added to just make the water alkaline to phenolphthalein, then we know that $CaSO_4$, $MgSO_4$, $Ca(HCO_3)_2$, and $Mg(HCO_3)_2$ are all decomposed, and the water cannot produce scale, though no doubt the use of lime in addition to soda is desirable in most cases to reach the same end more economically.

In regard to tap-water (Rand Water Board), the hardness is generally about 22 (as $CaCO_3$ per 100,000), and the CaO and MgO are about 5 and 4 parts respectively. Nevertheless its *true* alkalinity is very small, about 3 parts per 100,000 (as NaOH), owing to CO_2 being present in some excess. It is, therefore, not alkaline enough to be used for dissolving cyanide without loss by hydrolysis, and, in fact, this seems to be well-known in practice.

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* Contained 17·0% Al_2O_3 on ignition.

The following further experiments form a continuation of Table I. of the original paper, and enable some other indicators to be classified. 100 cc. tap-water taken.

TABLE C.

Indicator.	Reaction of water.	cc. $N/5$ acid to reach full acid stage.	cc. $N/5$ NaOH to reach full alkaline stage.
1. Nitrobenzeneazoguaiacol	Rather alkaline	0·40	0·10
2. Orange I.	Just alkaline	0·15	0·05
3. Gallein	Strongly alkaline	1·7	Indefinite to olive
4. Neutral red	Rather alkaline	0·3	nil
5. Phenol-phthalein ...	Just acid	nil	0·10
6. Cresol-phthalein ...	Just acid	nil	0·15
7. Thymol-phthalein ...	Quite acid	nil	0·5 to 0·6
8. Carvacrol-phthalein ...	Quite acid	nil	0·6 to 0·8
9. Tetrabromphenol-phthalein	Quite acid	nil	0·6 to 0·7
10. Meldola's dye ...	Quite acid	nil	0·5
11. Hewitt's indicator ...	Just acid	nil	0·20

Consequently, cresolphthalein is No. 41A, Hewitt's indicator 41B, carvacrolphthalein 46A, Meldola's dye 45A, and gallein 23A, in order of strength as in the table of the original paper. Gallein was placed as No. 28 from the *data* in the literature, but by experiment it seems to be about the same as methyl-red: also nitrobenzeneazoguaiacol should be farther up the list at, say, No 39A: it has too wide a range of colour-change to be placed definitely.

In conclusion, I wish to raise an important point, which is: that analysts generally use too little indicator. The text-books, of course, warn one against using too much; yet I find that the quantity of methyl-orange recommended for 100 cc. liquid is only 0·00007 gm., which is only $\frac{1}{25}$ of the quantity required to combine with 0·01 cc. $N/5$ acid. I think consequently that there must be extensive hydrolysis of the salt $C_{14}H_{16}N_3SO_3Cl$, leading to insensitiveness. The case of phenolphthalein is similar, as the quantity recommended is about $\frac{1}{12}$ of what is required to combine with 0·01 cc. $N/5$ NaOH. The concentration of $C_{20}H_{13}O_4Na$ in such a solution is about $N/120000$, and by theory hydrolysis should be nearly 100%, and the indications quite incorrect. By calculation I find that at least 3 drops of a $\frac{1}{2}$% phenolphthalein solution (or 15 drops of a 1 in 1,000 solution) are necessary to avoid this hydrolysis; and a similar quantity of methyl-orange if used in the same strength. This is rather revolutionary, but I am satisfied that the calculations are correct, and if the deductions are wrong, it must be the theory that is at fault: personally I believe in the theory, and think that enough indicator is not being used in practice.

Another practical point is that if much $FeSO_4$ is present in mine waters, on titration with NaOH, ferrous hydrate is formed, which is an extremely powerful reducing agent, and destroys all the indicators by hydrogenation. In such a case, more of the indicator should be added as the end point is approached—but the results are always unsatisfactory.

ADDENDUM.

New test for neutrality or otherwise of water.

By adding to the water a small quantity of solution of 2-5-dinitroquinol, its true reaction can be at once determined by the change of colour. Neutral water becomes orange-brown: faintly acid water at $N/10000$ becomes yellow orange: markedly acid water at $N/1000$ becomes greenish yellow: whilst faintly alkaline water becomes red-brown and definitely alkaline water becomes purple. For details see Henderson and Forbes, *Amer. Chem. Jl.*, 1910, 687. I have tried it and am satisfied it is a good thing. The commercial name (German) of the indicator is *Dinitrohydrochinon.*

The Chairman: I should like to thank Dr. Moir for what is really another paper on the subject. He has added considerably to our knowledge of what are the correct indicators to use.

THE UNION PATENTS BILL.

(Read at December Meeting, 1913.)

A. L. Spoor and W. E. John (Members).

DISCUSSION.

Mr. W. Cullen (*Past-President*): The authors of this most excellent paper deserve our thanks. I have heard it said that it is unsuitable for our Society, but I differ. It deals with matters in which we are all interested, and if papers of this sort are to be excluded from our Proceedings where are they going to be published? As the leading technical society, I think it is our bounden duty to receive a paper of this sort, and I go even further and say that if we, as a Society, think that any legislation which affects us—whether pending or existing—requires modification, then we should make our views known in the proper quarter. I hope, therefore, that our Council will bear this in mind. It is hardly likely that the Draft Bill will be introduced this session, but all the same we should be ready. It will not be surprising if few of our members take part in the discussion. After all patent law is a very dry subject, and most of us prefer to take advice from specialists, like the authors. I have therefore considerable hesitation in venturing to criticise, because I am not a specialist in any sense of the term. I must confess, however, to a good many appearances in the law courts, both as plaintiff and defendant, in connection with patents, and I have also fathered quite a few patents.

The authors have taken the Draft Union Patent Bill as their subject. This Bill was drafted by a very strong commission of judges, and I agree with the authors that on the whole they have done their work admirably. The draft is prefaced by a memorandum which gives a very succinct account of the reasons for recommending certain alterations to suit South African and Union conditions, and I commend this memorandum to anyone interested. It is gratifying to note that the Commission have followed the existing Transvaal Act, or Proclamation (as it really is) pretty closely, and in this connection it is somewhat remarkable that this Act, in this new country, has stood on the Statute Book practically without amendment for twelve years, nothwithstanding the fact that it was enacted while the war was still in progress.

The idea of having a central office is, to my mind, sound, and I cannot see what good purpose would be served by having a branch or Provincial offices. I have not got out the statistics, but I think I am safe in saying that 90% of the total South African patents emanate from the Transvaal. It is suggested in the memorandum already referred to that a reference register of patents could be kept at the Provincial offices, and that in cases of opposition witnesses could be examined locally by a deputy commissioner. The register might be examined once or twice a year, and witnesses probably as often, but surely this would not justify the establishment of branch offices, therefore I am all for the English system. But in any case, even to-day, there is not a Transvaal register of patents, and this surely is one of the first things awaiting the attention of the new organisation which it is proposed to set up.

Then the personality of the Commissioner of Patents is an important point. Mr. Bucknill, our late Transvaal Commissioner, was, to my mind, an ideal man for the post, and a similar appointment would give great satisfaction.

I cannot quite follow the authors on their proposal to delete "any foreign country" from the definition of inventions. These words, it is true, did not appear in the first Transvaal Proclamation, but they were inserted in an amending Proclamation, which appeared only a month later. Surely the very fact of a thing being known anywhere is a bar to the granting of a patent, because under these circumstances it really cannot be called an invention. However, the point is not a very important one at the moment.

The Bill very wisely does not provide for "examination" or "search." We are too young for a scheme of that sort yet, but it does open the portals of the "grounds of opposition" very wide. The authors object to (J) (December *Journal*, p. 299) in particular, but I think that both (G) and (I) are open to objection. Surely in return for the fees paid by the inventor, the Commissioner or the Patent Office can accept responsibility for seeing that (G) "the invention is not contrary to . . . . public order and good morals" (I) that the specification does disclose or explain the invention, and (J) that the title does truly indicate the subject matter of the invention. All these are elemental points of examination.

One of the most useful new provisions relates to surrender. In the Transvaal Act there was no surrender arrangement except the automatic one of the lapsing of a patent through the non-payment of fees. Before these fees become due, however, the patentee was liable to be attacked on any of the usual grounds, and no matter how willing he was to let the patent "go hang" he could not do so. On behalf of my company I was involved in a suit of this description quite recently, and the costs which were awarded against us were quite considerable. A patent, therefore, is not always an asset.

The question of compulsory working is a very thorny one, and most people are sentimentally in favour of it, as indeed I am myself, but we have to look to the practical aspect of the question. It will be very many years before South Africa becomes an industrial country in the sense that England is, and I think therefore that the provisions are much too drastic (pars. 77-80). If these provisions pass, few oversea patentees will have their patents very long in the official register. I notice also that the English patent agents, in connection with this same matter, have recently sounded a note of warning. Owing to the modern organisation of industrial enterprise it is becoming increasingly difficult for a genuine inventor to get his invention taken up, in fact few reap any reward at all. It seems rather hard, therefore, that par. 77 (1) "Any person interested may after the *expiration of two years* from the granting of a patent present a petition to the Commissioner alleging that the reasonable requirements of the public with respect to a patented invention have not been satisfied, and praying for the grant of a compulsory licence, or in the alternative for the revocation of the patent should be enforced." I believe with the authors that the enactment of the "compulsory" working provisions of the Bill will do more harm than good.

The little bit of special pleading under the heading of "Patent Agents" (December *Journal*, p. 300) will, I am sure, have the hearty support of all our members.

THE SAND-FILLING OF MINES.

(Read at September Meeting, 1913.)

By W. A. Caldecott (Past-President) and
O. P. Powell (Member).

DISCUSSION.

Mr. E. M. Weston *(Member)*: I should be glad to learn whether the experience of the authors shows that the boreholes used for sand-filling are practically indestructible as regards wear.

REPLY TO DISCUSSION.

Dr. W. A. Caldecott *(Past-President)*: In replying to the discussion on their paper the authors desire to thank those members who have taken part for the constructive criticism advanced, and for the data and information supplied as to sand-filling operations on various mines. Such information is of much value to members, but calls for acknowledgment rather a detailed reply.

Mr. Pam's remarks deserve close attention on account of his lengthy and varied experience in sand-filling, and his admission that the working costs on the Rand of the Silesian method are higher than those of the system described by the authors is the more gratifying in view of his somewhat different opinion of three years ago.* His view, however, that the low operating costs quoted by the authors are due to the good fortune of natural facilities on the mines referred to appears to confuse cause (*i.e.*, adaptation of system to local conditions) and effect (*i.e.*, economy), and is not borne out by Mr. E. E. Hardach's contribution, wherein the latter shows that the Wit. Deep sand-filling costs are even lower than the Simmer & Jack costs, the same system being employed in both cases. The fact, however, that careful consideration of local conditions has enabled a system of sand-filling to be evolved far more economical than the Silesian method originally introduced here by Mr. Pam in no way detracts from the credit due to his pioneer work.

The authors regret to be unable to describe the mixing bin at the Robinson Deep, as it does not exist. As an inspection of the accompanying photograph will show, the trucks are merely dumped into the open end of the tunnel, at a maximum rate of five per minute, and sluiced down as fast as dumped.

The authors cannot agree with Mr. Pam that the system employed at the Robinson Deep is very similar to the Silesian method, inasmuch as the lowering of sand being continuous instead of intermittent no storage bin is needed, a long tunnel and a deep borehole with free falling pulp is used instead of a pressure head in a pipe in the shaft involving the difficulties which Mr. Pam emphasises, mechanical surface transport of sand from vats to shaft is unnecessary, and very thick clean sand residue pulp is lowered.

If mines on the Rand had been originally started with sand-filling in view, and surface construction subordinated to truck conveyance of sand residue from the treatment vats to bins at the shaft, no doubt Mr Pam's advocacy of truck transport would be correct apart from the capital and operating cost, but unfortunately the prior existence of paths and roads, buildings, railway tracks and tram lines, cuttings and other obstructions render in most cases the installation of the usual double track for trucks with mechanical haulage from vats to shaft a practical impossibility, besides requiring a considerable amount of native labour, which might be more usefully employed elsewhere. In any case, as the sand after leaving the surface will be transported underground as a pulp, the obvious course is to

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*See this *Journal*, Nov. 1910, p. 202.

secure the facilities of the pulp form as soon as possible. It may be here noted in regard to sand transport that in dredging practice sand-pulp is pumped by centrifugal pumps on dredgers as far as 1950 feet upon adjacent low-lying land,* and that the immense wall of the Gatun dam at the Panama Canal was constructed by pumping sand to fill the space between two concrete walls as the cheapest and best form of transportation.†

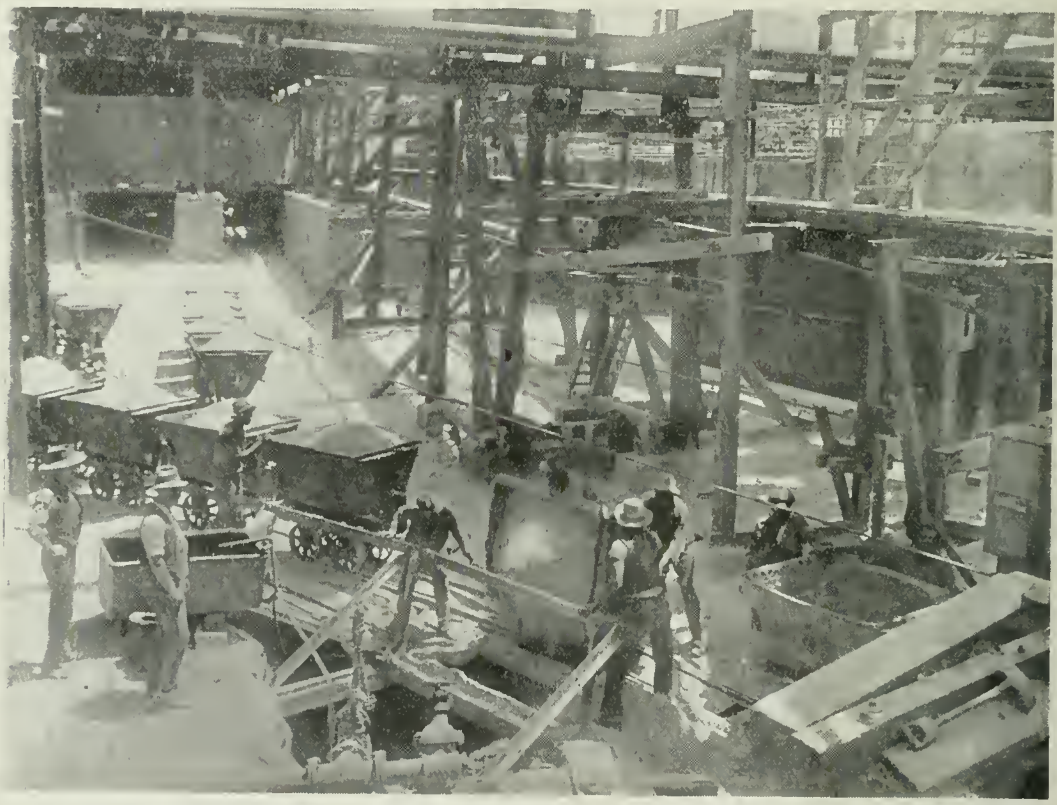

Illustration of surface disposal of sand residue for mine filling, showing truck transport of treated sand from the leaching vats in the back-ground. The trucks are dumped as shown into the mouth of the inclined tunnel and the sand sluiced as a thick pulp, with water and a solution of permanganate of potash as a cyanicide from the dissolving box in the left foreground, down the tunnel and borehole into the mine. (See this *Journal*, pp. 126 and 129, September, 1913). Robinson Deep G. M. Co., January, 1914.

The point may be emphasised that in sand-filling the cost of the definite operation of transporting sand residue from the treatment vats to the worked-out stopes underground is the crucial factor, and is not complicated as in mining with questions of stope widths, or as in ore treatment with considerations of percentage extraction.

The following are the tabulated costs of lowering 103,261 tons of sand on the Robinson Deep during the four months period from 1st October, 1913, to 31st January, 1914. During this time 105,290 tons of sand were treated so that 98·1% of the total production of sand residue was used for mine filling. It will be observed that the surface cost is only about one-half of that usually required for surface dumping, and about three-fourths of the underground cost is due to barrier construction.

Mr. Chilton's historical reminiscence of sand-filling with untreated sand a quarter of a century ago on the Wemmer Mine before the introduction of the cyanide process is of much interest, but it is to be feared that the miner of the future will not derive much benefit from the exploitation of the one-third dwt. sand residue used for filling nowadays.

Mr. H. A. White's suggestion as to the need for data of the launder grades required for different water ratios in pulp has been replied to by Mr. Sawyer, and his query regarding the

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*Encyc. Brit., 11th Ed., Vol. VIII, p. 566.

†Prof. Schwarz in *The Farmer*, 17th October, 1913.

	Surface Cost.		Underground Cost.		Total Cost.	
	Total.	Per ton of sand lowered in Pence.	Total.	Per ton of sand lowered in Pence.	Total.	Per ton of sand lowered in Pence.
White Labour	£140 11 4	0·327	£1,311 12 5	3·065	£1,459 3 9	3·392
Native Labour	488 1 8	1·134	793 4 8	1·843	1,281 6 4	2·977
Stores ...	71 14 7	1·167	1,088 18 6	2·531	1,160 13 1	2·698
Workshops ...	—	—	86 1 2	0·200	86 1 2	0·200
Power ...	66 9 3	0·154	—	—	66 9 3	0·154
Totals ...	£766 16 10	1·782	£3,286 16 9	7·639	£4,053 17 7	9·421

effect of $FeSO_4$ on testing for cyanide has been answered by Mr. McA. Johnston. As the originator of the use of permanganate as a cyanicide Mr. White's remarks upon its practical application are of much interest and value, as also his quotations from various authorities upon the effect of pressure on settled sand.

Mr. Sawyer confirms the value on the East Rand Proprietary Mines of the clean sand underflow from diaphragm cones lowered through a borehole for facilitating reclamation, and emphasises the advantage of rapid and efficient drainage underground, whilst the operating costs he gives are lower than the figures quoted by any other speaker.

Since the Cinderella Consolidated Mines system of sand-filling is no longer in operation owing to the closing down of the mine it does not appear necessary to discuss it in detail, but Mr. Veasey and other members interested will find the system already criticised in the columns of this *Journal*, p. 447, Vol. XIII., March, 1913.

The thanks of the authors and of members are due to Mr. Hardach for his detailed account with costs and diagrams of the successful operations on the Wit Deep*, involving the lowering of nearly one-third of a million tons of sand. The advantages secured are not stated in terms of cash, but apart from safety considerations the recovery of ore in pillars probably represents a very satisfactory return upon the capital expenditure and low operating costs. The confirmation of the statement in our paper of the negligible wear in pulp delivery pipes is of interest, and is probably due to the particles of sand being in suspension in the fluid medium, instead of rolling or sliding upon the bottom to a large extent as in the case of launders.

In reply to Mr. Weston's question regarding the wear of boreholes used for sand filling, we may say that one of the boreholes on the Simmer and Jack has been in use for nearly four years, and that no trouble whatever has been experienced from wear. With thick free falling pulp the cross sectional area of the stream in the borehole rapidly decreases as its velocity rises under the influence of gravity, so that it only comes in contact with the sides of the boreholes at distant intervals, where the borehole deviates from the vertical. Such deviation is of course less likely to occur with jumper-drilled than with rotary-drilled boreholes, but in any case any erosion of the borehole so as to increase its diameter by a few inches is of no importance, whereas with a pipe in a shaft perforation would long before have taken place.

In reply to the President's query we may say that the Robinson Deep borehole was put down by means of a steel-shot drill in nine months at a cost of £2,625, and thus cost a little over 30s. per foot. The Simmer and Jack boreholes were put down with jumper drills, which can be used to much greater depths than those required in this case, at the rate of eight feet per day at a cost of from 17s. to 20s. per foot.

PAPERS AND MEMBERSHIP.

The Chairman: I think our members should now provide us with some more papers for discussion, and I should also like to make an appeal in regard to the membership of the Society. No doubt many members know of other men who would like to become members, and I should like to ask you all to do your utmost in that direction. The members of the Society are not increasing in numbers as they ought to do, but I think we can do quite a lot in altering that. Also with regard to the attendance at meetings; that has been somewhat poor recently, and I think if we all took care to bring some other member that we should nearly double our attendance.

The meeting then terminated

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*See this *Journal*, January, 1914, p. 358.

VISIT TO THE CROWN MINES, LTD.

On March 28th, at the cordial invitation of the management, members of the Society, to the number of 150, visited the surface and underground works of the Crown Mines.

At the conclusion of the visit, members were hospitably entertained in the Company's large recreation hall.

The President congratulated the management on the efficiency of the lay-out, and proposed a hearty vote of thanks to the general manager, Mr. R. C. Warriner, and those of his lieutenants who had assisted for a highly instructive and enjoyable afternoon.

Mr. J. E. Thomas, Vice-President, seconded, and the vote was carried with acclamation.

Mr. R. C. Warriner replied on behalf of the management and Messrs. S. H. Peare and T. Simpson on behalf of the reduction and underground departments.

The underground party, consisting of about 60 members, met at No. 5 shaft and inspected the changing of an 8 ton rock skip for a 4 decker cage (carrying capacity 80 persons). A pair of these cages can be changed for skips or *vice versa* within 15 minutes. The party was then lowered to the 12th level. This is the first level in that shaft and the reef is struck by a cross-cut north. Proceeding along this cross-cut an underground drill sharpening shop and a waiting room for natives were noticed. The party then walked down No. 3 incline shaft to the 13th level. Here the grizzlies (on to which the 12 ton electrically drawn ore-wagons discharge) and the ore pass were examined. Below these grizzles are placed rock-breakers which deal with any unduly large lumps which might possibly choke the loading chutes. The party then entrained on an ore train, consisting of an electric locomotive working at 500 volts and drawing 5 to 8 cars and were taken in to one of the main ore chutes on the eastern side. An opportunity was given of seeing a set of cars loaded and the same train was taken back to the main pass. From here the electric pumping station was visited. The intallation consists of 2 sets of Sulzer turbo-pumps and 2 Gould three-throw pumps. Each turbo set consists of 7 vanes on the L.P. side and 8 vanes on the H.P. side with a 500 B.H.P. motor running at 1,480 r.p.m. in between. The actual gauge pressure is 1,000 lb. per sq. in., the Head being some 2,100 feet. The settling sump in connection with the pumps is designed to throw down a maximum of silt before the water passes into the large clear water sump. So much silt is there in the water that the settling sump has to be cleaned twice a week. This is done by hosing down and draining off into the rock skips.

On reaching the surface the winding engine room was visited. There are installed here two large Ward Leonard hoists and excavations are being made for the installation of a mammoth winder capable of 4,425 h.p. continuous delivery, and 7,000 h.p. every other minute. This winder, also of the Ward Leonard type, will operate the skips hoisting from the 19th level, a depth of some 3,500 ft.

After this the crusher station was visited. A noticeable feature here was the extraction of all dust from the crushers by means of a fan which exhausts on the outside of the building.

DESCRIPTION OF CROWN REEF REDUCTION PLANT.

The mill consists of 160 stamps and is in general construction similar to the Simmer Deep mill described in Vol. IX., 402-406.

The stamp weight, when new = 1,670 lb., dropping 8 in., 100 per min., with a duty of 16 tons per stamp per diem. Screen used, 9 mesh, 17 gauge, aperture = 0·276 in., 4 to 1 water feed is used. No amalgamating plates are provided here and the pulp leaving the mortar boxes flows over narrow apron plates by the main launder to the primary sand pumps. These are duplicate 12 in. Robeson Davidson pumps running 450 rev. per min. and driven by 100 h.p. motors. These only differ from the standard type in that the launder is connected low down in the hopper instead of the top—thereby gaining in fall and reserve space in case of stoppage. A lift of 45 ft. delivers the pulp to a distributing cone for supplying the tube mills.

There are 9 tube mills each 22 ft. × 5 ft. 6 in. and fitted with 5 ft. Schmidt feeders. Generally 6 tube mills deal with the pulp from the stamp mill and 3 deal with the oversize returned from the final classifiers. The tube mill classification consists of a 5 ft. 6 in. × 7 ft. 3 in. cone for each tube: the overflow of each pair of these passing an auxiliary cone 2 ft. 6 in. × 3 ft. 3 in., the underflow of which joins that of the bigger cone passing into the tube. The overflow is led to a distributing cone 4 ft. 6 in. × 3 ft. 6 in. which regulates the amount passing over the plates from any tube. The tube mills revolve at $30\frac{1}{2}$ per min. and are driven by 100 h.p. motors and the pebble load is regulated to take that amount of power, irrespective of the state of the liner. The rock pebbles are brought in 40 ton trucks to a bin at the end of the mill and fed into the tube mill hoppers by a mono-rail truck. From the hoppers a butterfly door and chute allows of rapid feeding—two natives are easily able to attend to the 9 mills, which take something over 8 tons per day each. Sectional "Osborne" liners are mostly used, the scrap being made up into peg blocks for ends or entire liners. The pulp feed to the mill is about 300 to 350 tons per day and the −90 mesh reground per day about 140 tons. The plate house, is an all iron building containing 63 stationary plates or 7 plates to each tube. The cement floor of this and that under the tube mills is graded to allow of being hosed down to drains leading to the secondary pumps. The plates taking the pulp from the first 6 tubes are adjusted to 13% fall and the others taking the returns require 18% by reason of a more pyritic pulp and less water ratio. This grade is adjustable by means of wedges to suit varying circumstances. The plates are scraped daily and otherwise dressed every 4 hours. The clean-up room adjoins the plate house and contains two small barrels 2 ft. × 2 ft. and two large barrels 4 ft. × 3 ft. These all discharge into a batea pan in the usual way. A pneumatic amalgam press is provided for the amalgam. The pulp from the plates flows over an amalgam trap at the end of each line of 7 plates and then over a very large final trap before elevation by the secondary pumps. These are similar to the primary pumps, except that they are provided with Dowsett regulators and have a lift of 75 ft. to the return cone classifiers. These consist of 10 cones 4 ft. × 3 ft. with 1 ft. of 10 in. pipe, the latter added to bottom to provide cylindrical classifying columns in which are provided Brazier hydraulic nozzles. The underflow from these are led by launders to the sand collectors and the overflow passes on to a return spitzkasten or rectangular box 24 ft. × 6 ft. × 5 ft. deep divided into 4 compartments. The underflow of this is led back to the secondary pumps and the overflow passes on to the slime collectors.

The Sand plant consists of 5 collectors 50 ft. × 10 ft. and 12 treatment vats 50 ft. × 12 ft. The collectors are provided with Butters and Mein distributors and overflow rims and hold a charge of 800 tons, 23·3 cu. ft. being taken for a ton. The sand is transferred by shovelling on to 2 conveyor belts under the vats, thence to a cross belt to the shuttle belt feeding the Blaisdell distributors on the treatment vats. The belt runs about 400 ft. per min. and carry 150 tons per hour. The treatment vats take a charge of 720 tons each and are given a 7 days treatment, or about 300 tons of 0·10% KC, followed by about 300 tons of weak (0·02%) and about 200 tons slime solution. In the slime plant there are 6 collectors 56 ft. × 10 ft × 4 ft. cones, taking a charge each of about 400 tons; this works out to about 31 cub. ft. per ton, but the actual tonnage is reckoned from sp. g. test from the agitators. The settled slime gravitates through 12 in. pipe to the agitators of which there are 2, each 32 ft. × 26 ft. with cone half way up the side. The charges average 240 tons sp. g. 1·36 (1·4 to 1) which are agitated by air, through central column for 4 hr. and then transferred to the Butters plant. In this plant there are 4 stock vats each 30 ft. × 20 ft. with 3 ft. cone. Three are used for pulp and 1 for wash solutions: they are provided with bevel driven four armed stirring gear revolving 6 rev. per min., in place of air agitation formerly used. Since the last visit of the Society, 120 leaves have been added to the plant, making now 420 leaves in all. The Gould vacuum pumps have been replaced by Rees Roturbo centrifugal vacuum pumps and a simpler method of leaf renewal adopted, otherwise the plant is practically the same.

The extractor house contains about 17 extractor boxes of the double type, each half containing 6 compartments 5 ft. × 2 ft. 2 in. × 2 ft. 1 in., or 20 cub. ft. each. They are constructed of steel with sloping bottoms and with side launder to facilitate cleaning up. Launders leading to a central clean up vat are provided similar to the City Deep plant. Other items consist in two acid treatment vats 10 ft. × 8 ft. with mechanical stirring gear, two 8 in, Worthington pumps throwing 4 tons solution per minute to sand treatment vats, two 30 in, centre feeling clean up presses, various pumps and one Betty zinc lathe with 40 sheet load and 3 tooth feed cut.

The old Crown Reef plant has all been utilised thus:—Part of the slime vats are used for stirring the excess slime sent from the Crown Deep plant for treatment in the Butters plant; the remainder acting as mill water return storage. Of the old sand plant, the original cement treatment vats are converted into solution sumps and the four upper vats formerly used as collectors are now used as slimes solution clarifiers, and it might be mentioned that the sand obtained from the classification plant described is sufficiently free from slime to utilise in the clarifiers, being brought thereto direct from the treatment vats. Even the old cement solution sumps in the former extractor house (now demolished) are being utilised as septic tanks in the new sanitation scheme.

The gradings of the pulp at the various points are:—

	× 60	× 90	− 90	− 200
Battery pulp	65	9	26	
Entering primary tube mills (moisture 30%)	85	9	6	
Leaving primary tube mills...	17	23	60	
Entering return circuit (moisture 35%) ...	49	40	11	
Leaving return circuit...	9	30	61	
Final pulp	2	12	86	
Sand residues	6	35	47	12
Slime residue	—	—	10	90

		Washed.	Last Drains.
	dwt.	dwt.	dwt.
Average sand residues (40% of pulp treated)	0·32	0·20	0·08
Average slime residues (60% of pulp treated)	0·15	0·10	0·12

Average combined residue from pulp treatment 0·210 dwt.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE DETERMINATION OF CHLORINE IN NATURAL WATERS.—"Determination of Cl by Mohr's method gives high results because too little K_2CrO_4 indicator is used, and consequently the titration is carried too far. Winkler's modification in which a diluted K_2CrO_4 solution is used and a correction for each determination is applied is accurate. Experiments show that it is unnecessary, however, if at least 1 cc. of 10% K_2CrO_4 is used per 100 cc. of sample and the first dark colour is taken as the end point. If Cl content is below 7·5 p. p. m., the sample must be concentrated. Fe waters are treated with ZnO and filtered, Cl being determined on the Fe-free filtrate. In Mn waters, the insoluble Mn is filtered off. Dissolved Mn has no effect. Acid waters are neutralised with MgO or $NaHCO_3$. Peptone, phenol, soap, albumin in concentration of 100 p. p. m. have no influence."—J. TILLMANS and O. HEUBLEIN (*Chem. Ztg.*) *37*, 901-3), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1913, p. 4025. (J. G.)

DETERMINATION OF SMALL AMOUNTS OF HYDROGEN SULPHIDE IN NATURAL WATER.—"A sample of 100 cc. is collected in a glass-stoppered flask in such a way that no H_2S is lost. A long-stemmed pipette serves to transfer to the bottom of the flask 5 cc. of the reagent (10 gm. $KNaC_4H_4O_6.4H_2O$, 10 gm. NH_4Cl, 0·1 gm. $Pb(C_2H_3O_2)_2$ and 5% NH_3 solution to 100 cc.). The colour developed on mixing is matched by adding 5 cc. of the reagent to 100 cc. of H_2O and running in Na_2S solution of known strength. The Na_2S solution is made up just before use by dissolving in $NaNO_3$ solution (50 gm. to 100 cc.) $Na_2S.9H_2O$ previously weighed out and preserved in a small tight container. If the sample contains over 1 mg. Fe per litre, 5 cc. concentrated $KNaC_4H_4O_6$ solution is added before the reagent. With coloured waters the reagent is added to a sample which is then shaken with air till the PbS is oxidized and this solution is used for the addition of Na_2S for the standard to compare with another sample of the water treated in the regular way. A content of 1·5—2 cc. H_2S per litre may be determined within about 10%."—W. WINKLER (*Z. anal. Chem.*, *52*, 641-5), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1913, p. 4026. (J. G.)

SEPARATION OF PALLADIUM FROM THE NOBLE METALS GOLD, PLATINUM, RHODIUM AND IRIDIUM. APPLICABILITY OF DIMETHYLGLYOXIME.—"In hot HCl solution dimethylglyoxime precipitates Au as metal and Pd as the yellow dimethylglyoxime compound permitting the separation of these two metals from the other metals of the Pt group, except Pt. *Au from Pd*: to slightly acid (HCl) solution of metals add excess of dimethylglyoxime dissolved in hot H_2O, heat for some time, cool, filter on a hardened paper, wash with 1% HCl, dry and ignite carefully, finally blasting. Dissolve in aqua regia and evaporate to dryness repeatedly with HCl, take up with warm H_2O, add 1—2 gm. of $(NH_4)_2C_2O_4$ and heat not above 60° for some hours; add 10—15 cc. dilute H_2SO_4, filter off Au on hardened paper and wash with dilute H_2SO_4 and then 1% HCl; dry, ignite and weigh Au. In filtrate neutralize excess of acid with NH_4OH and add dimethylglyoxime to boiling solution, cool, filter and wash with with warm H_2O; ash carefully, reduce in H, cool in CO_2 and weigh Pd. *Pt from Pd*: the Pt is first determined by precipitating with NH_4Cl; precipitate is finally reduced in H, cooled in CO_2 and Pt weighed, Pd is determined in filtrate as above. *Pd from Rh:* to the chloride solution of the metals add a few drops of HCl, heat to boiling and precipitate Pd as above. Evaporate filtrate to small volume and in a covered beaker add excess of HCl and successive portions of $NaClO_3$ to destroy the dimethylglyoxime, heat to expel Cl; dilute with H_2O and precipitate the Rh with Mg (solution becomes colourless). After all Mg is in solution, filter and wash with 5% H_2SO_4; ash, reduce in H, cool in CO_2 and weigh Rh. *Pd from Ir:* as in the separation of Pd from Rh. The dimethylglyoxime must be completely destroyed or the reduction with Mg will not take place. Results on known mixtures excellent (cf. *C.A.*, *7*, 1148, 1461)."—M. WUNDER and V. THÜRINGER (*Z. anal. Chem.*, *52*, 660-4), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1913, p. 3941. (J. G.)

PRESERVATION OF HYDROGEN PEROXIDE BY MEANS OF ACETANILIDE.—"The addition of 0·05% of acetanilide to hydrogen peroxide solution increases the stability to such an extent that only about 2·7% is decomposed in 5 months. The presence of mineral acids and of salts of the alkalis and alkaline-earths has no influence on the stability of the peroxide solution; traces of copper and iron increase the rate of decomposition."—A. M. CLOVER, *Journal Soc. of Chem. Ind.* (*Amer. J. Pharm.*), Jan. 15, 1914, p. 21. (J. W.)

DETECTION OF MOLYBDENUM.—"The detection of Mo by the blood red colour given by KCNS in a solution reduced by $CaCl_2$ or $Zn + H_2SO_4$ cannot be depended upon if too much $SnCl_2$ is added or if the $Zn + H_2SO_4$ is allowed to act for too long a time, owing to the great diminution in the intensity of the colour. The following modification gives dependable indications in doubtful cases. Reduce by either method, and then add the KCNS. Extract the Mo thiocyanate with ether which has been distilled over $SnCl_2$ to destroy any H_2O_2. Nevertheless, reduction of the Fe‴ should not be too greatly prolonged."—E. KEDESDY (*Mittkgl. Materialprüfungsamt.*, *31*, 173-6), *Chem. Abs.*, Vol. 7, No. *23*, Dec. 10, 1913, p. 3940. (J. G.)

DETERMINATION OF COBALT AND URANIUM IN STEEL.—"Modern high-speed tool steels may contain several % of Co. Using a 2 gm. sample, the W is separated as WO_3 and most of the Fe removed by Rothe's ether method; the Cr is then separated from the reduced solution by adding to the cooled and neutralized solution some 5—10% KCN. In the filtrate (half of which is taken) the Ni is separated from the CO by the well known KCN-Br method, precipitated by dimethylglyoxime and weighed as NiO. In another sample (1 gm.) the Fe, CO and Ni are deposited together electrolytically from $(NH_4)_2C_2O_4$ solution, the deposit is weighed, dissolved in diluted H_2SO_4. $NiSO_4$ is added to mask the Co pink, and the Fe is titrated with $KMnO_4$. The Co is determined by difference. To determine U a 2 gm. sample is dissolved in aqua regia, the WO_3 is filtered off, the HCl fumed off with H_2SO_4 and the Fe, Ni and Co removed by electrolysis in $(NH_4)_2C_2O_4$ solution. Electrolysis is continued till the oxalate is completely converted into carbonate, the solution is then boiled down from 300 to 100 cc. to destroy the $(NH_4)_2CO_3$ and thus ensure complete precipitation of the U. The precipitate, which contains traces of Co, V, Mn, Fe and Al in addition to all the U, is dissolved in diluted HCl, the small amount of CrO_4 is reduced by a few drops of alcohol, and the solution oxidized with HNO_3 and transferred to a small Erlenmeyer. To the cold weakly acid solution, $(NH_4)_2CO_3$ is added in excess, then $(NH_4)_2S$, the flask is corked and allowed to stand about 12 hr. U and V remain in solution. The filtrate is acidified with AcOH, boiled, the V removed by filtration, the filtrate oxidized with HNO_3, and the U precipitated by boiling with NH_4OH. The green-yellow precipitate of U hydroxide is filtered off, washed with NH_4Cl, ignited to U_3O_8 in a Pt crucible or to UO_2 in a Rose crucible in a stream of H. and weighed."—H. KONING (*Chem. Ztg.*, *37*, 1106.7), *Chem. Abstracts*, No. *23*, Vol. 7, Dec. 10, 1913, p. 3941. (J. G.)

THE VOLUMETRIC DETERMINATION OF CALCIUM. —D. has made a careful study of this determination by the direct and indirect or difference method. By precipitating the Ca as oxalate, dissolving the precipitate in dilute H_2SO_4 and titrating with 0·1 *N* $KMnO_4$ he obtained a series of results which were accurate and showed the reliability of the method. When the amount of Ca was less than 0·1 gm. the error was negligible. With 0·1—0·2 gm. Ca the maximum error was 0·5%. Another series of results showed that the difference method, *e.g.*, precipitating with an excess of standardized oxalate solution and titrating back the excess with 0·1 *N* $KMnO_4$ solution, gives accurate results only when the CaC_2O_4 precipitate is very thoroughly washed on the filter. An aliquot of filtrate and washings is taken for titration with $KMnO_4$ solution."—T. DORING (*Z. angew Chem. 26*, 478-80), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1913, p. 3940. (J. G.)

THE DETECTION OF SMALL QUANTITIES OF NITRATES IN WATER.—"The diphenylamine reaction is sensitive to less than 1 mg, $N_2O_5(N_2O_3)$. 1 gm. diphenylamine is dissolved in 100 cc. of purest H_2SO_4. 1 cc. of this solution is placed in a measuring cylinder, 1 drop of HCl (d. 1·19) is added and the whole made up to 100 cc. with purest H_2SO_4. The solution contains 0·01% diphenylamine. 20 cc. of the reagent and 10 cc. of sample of H_2O are mixed, forming a blue colour with the nitrate, or are stratified, forming a blue ring. The colour is very permanent."—S. ROTHENFUSSER (*Chem. Ztg.*, *37*, 897), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1914, p. 4026. (J. G.)

TITRATION OF CALCIUM AND MAGNESIUM IN THE SAME SOLUTION.—"The $MgNH_4AsO_4$ is precipitated by $(NH_4)_3AsO_4$ after the CaC_2O_4 has been thrown down, the two precipitates, are filtered off, washed, dissolved in H_2SO_4, the liberated $H_2C_2O_4$ titrated with $KMnO_4$, and the Mg determined in the same solution by adding KI and titrating the I liberated by the As_2O_5 with $Na_2S_2O_3$ (*J. Am. Chem. Soc.*, *21*, 746, and *29*, 1464). The titration of the Ca does not present any difficulty, but a correction must be made in the As titration for the I liberated by light. Starch and methylene blue are useless as indicators. The disappearance of the brown colour gives a sufficiently sensitive end point if a 0·2 *N* $Na_2S_2O_3$ is employed. The factors for the $KMnO_4$ and $Na_2S_2O_3$ should be determined by direct titration. F. used Iceland *spar* and specially prepared MgO. Twelve determinations showed good results for Ca, but for Mg the error is as great as 2 mg. MgO, and averages about 1 mg. irrespective of the absolute amount of MgO present."—PAUL J. FOX (*J. Ind. Eng. Chem.*, *5*, 910-3), *Chem. Abstracts*, Vol. 7, No. *23*, Dec. 10, 1913, p. 3942. (J. G.)

ASSAY OF GOLD AND SILVER BY IRON NAIL METHOD. "Mixtures of silica and iron pyrites, containing from 6% to 89% FeS_2, in addition to 18·1 oz. Au (as finely-divided metal) and 147·54 oz. Ag (as sulphide) per ton of 2,000 lb., were assayed in parallel by the iron nail, iron-nail and argol, and nitre method, ½ A. T. (14·58 gm.) of mixture being used in each case; and determinations made of the total and sulphide-sulphur present in the slags from the iron-nail charges. Except when the sulphide content of the charge was practically all oxidised by the litharge present, low results for silver were obtained by the nail methods; the loss of silver increased with the sulphide content, attaining a maximum of 10% with charges in which nails alone were used, and of 15% with those containing both nails and argol. The results for gold by the nail method, with and without argol, were irregular but generally about 1% low. Uniformly good results for both gold and silver were obtained by the nitre method, although as much as 30 gm. of nitre was necessary for some of the charges."—E. J. HALL and C. W. DRURY, *Journal Soc. Chem. Ind.* (*Eng. and Min. Jl.*), Jan. 15, 1914, p. 29. (J. W.)

METALLURGY.

SEPARATION OF BLAST-FURNACE DUST BY ELECTRICAL PRECIPITATION.—"Another method which possibly may be applied to the separation of dust from blast-furnace gas is the electrical precipitation of suspended particles. This system has not so far been applied to any great extent to the purification of blast-furnace gas, but is claimed to have been successful in other directions in removing suspended matter from gas and from air. An example of this is the separation of dust in cement plants. In this system the dust is deposited upon rods or wires electrically charged by high-tension direct current which is intermittently stopped to allow the collected dust to fall from the collecting rods or wires into a hopper below."—W. A. FORBES, *Iron and Coal Trades Review*, Nov. 21, 1913, p. 892. (A. R.)

TREATING TAILINGS DUMPS.—"I read with great interest the description of recovery of valuable material from cyanide-tailing dumps appearing in the *Engineering and Mining Journal* of March 22, 1913. As this information was new to me, I had the crusts of our dump assayed at once and found that they contained sufficient gold and silver to warrant re-treatment. I would appreciate a full description of methods.

[*Reply.*—The crust appearing on the surface of cyanide-tailing dumps is due to the evaporation of water from solutions containing dissolved gold and silver. Loss of water from the surface induces the rising of more solution through the capillary openings in the tailing, resulting in a concentration of gold and silver salts in the surface crust. These salts are, of course, soluble in water, and all that is necessary to recover the valuable materials is to dissolve, separate solution and solids and precipitate. When an operating mill is at hand, the recovered skimming is simply added to the material under treatment. Skimming the dump may be performed in several ways, depending upon the result desired. If a very rich product is desired for sale to a smelting or milling plant, an extremely thin layer should be taken up, confining the material recovered, as nearly as possible, to the pure surface crust. This can best be accomplished by sweeping the surface lightly with a broom, operating on a quiet day, so that the valuable dust may not be scattered by the wind. The dump surface, especially when it contains much slime, is very likely to be full of cracks through which most of the valuable dust may be lost. This difficulty is avoided by going over the surface to be swept and tamping the cracks with a shovel handle, or some similar instrument, closing the cracks and allowing the surface to be swept without losing the dust. Should an operating mill be at hand, it will very likely be desirable to take up more than the surface crust, for the upper half or threequarters of an inch will be impregnated with the valuable salts. In this case a square-pointed, long-handled shovel is used, taking up the surface for the required depth and forming little piles, which are later taken to the mill and treated along with the ordinary material. The depth which it is advisable to recover should be determined by experimental testing. After a dump surface has been thoroughly skimmed, the whole process may be repeated by flooding the surface with a small quantity of water and allowing it to dry off again The process is assisted by loosening the dump surface by plowing or spading, allowing the water to reach considerable depth. If running water is not convenient, rain will accomplish the desired result as well. Current tailing from a mill may be disposed with a view to future skimming, using new dump locations or depositing it over old dump which has been skimmed so often that further working is not profitable."]—*Engineering and Mining Journal*, Nov. 29, 1913. (A. R.)

STAMPS *v.* TUBE MILLS.—"In the annual report of the New Kleinfontein Mine for 1912, Consulting Engineer E. J. Way again returns to the question of stamps *v.* tube mills, and reiterates the opinion that for the Kleinfontein ore stamps are to be preferred to tube mills. Acting on the advice of E J. Way, the New Kleinfontein Mine was one of the last mines to introduce tube mills, and while practically every other mine on the Rand has declared in favour of tube mills, E. J. Way still maintains that stamps alone are more economical and efficient. It is explained that the ultimate decision to employ tube mills at Kleinfontein was entirely due to the fact that the site of the battery was unsuitable for erecting additional stamps, and not to any change of opinion as to the merits of tube mills. Four tube mills, centrifugal sands pump, Caldecott table, and additional slimes plant were started during May, and for comparison the periods selected are October, 1908, to March, 1909, and October to December, 1912. During the former period, says Mr. Way, the old plant was working at the normal capacity of the cyanide works, and during the latter period the enlarged plant with the four tube mills was in full commission, and also working normally. With regard to the reduction costs per ton milled, it should be noticed that the figures given for the original plant (4s. 2·216d.) includes an amount of 1s. for maintenance of all surface plants, which item was formerly kept separate in the costs. It therefore follows from the statement given below, that the original plant was comparatively more efficient in every way than the figures themselves show :—

	Reduction. Cost per ton.	Headings. Value.	Tailings. Value.	Percentage. Extraction.
Original plant (stamps) ...	4/2·216	8·648	0·364	95·79
Enlarged plant (stamp-tubes)	4/5·695	7·039	0·380	94·60

The above statement shows that the stamp and tube mill combination cost more per ton milled, and made a lower extraction, than the stamps alone; thus bearing out his contention with regard to the ore of this mine, that stamps alone are more efficient 'than a combination of stamps and tube mills.' It ought to be pointed out, however, that the manager in dealing with the additional reduction plant says: 'It is now possible, with 20 stamps hung up, to crush nearly 50,000 tons per month to a low degree of fineness, resulting in an increased extraction of 3 % compared with the beginning of the year.'

This question of tube mills *v.* stamps has also been taken up by Mr. Stadler, late of the Chamber of Mines Trial Committee, who supports Mr. Way, and maintains further that, generally speaking for the Rand, the cost of tube milling has not been always compensated for by the increased extraction of gold.

The metallurgical experts of the Consolidated Goldfields, on the other hand, however, are not only adopting tube mills as auxiliaries to stamps, which is the general practice on the Rand, but are actually displacing stamps by the use of additional tube mills. Quite recently at the Robinson Deep Mine the number of tube mills has been doubled, and brought up to 10, while out of the total equipment of 300 stamps, only 160 are now running, it being claimed that tube mills to this proportion are more economical and efficient than stamps. At the Simmer Deep, also, where 160 stamps and 9 tube mills are in use, it is understood that the monthly milling capacity is to be increased, not by the erection of further stamps, but by the addition of more tube mills.

According to the annual report of the New Kleinfontein Company, the ore reduction costs with 220 stamps and 4 tube mills only 4s. 5d. per ton, and milled 549,730 tons in 1912. At the Simmer Deep with 160 stamps and an average of 8 tube mills running, 594,650 tons was milled and treated at a cost of 3s. 5d. per ton. No figures are available as yet from the Robinson Deep, as the additional tube mills in place of stamps have not been running long enough to allow a fair comparison to be made. At the mines belonging to the Barnato group, however, the tube mills are being introduced with considerable success, and the general opinion on the Rand is that, as was the case with the Benoni process of treatment, Mr. Way has adopted a wrong course in delaying the introduction of tube mills so long at the New Kleinfontein."—R. GASCOYNE, *Queensland Government Mining Journal*, Dec. 15, 1913, p. 655. (H. A. W.)

ELECTROLYSIS OF AQUEOUS SOLUTIONS OF THE SIMPLE ALKALINE CYANIDES.—"The following conclusions were at, arrived after considerable experimenting, and presented at the twenty-fourth general meeting of the American Electrochemical Society at Denver, Colorado: (1) The bulk of the decomposition of cyanide occurring during electrolysis is due to oxygen liberated at the anode through the decomposition of water. (2) The principal final reaction in solutions of cyanide containing protective alkalinity, which in cyanide mill practice is usually due to lime, is the formation of the corresponding alkali carbonate. This accounts for the formation of large amounts of calcium carbonate during the electrolytic precipitation of gold and silver from cyanide solutions. (3) A consideration of the reactions previously discussed will explain the failure in many cases to obtain regeneration of cyanide when such regeneration is theoretically possible. It is apparent that the decomposition of cyanide may easily be equal to or greater than that regenerated through the precipitation of the metal in solution (4) The Liebig titration for the estimation of cyanide in an electrolyzed solution gives results which are low. (5) The changes taking place in fresh and electrolyzed solutions, upon standing, are different. (6) While not wishing to discourage investigators working along the lines of aiding extraction with refractory ores by electrolysis, it is of importance to call attention to the disadvantages which may arise through the indiscriminate use of electrolysis with an ore pulp. First, a large loss of cyanide may result from which there is no benefit derived, and, second, electrolysis may actually interfere with extraction through the formation of a coating of calcium carbonate upon the ore particles."—CLEVENGER V. HALL, *Mining and Scientific Press*, Dec. 13, 1913, p. 927. (A. McA. J.)

MINING.

INTERNATIONAL COMMISSION FOR UNIFICATION OF THE TESTING METHODS OF THE STABILITY OF EXPLOSIVES.—"The Commission recommends that for the transport of industrial explosives the packing should be made uniform, and that the following three tests, the first of which would eliminate at once explosives unfit for transport, should be applied. (1) Two samples warmed at 75 C for 48 hours must show no decomposition or alteration. (2) The powdered dry explosive must not explode in the falling weight test for an average fall weight in excess of that for picric acid and the sample must be at least as insensitive as picric acid when rubbed in an unglazed china mortar warmed to 20-30 C. (3) (*a*) Explosives taking fire in a glass tube from a fuse burning at the rate of 1 cm. per sec. are regarded as easily inflamable; (*b*) explosives passing (*a*) are poured in quantities increasing from 0·5 to 5 gm. into a red-hot iron capsule; (*c*) 100 gm. of explosive are tested on an asbestos sheet with an iron bar heated to 900° C; it must burn slowly, without explosion, and should go out on removing the bar. Explosives will fall into various classes according to their behaviour under these tests."—*Analyst*, Aug., 1913, p. 377, from Eighth International Congress of Applied Chemistry, Vol. 25, p. 305. (T. D.)

USE OF MULTIPLE DETONATORS.—"With long charges of high explosives in blasting work, it has sometimes been the custom to place detonators at intervals in the charges, in the belief that the work accomplished by the explosives would be increased. The results of an investigation on this subject are published in the U.S. Bureau of Mines Bull. No. 59 The *Engineering and Mining Journal* states that tests were made with insensitive dynamite, measuring the rate of detonation of different charges through which variously spaced detonators were distributed. The results indicate that extra detonators distributed 5 in. apart in the cartridge file of an insensitive explosive 40 in. long have a slight tendency to increase the propagation of the explosive wave, but that extra detonators spaced 10 in. apart offer no advantages. With an insensitive dynamite, such as that used, the influence of the detonator probably did not extend further than 5 in., and since the explosion wave and the detonation of the explosive surrounding the detonator probably precedes that of the detonator, it is evident that the latter would have no effect upon the detonation of the charge. It should be noted that these detonators were placed by themselves without fuses. It is, however, often advantageous to fire simultaneously two or more electric detonators placed in different parts of a long charge. Under such conditions, the time of detonation is reduced, and the shattering effect increased. In a 6-in. by 100-ft. churn-drill hole in a quarry, filled with explosive for 30 ft. from the bottom, it has been found that one electric detonator on top of the charge will not always produce a complete detonation throughout the charge, and two or more detonators are therefore often distributed. In such a charge, 30 ft. long, a violent effect is obtained by placing one electric detonator 5 ft. from the bottom, one 5 ft. from the top, and one in the centre of the charge. By this means the time consumed in the entire explosion is reduced to one-sixth and the force of the explosion greatly increased."—*Queensland Government Mining Journal*, Nov. 15, 1913, p. 577. (H. A. W.)

"ANNUAL REPORT OF THE CHIEF INSPECTOR OF EXPLOSIVES FOR THE UNION OF SOUTH AFRICA FOR 1912."—"There are, in South Africa, three large explosive factories, and, during the year three small 'tshisa' stick factories were started in the neighbourhood of Johannesburg. The quantities of explosives manufactured were 6,405 tons at the B.S.A. Explosives Co., Modderfontein, 10,554 tons at the Cape Explosive Works, Somerset West, and 5,931 tons at the works of Kynoch Ltd., Umbogintwini. There were no accidents of note at Somerset West and Umbogintwini, but two, each involving the loss of six lives at Modderfontein. In the Transvaal there were 214 magazines under licence. In many cases, owing to the exceptionally hot weather, the temperature rose beyond 95° F. (35 C.), laid down as the maximum permissible in the regulations. In the wood and iron magazine, required by former regulations, and constructed on the 'sub-surface' type, the temperature rose as high as 98° F. (36·7° C.), whilst the maximum attained in a stone magazine was not more than 82 F. (27·8° C.). The number of licensed magazines in Cape Colony was 190. The amounts of explosives manufactured, imported, and exported during the year were:—

	Manufactured.	Imported.	Exported.
Blasting gelatine tons	13,139	43	229
Gelignite ... ... ,,	5,341	62	258
Dynamite ... ... ,,	4,371	79	18
Blasting powder... ,,	—	30	—
"Permitted" explosives ... ... ,,	40	53	—
Total ... ... ,,	22,891	267	496
Detonators ... ... No.	—	49,415,000	72,000

Nitrocotton is imported by two of the factories, the Cape Explosives Works being the only one to manufacture its own. 19,886 cases of safety fuse (500 coils each) were imported from the following places: Great Britain (13,050), Germany (5,045), and Belgium (1,791). Fifty seven samples of safety fuse were tested during the year. Complaints of 'running' fuse have been of less frequent occurrence than formerly. 185 samples of explosives were dealt with in the Johannesburg laboratory. The chief items were: blasting gelatins (46), gelignites (39), and dynamites, dope (45). In the Durban laboratory 291 samples were examined, of which 261 were importations. At Cape Town 115 samples were dealt with, the chief items being: blasting gelatins (33), gelignites (31), and guhr dynamites (24)."—MAJOR A. B DENNE (Union Gov't. Report, 41, 1913), *Journal of the Society of Chemical Industry*, November 15, 1913, p. 1,033. (J. W.)

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Abstracts of Patent Applications.

(C.) 1/13. Emil Ullman. Improvements in bottom discharge trucks and the like. 2.1.13.

This application refers to hook shaped connecting rods operated by a central shaft which close the doors, and the position of the connecting rod when the door is closed is such that the tendency to revolve the shaft is at a minimum. The longitudinal shaft operating these connecting rods is in turn operated by a worm wheel and worm.

(C.) 85/13. Robert Lafford Towne (1), Cyrus Robinson (2). Improvements in and relating to processes for precipitating and separating metal from solutions. 14.2.13.

This application is for a method of using zinc-dust precipitation in which a spitzkasten is substituted for the customary filter press. The zinc dust is added to the solution in the usual way, and carried along a conduit of considerable length to effect precipitation, the end of the conduit being deeply immersed in the spitzkasten. Near the top of the spitzkasten is carried a tray in which is a bed of the finely divided precipitant which both acts as a filter to the uprising solution and as a precipitant of any metal left in solution. The spitzkasten is also provided with canvas filters at the overflow.

(C.) 592/13. Henry Squarebrigs Mackay. Process for extracting metals from low percentage ores. 11.12.13

This application relates to a process for the extraction of copper, which consists in the roasting of copper sulphide ores in such a manner as to leave a residual amount of sulphur as sulphate of copper (in the case of non-sulphide ores sufficient sulphur is added to the roasting furnace) to render portion of the copper soluble. The sulphate of copper is leached out, and the copper electrolytically precipitated, the resultant sulphuric acid solution being returned to the leaching vat to dissolve the oxides of copper formed during the roast.

Means also are provided for purifying the leaching solution with lime before precipitation

Alternatively hydrochloric acid may be formed by the addition of sodium chloride to the sulphate solution and the copper extracted as a chloride.

(C.) 593/13. Carl Davenport. Winding and haulage drums. 11.12.13.

This application refers to an arrangement of a hauling drum with the driving mechanism located inside the drum by which arrangement it is claimed that engine and gearing are perfectly protected from dirt and dust and that a constant lubrication is provided.

(C.) 614/13. Thomas Kemplay Irwin. Improved apparatus for the aeration and gasification of liquids. 27.12.13

This specification deals with an improved apparatus for the aeration and gasification of liquids, especially applicable in the treatment of waste liquids or sewage effluent.

The apparatus, which the inventor calls a hydro-extractor is devised with a cylindrical or other suitably shaped cage perforated or open where desired, and mounted on a central spindle and provided with a surrounding annulus of porous material, through which the liquids are filtered or aerated.

(C.) 2/14. James Miners Holman (1), John Leonard Holman (2). Improvements in or relating to ore concentrating tables. 2.1.14.

This specification describes an ore concentrating table having a reciprocal and circular motion and with discontinuous riffles, the separation of certain particles of ore being assisted by electric or magnetic means.

(C.) 3/14. Frederich Carl Wilhelm Timm. Process of treating materials, especially metalliferous materials, for wasting, reducing, volatilizing, sintering or the like purposes. 3.1.14.

This specification describes and claims a process for roasting, volatilizing, chloridizing, sulphurizing, reducing or sintering ores by means of heated gases and the heat produced by the internal combustion of heating materials in a furnace charge consisting of the materials to be treated, the temperature being kept below the temperature necessary for the complete melting of the materials. The solid heating materials, or materials containing solid heating materials are repeatedly added to the hot furnace charge, so that during the internal combustion only a portion of the total quantity of heating material necessary for the treatment of the furnace charge is present in the same. Any gold volatilized in the exit gases is cooled and collected in water or acid solutions. An illustration shows a horizontal rotary cylindrical furnace for carrying out the process.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 617/13. Anthony William Richard Schieder (1), William Spence Wilson (2), George Innes Smith (3). Improvements in apparatus forming receptacles for storing explosives, tools, and the like in underground workings of mines or the like. 27.12.13.

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Gilbert, T. W., *l/o* Roodepoort; c/o Robinson Randfontein, Randfontein.

Gullachsen, B. C., *l/o* Johannesburg; 10, Pilgrim Street, Newcastle-Tyne, England.

Hill, J. W., *l/o* Scotland; Abosso G. M. Co., Abosso, W. Africa, *via* Sekondi.

Hunter, C., *l/o* Swaziland; Inst. of Mining and Metallurgy, 1 Finsbury Circus, London, E.C.

Macfarlane, T. M. M., *l/o* Mexico; Santo Antonio do Juquia, Municipio de Iguape, Estado de S. Paulo, Brazil.

Mabes, W. H., *l/o* Germiston; P. O. Box 39, Randfontein.

McLennan, J., *l/o* Roodepoort; c/o Union Club, Johannesburg.

Newbery, J. W., *l/o* W. Africa: 13, Austin Friars, London, E.C.

Nisbeth, G. F. A., *l/o* Rhodesia; P. O. Box 6221, Johannesburg.

Roberts, E., *l/o* Brakpan; 121, Prairie Street, Rossettenville.

Sim, A. C. M., *to* 2 & 3, Walter Mansions, Eloff Street, Johannesburg.

Thomas, H. T., *l/o* England: Champion Reefs G. M. Co., Mysore State, S. India.

Warren, H., *l/o* Johannesburg; New Heriot G. M. Co., Ltd., P. O. Box 10, Cleveland.

Weiskopf, Dr. E., *l/o* Japan: Authors' Club, London.

Wiley, H., *l/o* Ireland; Eileen Alannah Mines, Ltd., Eiffel Flats, Gatooma, Rhodesia.

Wilson, J. H., *to* Knights Deep, Ltd., P. O. Box 143, Germiston.

THE JOURNAL
OF THE
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OF SOUTH AFRICA.

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Vol. XIV. APRIL, 1914. No. 10.

Proceedings
AT
Ordinary General Meeting, April 18th, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, April 18th, 1914, Mr. Alex. Richardson (President) in the chair. There were also present:

33 Members: Prof. G. H. Stanley, Messrs. J. E. Thomas, H. A. White, J. Gray, H. Meyer, C. Toombs, F. W. Watson, John Watson, A. Whitby, Prof. J. A. Wilkinson, A. F. Crosse, Dr. J. Moir, J. R. Williams (Members of Council), J. Q. Braidwood, J. Chilton, J. G. Dealy, T. N. Dewar, J. M. Dixon, R. C. Evans, H. W. Gill, C. J. Gray, B. J. Hastings, W. E. John, E. A. Osterloh, F. D. Phillips, E. Roberts, A. Malcolm Sim, A. L. Spoor, J. A. Taylor, A. Thomas, E. M. Weston, P. H. M. Whyte, and H. B. Williamson.

6 Associates and Students: Messrs. H. S. Ball, G. J. V. Clarence, J. Gibson, G. F. Mathews, E. J. Moynihan, and W. Waters.

9 Visitors, and Fred Rowland (Secretary).

MINUTES.

The Minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBERS.

Mr. J. Watson and Prof. G. H. Stanley were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman declared that the candidates for membership had been elected as follows:—

Gooch, Harry van Breda, Crown Mines, Ltd., P. O. Box 101, Crown Mines. Cyanider.

Nelson, Henry Wilson Stanley, Crown Mines, Ltd., P. O. Box 101, Crown Mines. Cyanider.

Paterson, George Ogilvie, New Goch Gold Mines, Ltd., P. O. Box 1096, Johannesburg. Mine Surveyor.

The Secretary announced that since the last meeting of the Society the following Associates and Students had been admitted by the Council:

As Associates—

Bourne, Frederick Charles, B.S A. Explosives Co., Ltd., 612, Salisbury House, London, E.C. Manager.

Harding, John Powell, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Chief Sampler.

Olsson, Carl Hjalmar, Cluny House, Grace Street, Jeppestown. Painter.

Wraith, Charles Osborn, P. O. Leenwpoort, Transvaal. Mine Surveyor.

As Students

Meyer, Theodor Carl August, S. A School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

Vieux, Jean Le, S. A. School of Mines and Technology, P. O. Box 3911, Johannesburg. Student.

GENERAL BUSINESS.

Mr. Andrew F. Crosse *(Past-President)*: I should like once again to say that we want farmers and landowners to send in to our Mining Exhibition some of the mineral products of the country, as, for instance, specimens of asbestos, graphite, mica, salt, fireclay, limestones, magnesite, oil-shales and base metals.

The President: As far as matters have gone at present in connection with the Mining Exhibition, it bids fair to be a great success this year. In addition to the various exhibits we are going to have a series of lectures, as follows:—Rescue Apparatus, by Dr. Irvine; Safety Measures in regard to Explosives, by Mr. W. Cullen; Safety Measures in Mining, by Mr. Archibald; Electric Blasting, by Mr. Girdler Brown; and Safety Measures in Shaft Sinking, by Mr. C. B. Saner.

The Secretary: You were notified last month that the Rand Mutual Assurance Company had offered a number of prizes for Safety First Devices. This month I have to advise you that the South African Red Cross

Society has presented a prize of £5 and a Bronze Medal for the best stretcher suitable for use underground, a prize of £5 for the best ambulance appliance of any description (excepting stretchers), and a bronze medal for the best first aid box exhibited.

Prof. G. H. Stanley *(Vice-President)*: I think before we pass on to the next business there is one matter that should be attended to and put formally on our Minutes. I refer to the visit we paid recently to the Crown Mines: I think we should pass a hearty vote of thanks to the Directors and Management of the Crown Mines, as well as to the officials who showed us round. They were all very kind and most hospitable; and I am sure we all had a most instructive and interesting visit. The very least we can do is to let them know how highly we appreciated it.

A METHOD OF ASSAYING CONCENTRATES AND BATTERY CHIPS FOR GOLD AND PLATINUM METALS.

(Read at February Meeting, 1914.)

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

DISCUSSION.

Mr. F. W. Watson *(Member of Council)*: When Mr. Crosse's name was put down on the Agenda for a paper on Assaying Concentrates, etc., for Platinum and Gold, I, and I am certain many other members, thought we were about to receive some new light on this problem, which previously had been the subject of a competition in this Society. I must confess I was disappointed on reading the author's description of his method, as it appeared to me that he had taken "Brevity is the soul of wit" for his motto when compiling his contribution, the motto being most strictly adhered to in presenting experimental results to back up his method of procedure. In my opinion the author does not show sufficient proof that cadmium will prevent solution of platinum in dilute nitric acid, in presence of an excess of silver. When questioned on this point by Mr. Rusden, Mr. Crosse gave as his reply that cadmium is one of the metals which precipitate platinum. Can Mr. Crosse tell us definitely if this precipitating effect takes place quantitatively in presence of an excess of nitric acid and silver? If we examine the action of dilute nitric acid on zinc—a metal very closely allied to cadmium—we find that an oxide of nitrogen (N_2O) is liberated so that the action of zinc in the presence of HNO_3 cannot be compared to that in which it acts as a reducing agent, owing to nascent hydrogen being evolved (as in presence of H_2SO_4) or when it precipitates a metal by simple displacement. Cadmium also dissolves in dilute nitric acid with evolution of oxides of nitrogen, it is less electro positive than zinc, and consequently its precipitating effect is weaker. The author says himself that "these points require a lot more experimenting on," and I think this experimenting would come most fittingly from the propounder of the method, and it should not be left to someone else to do the spade work. Further, in Mitchell's *Practical Assaying*, a process is described for extracting platinum metals in which zinc is used, and the bulk of the zinc is removed by HCl. It is stated that nitric acid cannot be employed to remove zinc from the residue as (1) "Part of the platinum metals will be dissolved or, at best, so suspended in the fluid that filtration is impossible." Will this difficulty not obtain with a metal so closely resembling zinc as cadmium? At any rate, more definite and fuller results should be shown before an assayer is asked to go to the trouble of fusing a silver-cum-gold, platinum, etc.—bead with cadmium under a layer of sodium cyanide. The next stage calling for remark is the fusing of the residue with acid sulphate of potash in a platinum dish. Does the author consider that platinum is so immune to the action of fused bisulphate that it is safe to fuse a residue, which is being assayed for this metal, with bisulphate in a platinum vessel? If so, then I regret I cannot agree with him for, from actual experience, I have found that fusing bisulphate in a platinum crucible has a marked effect on the crucible and reduces its weight appreciably. Here is an instance in which a little explanation and less brevity would have been advantageous. Why include such a risky and faulty process in the assay at all? Is it to remove ruthenium or rhodium? If so, I might remind the author that he has never found these metals in the filtrate after fusion, and consequently has never had actually to remove them. Perhaps the author will enlarge on this point in his reply. It seems to me that, in the method under discussion, the assayer is recommended to use a complicated process, and perform too much analytical gymnastics (to quote Professor Arnold's term), with very little benefit for the supplementary labour involved.

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(1) Mitchell's Manual of Practical Assaying 6th Ed., p. 787.

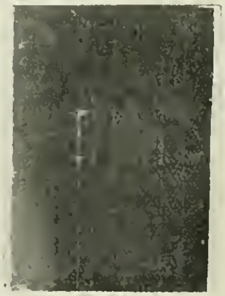
In precipitating gold from a solution of the chlorides of that metal and platinum, attention should be given to mass action. A reagent which precipitates gold and not platinum when the metallic chlorides are in separate solutions may precipitate some platinum along with the gold when the two are together. We have noticed this in precipitating gold in alkaline solution by means of hydrogen peroxide, and precautions have to be taken to purify the precipitated gold.

THE WITWATERSRAND EARTH TREMORS.

By H. E. Wood, M.Sc., F.R.Met.S. (of the Union Observatory).

Previously to the year 1908 it had been generally considered that South Africa, from a seismological point of view, was quite a stable part of the world, earthquakes being practically unknown phenomena over this region. Thus according to the late Professor Milne, writing in 1905:—" For ten years at least South Africa has not produced a single world-shaking earthquake. . . . At the present time the greatest seismic activity in the African continent is to be found in the high lands of Algeria and in the vicinity of the Great Lakes and Abysinnia. The origins of the displacements along the line of the Mozambique synclinal have apparently been too far from the African coast to be felt, but they have been recorded in very distant countries." Thus there are recognised centres of seismic activity in Northern Africa and off the East Coast, but South Africa itself was considered to be a very stable region.

In the year 1908 considerable interest was aroused in seismic questions in South Africa by the occurrence over the Witwatersrand of a continued series of local tremors or small earthquake shocks. None of these tremors was of sufficient intensity to do the least amount of damage to human life or property, but their comparatively frequent occurrence raised important questions as to their origin. The fact that they appeared to be localised on the Witwatersrand naturally suggested that they might be connected in some way with the extensive mining operations carried on there, but further investigation was necessary before any connection between the two could be established. Similar slight shocks were experienced for a series of years at a small village, Comrie, in Scotland, and in this case there were no mining operations in the neighbourhood. To obtain further evidence on the question of the origin of the tremors, earthquake records or seismographs were established at the Observatory and also by the Mines Department. These instruments have the great advantage that whatever evidence they give of an earth-disturbance is free from any personal factor. An earthquake or earth tremor is, as a rule, so sudden a phenomenon that probably no two people, standing in the same vicinity, would receive the same impression of it or would give similar accounts. What the seismograph does, when suitable precautions are taken, is to fix the actual time of occurrence and duration of any earth movement, indicate the nature and amplitude of the movement, and to give certain evidence as to the distance of the source of the disturbance. The Johannesburg seismographs at once showed that the local tremors were disturbances of the single wave type, that the amplitude of disturbance was extremely small, and that comparatively small masses of earth were concerned in the movement. To treat an actual case:—On January 30th, 1912, at 7.33 p.m., many people in Johannesburg felt two comparatively sharp shocks in succession. I felt these myself and made a note at the time that the stoep of my house seemed to ripple several times in a direction N.W. to S.E.



Record of Witwatersrand Local Tremor on January 30, 1912.

The seismograph record at the Observatory shows that there were three tremors within thirty seconds, of which tremors the first was the most severe. This tremor is represented by a sudden jerk of the recording needle from its equilibrium position, but there is no indication of any extended vibration having been set up. The instrument thus suggests that the tremor of which this is a type is caused by the sudden slight slip of a comparatively small mass of earth from one position of rest to another. This record of January 30th, 1912, was shortly followed, on February 20th, 1912, by a record of a very different type altogether. This record represents a true South African earthquake, the centre of which was situated without any doubt near the southern border of the Orange Free State, in the vicinity of Koffyfontein and Jagersfontein. It was of sufficient intensity to cause damage to buildings within an area of about 5,000 square miles and to be felt over an area of roughly 100,000 square miles. The Johannesburg seismographs indicated extensive oscillations of the earth which persisted for about twenty minutes. Yet this earthquake was noticed only by a comparatively few people in Johannesburg. The comparison of the record of a Witwatersrand local tremor with the record of a true South African earthquake, such as the one which occurred on February 20th, 1912, leads to the following conclusions:—

(i.) The origin of the local tremor is close at hand and probably on the Witwatersrand itself.

(ii.) The actual amount of earth movement is extremely small.

(iii.) A quite small mass of earth is concerned in the movement at the origin.

The following table will show that the frequency of these local tremors has been increasing during the past few years:—

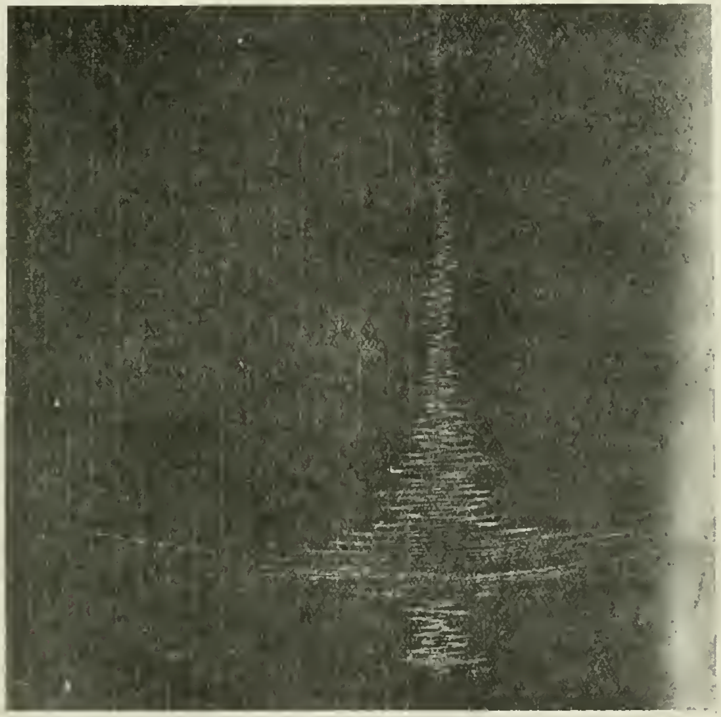
Year.	Number of Local Tremors felt.
1908 … … … … … … … …	7
1909 … … … … … … … …	3
1910 … … … … … … … …	12
1911 … … … … … … … …	14
1912 … … … … … … … …	38
1913 … … … … … … … …	82
1914 (to March) … … …	15

I am, therefore, led to the following conclusions:—That the Witwatersrand tremors are semi-artificial in origin, and that the ultimate cause of them is to be found in the extraction of large amounts of rock and water from comparatively small depths beneath the surface. I imagine that the combined effect of mining and water-pumping operations is to divide up the ground beneath the surface into small sections, which may be sub-divided further by existing faults. These small sections gradually become unstable, owing to the withdrawal of their supports, and suddenly slip by small amounts into positions of greater stability. This slipping communicates a slight jar to the surrounding ground, and the effect is radiated as a single wave of small amplitude. There is no evidence to show that this happens only in one particular region of the Witwatersrand, but the localities over which the shocks are felt are found more frequently along the Central and Eastern Rand than along the Western Rand. The existing records do not indicate that the local tremors are more frequent at any particular season of the year, so that it does not appear that their occurrence is facilitated by weather changes, etc. The length of the records is, however, too short to make any definite decision on this point.

We come now to the question whether the Witwatersrand tremors are similar to tremors which have occurred in other parts of the world. Between 1883 and 1885 small but distinctly sensible earth shocks were felt in portions of the town of Sunderland, England. They were accompanied by rumbling—sometimes dull, but often loud—by the rattling of crockery and furniture, and frequently by very distinct shakes of the entire framework of buildings. In a report on the origin of these tremors, Professor Lebour states (British Association Report, 1885) that the town of Sunderland stands upon the Permian Magnesian Sandstone, which is 300 to 400 feet thick beneath the town. This rock is riddled with cavities of every size and shape. Every thousand gallons of Sunderland water pumped up and ultimately discharged into the sea represents one pound of stone abstracted. It follows necessarily that the vaults of cavities must from time to time give way and, in collapsing, produce concussions accompanied by noise, but limited in the area over which their effects would be felt. In short, it seemed to Professor Lebour that in such natural stone falls at moderate depths a sufficient explanation of the Sunderland earth shocks was found.

In an issue of the Mining Journal, London, 1910, Dr. C. Davison discusses the relation between earth shakes or local tremors and true earthquakes. In an ordinary earthquake, he points out, the shock generally begins with a faint tremor which rapidly increases in strength until a series of strong vibrations is felt, and then dies away again in a tremor. An earth shake, as a rule, begins with one strong vibration, like that caused by blasting or the fall of a very heavy body, and this may be followed by a very brief tremor, such as would naturally follow either of these occurrences. Also the earth tremor lasts on the average for not more than two seconds. The Johannesburg seismograph records, as has been previously shown, give exactly this difference between a local tremor and a true earthquake. Such earth shakes or local tremors occur in mining districts in Great Britain, *e.g.*, in the Rhondda Valley. From the evidence collected, Dr. Davison concludes that the centres of the areas disturbed by the earth shakes lie close to, and on the downthrow side of known faults. By the withdrawal of the coal (or, it may be, by the lowering of the water-level by pumping in other parts of the mines) the rock above is deprived to a great extent of its support, and tends to sink down and close up the worked-out seam. Nowhere can this tendency be greater than where the rock is severed by a fault. Here the sinking would take place by small fault slips, each of which, by the friction resulting, would give rise to a rather strong shock. But as the slip would only affect a small region of the fault, and would occur at a slight depth, the intensity of the shock would fade away rapidly from the centre; the shock and sound would be of brief duration and would closely resemble those produced by the fall of a heavy mass of rock.

"If this be a correct explanation of the origin of the earth shakes—and it seems to account for all their peculiar features—it follows that the shakes cannot be classed either with true or with spurious earthquakes. They are of natural origin in so far as they are produced by fault-slips, but of artificial origin in that the slips are precipitated by human labour and not by the gradual cooling and warping of the earth."



Record of the Orange Free State Earthquake of February 20, 1912.

Some of the small earthquakes which have been felt in Germany are considered to be due to the falling in of the roof of enormous subterranean cavities formed by the long-continued solvent action of water on deposits of rock-salt, limestone and gypsum. Such causes, however, can have given rise to only very slight shocks, and must be quite subordinate to subterranean disturbances.

These reports on the origin of small local shocks in other parts of the world appear to confirm the conclusions at which I had arrived, namely, that the Witwatersrand earth tremors are due to the slipping of rock masses which have been rendered unstable by mining and water-pumping operations, and that the slipping is probably most frequent in the neighbourhood of existing fault-planes. In a paper contributed to the "Bulletin of the Seismological Society of America," Vol. III., September, 1913, I have dealt with the general question of the seismological state of South Africa, and have there expressed the opinion that, whereas on the whole, South Africa is extremely stable from a seismological point of view, there are, however, within it four distinct regions of instability:—the southern part of the Orange Free State along the Cape border, the Rustenburg district of the Transvaal, the Zoutpansberg district of the Transvaal, and the eastern slopes of the Drakensberg Mountains.

Mr. H. A. White *(Vice-President):* I should like to propose a cordial vote of thanks to Mr. Wood for this able and interesting paper. I think he has put it all very clearly before us, and has been rather modest in emphasising the conclusions he has come to. The paper is a result of the request made by the Council to Mr. Wood to give us his view on the matter, it having come to our knowledge that he had a lot of information on the subject which we felt should be put forward before the Witwatersrand generally. You will remember when we visited the Observatory not long ago we had the opportunity of seeing these instruments in action and seeing how very delicate they were.

Mr. Andrew F. Crosse *(Past-President):* I have much pleasure in seconding this motion. In the early days we never felt any earth tremors, but there have been a great many in recent years. Would the author inform us whether it is perfectly certain that known falls of rock have occurred at mines synchronising with earth tremors? That is an important point.

Prof. G. H. Stanley *(Vice-President):* I should like to know whether anything in the records obtained at Ophirton or the Observatory would lead Mr. Wood to think that one or the other was nearer the source of the earthquake?

Mr. H. E. Wood: I will deal first with Prof. Stanley's question. I take it the meaning of the question was that if the records at Ophirton and Johannesburg were compared with one another we might get some information as to the locality of the origin There has been no opportunity of doing that up to the present, but I hardly think it would lead to very much information. There might be a certain interval of time between the two tremors; that would be one thing that might be expected. Even that would not lead to very much information, because we most probably would not know the exact time at which the tremor itself occurred. If the exact instant at which the tremor occurred were known, and we got slight differences between the time recorded at Ophirton and Observatory it might be possible to say whether the Ophirton instrument was nearer the origin of the shock than the Observatory instrument, but unless the exact time of the shock at its origin were known it would be impossible to answer the question or derive any information from the two times.

Prof. G. H. Stanley *(Vice-President):* I have particularly in mind whether the amplitude in the two cases would give any indication.

Mr. H. E. Wood: The amplitude might be more likely to, but as we are concerned with such a small area, I think it would be difficult to derive any reliable conclusions from the two amplitudes.

Then as to Mr. Crosse's question. There were a few tremors reported previously to the year 1908, but they were only at the rate of one or two a year, and the information, although probably reliable, was not collated so well as it has been since then. Since that time there have been a great many tremors; and I am asked whether these tremors have been associated with falls of rock underground. There, again, I come to a matter about which I have not much information. I have only been informed of two cases in which the fall of rock underground was directly associated—that it occurred exactly at the same time as the tremor; but two cases out of the great number of tremors there have been form no evidence to go upon.

Mr. E. J. Moynihan *(Associate):* Does the seismograph give any indication as to the line along which the tremor is transmitted to the seismograph? If that is so, I would like to know whether the intersection of two lines from two instruments would not give practically the exact spot in plan underground at which the tremor occurred.

Mr. H. E. Wood: From a single seismograph you can get very little information as to the direction or bearings of the origin of a shock. In the case of the record of the South African earthquake, I have shown you the seismograph gives you certain information as to the distance of the origin, but no information as to the direction. The waves from an earthquake underground will travel up to the surface; they are refracted in passing through the solid earth; they are refracted in going from one stratum to the other; the direction may be changed by unknown causes, and when the wave passes from the wave underground to a surface wave it may again change its direction; consequently any direct evidence obtained from the seismograph is liable to be rather misleading than otherwise, as the direction in which the waves travel has been affected by the nature of the ground in which they have moved. If there were several seismographs placed in a circle round Johannesburg it might be possible to point to the direction of the origin. The Observatory seismograph picks out two components of the earth disturbance—the North and South component and the East and West component. If there were a shock lying in a line exactly north and south of the Observatory, either to the south or to the north, theoretically it might be recorded on only one component and not on the other component; but in most cases we find that the two needles are disturbed almost alike by a local tremor, so we have not been able to point to any particular direction, as viewed from the Observatory, in which these shocks occur; but from the records sent in by people it has seemed to me that the shocks are chiefly concentrated just to the east of Johannesburg—that is to say, the number of shocks reported from the Western Rand is considerably less than the number reported from the Eastern Rand, and most of the shocks seem to have their origin a mile or two to the east of Johannesburg proper.

The President: A paper on this subject is welcome, because the increasing frequency of noticeable earth tremors here is causing a slight disquietude in unscientific circles. It will be reassuring to many to know that these vibrations are merely advertisements of unstable conditions induced in limited rock masses by mining operations, and not premonitory hints of an approaching disaster. Although, as Mr. Wood points out, the settlement of insufficiently supported areas is encouraged by faults, the tremors we are discussing are by no means always referable to such planes of discontinuity. For instance, about six years ago numbers of residents on the mines near Cleveland railway station were wakened one Sunday night by a sharp shock. A serious collapse of some very large stoped area was feared, but an inspection failed to disclose anything of that nature; and it was only by a diligent search that the true origin of the shock was discovered. This was a crack, just wide enough to insert a match into, in the hanging wall of one of the stopes, at a depth of some 1,200 ft. below the surface, reaching from level to level, a distance of 300 ft. This sudden relieving of the strain, which was unassociated with any fault, and unaccompanied by any subsidence, was the cause of a severe, though purely local, tremor.

The Rand, should it ever be visited by a genuine earthquake, is not likely to suffer excessive damage. The underground workings of mines have rarely been disturbed in any way even by earthquakes which caused damage to the surface works; and the absence of deep alluvial soil from the town and suburbs is a very favourable feature as regards the resisting powers of Johannesburg's buildings to earthquake shock.

Mr. A. C. M. Sim *(Member)*: Earth tremors on the Witwatersrand concern all those who are interested in the industry here, and to my mind I have always felt that these earth tremors, which are sometimes attributed to blasting and sometimes reported from the Observatory as earthquakes, are due chiefly to another cause. As is well known, on the Village Main Reef Mine there is a line of fissure in the earth, which has caused an extraordinary fault through the three reefs on that mine, and unlike the ordinary faults which occur frequently along the Witwatersrand series. The fissure is very apparent in the Village Main Reef Mine, after leaving which it goes right through Lower Doornfontein, and through the Bezuidenhout Valley (south of Bedford Farm). That means that the whole of the Hospital series are resting on that fissure of the earth's crust, which fissure runs almost diagonally across the Reef and up the Valley. Therefore it is obvious from time to time, as the ground settles, the people who live on and north or north-west of the fissure will feel the shock, and people south-east of it do not feel it to any extent. Though Dr. Corstorphine makes little mention of this interesting occurrence in his book on the formations and geology of the Transvaal, I know that the fissure does run through the Reef, and I think that is the reason for nearly all these " earthquakes " of which we hear so much from time to time in the local papers.

THE UNION PATENTS BILL.

(Read at December Meeting, 1913.)

A. L. Spoor and W. E. John (Members).

DISCUSSION.

Prof. J. A. Wilkinson *(Member of Council):* We owe a debt of gratitude to the authors of this paper for having taken the trouble to give us the results of their minute dissection of the new Bill relating to Patents, and to those members of our Society who feel that at some time or other they may require to take out a patent, their paper will repay careful study. The opinion recently expressed that such a paper should find no place in our proceedings was rather startling, and I agree with the last critic in his remarks anent this. By way of endorsing what he said, I may remind members that this is not the first time that this Society has placed on record its views concerning matters relating to patents and their official administration. In February, 1907, at the instance of the Attorney-General of the Transvaal an Inquiry Committee was formed to investigate

1. Whether the system of administration of the Transvaal Patent Office should be continued in its present form; and, if not, what changes is it desirable to introduce?
2. What amendments, if any, are required in the present Patent and Trade Marks Laws?
3. The advisability of the introduction of a law relating to designs.

The reply given by the Society to these points will be found in the Annual Report of the Council for the year 1906-7 (Journal Vol. VII., pp. 397, 400-402). Many of the recommendations then approved have made their reappearance lately, and so I do not intend to traverse that ground again. It may, however, be of some interest to state that at that time the then Transvaal Institute of Mechanical Engineers sent in a similar petition, the recommendations submitted being practically identical with our own. Unfortunately these proved of no avail. The Commissionership was abolished and the work relegated to the Deeds Office. No publications have been issued, and searches are difficult and expensive. This method of administration it is proposed to change in some respects by the new Act. This Bill, we are told, was drafted by a Commission, and a glance at the names of its members is sufficient to show that, legally speaking, the Act is all that could be desired, but from a scientific or technical point of view it may be possible to advance some criticism. *Nolens volens*, such a Commission must consider every question from the legal point of view, and with all due respect to these gentlemen, I am of opinion that scientific and technical representatives should have found a place.

It is worthy of note here that the recommendation of the Commission for the post of chief administrative officer is the appointment of a man versed in both science and the law. This, as the authors say, is one which will meet with hearty approval, the more so here in the Transvaal, because we have had experience of both methods, namely, with and without such an officer. In the former case the Commissioner held his own Court and his verdict was final. The expenses were small compared to those incurred after the abolition of the office, when cases were tried before the Supreme Court and the parties to the action were compelled to be represented by Counsel. In this respect, therefore, the re-creation of this office for the Union marks a considerable step in advance, and one that should be welcomed by all classes of the community. The new officer will preside at a central office, and there is no more suitable position for this than the administrative capital, in fact, this should be specifically stated in the Bill itself.

There are also to be sub-offices. The authors of the paper do not state their views on this matter very explicitly. I agree with them, however, in the view that it is impossible to see what value, commensurate with the expenses of their upkeep, is to accrue to the community. Furthermore, there is more than a trace of provincialism in such a recommendation. If we consider for a moment the towns in the Union where the establishment of these sub-offices might possibly serve a good purpose, we cannot get rid of the notion that Johannesburg should be considered first of all, merely because the greater number of patents taken out have their origin here. I am of opinion, however, that the money could be utilised in a manner which would benefit the whole community in a far higher degree, and the proposal put forward is that in place of these sub-offices, the central office should print separate sheets of a definite size of every complete specification with drawings upon which the grant of a full patent is made, and that these should be distributed gratis to every public library throughout the country, with full directions as to how they should be filed for reference. Also a small number of extra copies should be kept at the central office at Pretoria for sale at a small sum to that section of the general public who may be interested in any particular invention. For reprints of these it would be necessary to charge still further in order to cover the cost of the extra expenses thus necessarily entailed, but even then the cost need not be excessive, and it would certainly not be the expensive process it is at present.

In addition to this, I would very strongly advocate the complete indexing of the patents already issued in the various Provinces, and their transference to the central office at Pretoria, which would be their future home. These indexes would, after completion, be published and distributed as above to public libraries. The method I would propose in order to deal with this in a proper manner would be to secure the services of an expert from the Patent Office, London, for a short period of, say, two or three years, who would be able to carry out the work with that experience and ability necessary for its satisfactory execution. I feel sure that the Government could secure the services of such an official on application to the British Government, who would be willing to second him for this purpose so that there would be no break in his period of service. During the post-war administration such cases have been of frequent occurrence, and the benefits derived by a comparatively new country such as this from such services are great. If we consider inventions from the point of view of the inventor alone, then patents may be looked upon as a monopolistic grant by the State to the patentee, who reaps the reward of his talent by exploiting the public in his own interests. But under modern social conditions such a view is untenable. The grant of a patent by the State nowadays allows the patentee the right of exploitation for a term of years, but the reward for this involves that the addition which has thus been made to the sum of the world's knowledge should be available to the community, in order that the wheels of progress may be assisted forward step by step and evolution thereby quickened and stimulated. According to the present Bill, the one privilege is granted, namely, to the patentee, but the other privilege is denied, the rights of the community being neglected, and, hence, the Act becomes a piece of class legislation for the individual rather than for the welfare of the State as a whole. There is no necessity to labour this point, but it is one to which legislators should attend if they are to be deemed worthy of the trust reposed in them by the community. There is one other feature of the efficacy of patent publications, also affecting directly the common weal, namely, that exchanges with all other countries can by this means be legitimately effected. This adds enormously to the efficiency of any Patent Office library, and, hence, should not be overlooked in any scheme of administration brought forward. The eve of consolidation of the various Provinces is the opening of the most important chapter in the history of this country, and at such a time surely no stone should be left unturned by every section of the people to lend their aid in that pioneer work which is to be the corner stone of future progress. From this point of view alone the necessity for publications of this character should be deemed a *sine quâ non*.

This brings me to the use of the words "or in any foreign country" in the definition of invention given in the paper. These words should either be deleted altogether or amended in some manner so as to meet the objections raised by the authors, with which I find myself in full agreement. Under the above scheme of proposed publications it will be obvious that publications of foreign patents will find their way to the central office, and in such cases I would deem that to be sufficient ground for the refusal of a grant to a patent to any one except the inventor. If, however, no patents had been granted in a foreign land for a specific process, and some one with a knowledge of that desired to take out a patent here, then a patent should be granted, provided that it could be proved that that knowledge had not reached this country by other means before the date of the filing of the specification. As the wording stands at present, instances will readily present themselves where serious injury might be inflicted upon intending patentees, and, if the possibility is there, the law should be amended.

With regard to compulsory working my views have undergone a change since 1907. At that time I was strongly in its favour, but on more mature reflection I feel that the time is not yet ripe when such a clause could be inserted. It will be conceded by almost all people that industrial activity, excluding mining, is at yet in its infancy in this country, and also that the means for its prosecution must be sought in older countries. The effect which this clause would have if inserted in the Bill would be to prevent the granting of patents in these industries, and hence the country would be all the poorer on account of this lack of knowledge should any of these industries at any future date be worked here. On the other hand, some means should be devised to meet the objections raised by Mr. White to prevent the blocking of local industry by patentees who have no intention of using the rights granted them by the State here.

The President: I am sure we are much obliged to Prof. Wilkinson for bringing this matter before us.

On the proposal of Prof. Wilkinson, seconded by Mr. J. R. Williams, the following Committee was appointed to act, in conjunction with the representatives of other technical societies, to make representations on the subject to the Minister of Justice:—The President, Mr. W. Cullen, Mr. J. Q. Braidwood, and Prof. Wilkinson.

VACUUM FILTRATION AT THE VICTORIA MILL OF THE WAIHI GOLD MINING CO., LTD., NEW ZEALAND.

(Read at May Meeting, 1913).

By William Macdonald, A.R.S.M. (Associate).

REPLY TO DISCUSSION.

Mr. Wm. Macdonald *(Associate):* At the annual dinner of the Society, held some three years ago, the President at that time specially requested some member to come forward with a paper on the Butters' process as practised on the Rand. After the lapse of some time, during which no paper on this subject was forthcoming, I essayed to write the one which has been placed before you, describing the work done by the rival process at Waihi. I had hoped, in submitting this paper, that someone would be tempted to demolish both the author and his paper, and, incidentally, bring forward some facts and figures which would enable a comparison to be made as between the merits of the two processes, more especially as regards metallurgical results as shown by the losses in dissolved gold. I fully described the character of the slime dealt with at Waihi, and the conditions under which it was produced, and all the data necessary for an interesting discussion were available if our friends on the Rand who have had experience with the Butters' process had cared to come forward with similar information. I have been away from Waihi for about a year, and regret that I am not able to furnish some of the information asked for, and in other cases must rely on memory.

In reply to Mr. J. E. Thomas, the duty per stamp is 5¼ tons. I did not give this as it did not appear to have any essential bearing on the important matter of the paper. The duty may appear to be low when compared with South African figures, but the ore is very compact and the gold very finely distributed throughout. It has not been found advisable to crush with a coarser mesh than 10, and, as a matter of fact, a mixture of 15, 20 and 25 mesh is used in order to overcome certain difficulties with regard to fall of pulp.

Mr. A. Salkinson refers to the fact that only 20% of sand and concentrate is made, as compared with a minimum of 40% sand on the Rand. This, of course, is a result of the finer distribution of gold in the ore, necessitating finer grinding in order to obtain the maximum extraction. Whereas South African practice may be said to be "all through 90," Waihi practice aims at 90% through 200.

Mr. J. M. Neill refers to the "high residue value of both gold and silver." This is no doubt higher than one is used to in South Africa, but the head value is also much higher. In the case of Waihi slime, the average head value is about 6 dwt., and the extraction of gold well over 90%. The percentage extraction could be increased by still finer grinding; but there is a point beyond which the increase in cost of grinding would not be covered by the increased recovery. Increased extraction in the case of the silver seems to be more a question of strength of cyanide and time of agitation than finer grinding, but here, again, an increased recovery can only be got at too great a cost, or at too great a risk of overstepping the line of *profitable* extraction.

Mr. Neill considers that the 1% of pyrite in the final slime product may be the "fly in the ointment." This point was carefully investigated. Separation of the mineral matter in the slime residue showed that it was responsible for throwing up the value of gold to the extent of about 4 gr. per ton. Experimental work with canvas and concrete tables was subsequently carried out on a working scale with the object of separating the mineral from the slime. The result of these experiments led to the conclusion that the increased recovery did not warrant the outlay of capital required for a complete slime concentration plant, plus the cost of further grinding and treating the concentrate. Mr. Neill cites a case of "premature precipitation" of gold, but I fail to see any analogy between this and the case under review.

I now turn to Mr. Thos. B. Stevens' contribution, which is very much to the point, as it deals with essential features of a vacuum filtration process, besides providing us with some interesting additional information. Mr. Stevens makes reference to the fact that the moisture in the filter cakes from the "thickening plant" is 35 % to 40 %, as against 30 % in the final residue cakes from the "washing plant." This difference is accounted for by the fact that two different designs of plants are in use. The "thickening plant," which is of novel design, is described by Mr. Charles A. Banks in the "Mining and Scientific Press" for July 26th, 1913. The separation of the water is not so complete in this plant as is the separation of the washing solution in the "washing plant," but the cost of thickening by the method adopted is so low as to more than compensate for the extra 5 % to 10 % of water left with the slime. As pointed out by Mr. Stevens, this excess water has afterwards to be got rid of in the form of waste solution. It makes its appearance as an excess of solution drawn off from the cake over what is required for thinning down the pulp to the requisite consistency for agitation. After precipitation, this excess of solution mixes with the washing solution, and after fulfilling its function of washing the cake, again passes to the precipitation boxes before running to waste. By this time its value in both gold and cyanide is very low, and is the same as the average given for the washing solution in Table "D" of the paper.

The loss in dissolved gold in solution run to waste is accounted for in the summarised return as a separate item, a daily record being kept of its tonnage and value. The tonnage is theoretically equal to the amount of excess water brought into the plant, and as this does not exceed 10 % of the tonnage of slime treated, the amount by which this loss would raise the residue slime value per ton would be a very small fraction of a grain, and for practicable purposes negligible.

Mr. Stevens would seem to think that the low value solution used for washing the cakes is the solution used for making up the pulp from the "thickening plant." The solution used for this purpose is the precipitated solution drawn off from the cake whilst forming, and is higher in both gold and silver than the washing solution. Under favourable circumstances Waihi slime can be thickened by settlement down to 1 : 1, but it would not be possible to thicken the whole of the daily output of slime down to this point with the present capacity in settling vats, and taking into account the low cost of thickening by vacuum, the alternative suggested by Mr. Stevens of thickening by settlement only and running more spent solution to waste would certainly not be profitable. With regard to the material between the cloths on the filter frames, the corrugated steel has the advantage of being easily cleaned when renewing cloths, and under normal working conditions no trouble is experienced in getting rid of the cake. The cakes are air dried and the frames emptied of solution by the time the cage reaches the discharge hopper. Consequently, no solution comes away from the frame with the cakes as noted by Mr. Stevens.

Concluding, I wish to thank the gentlemen I have named for their contributions to the discussion, and more especially for their generous remarks of appreciation.

ELECTRIC BLASTING.

(Read at October Meeting, 1913.)

By Wm. Cullen, (Past-President), Thos. Donaldson (Member of Council), and W. Waters (Associate).

REPLY TO DISCUSSION.

Mr. W. Waters *(Associate)*: The discussion on our paper on "Electric Blasting" has not been quite so energetic as we anticipated, and we are somewhat disappointed that a large number of leading mining men have not come forward to take part in it. Nevertheless, we thank all those who offered criticisms, including Mr. Weston, who, however, did not seem to be too full of the milk of human kindness. It is only fair to ourselves to state that the paper is the result of experiments and observations carried out over a number of years, and that for a few months prior to its being written, an average of six competent qualified scientific observers were constantly at work on experiments connected with it. Before dealing with individual criticisms, we also desire to say that we do not for one moment imagine that electric blasting is a panacea for all the difficulties attached to mining. We know only too well that it would be foolish to recommend its introduction into some mines, or even sections of some mines, but having said this, we stand in the main by our original statements.

We now pass on to individual criticisms. Mr. Pam raises some very interesting points. As that gentleman pointed out, it is quite true that safety fuse is still used in practically all the English metalliferous mines, but we advocate electric blasting on the Rand because the conditions are so different. We would, therefore, again remind him that our two most important reasons are, first, "health," and, second, "safety." Mr. Pam referred to the question of regularity of burning speed of safety fuse. It appears to be impossible for manufacturers to make fuse which has not a certain amount of variation in the burning speed, but, personally, we feel quite convinced that what is scientifically understood as a "running fuse" cannot exist in actuality. The designation "running fuse" means that a given length which should, we will say, burn through in about 100 seconds, either burns through practically instantaneously or in a few seconds only. A piece of fuse which should burn through under ordinary conditions in 100 seconds, but actually burns through in, say, 70 seconds cannot be called a "running fuse," but should be described as a very irregularly burning fuse. One of us has, in the course of his experience, tested tens of thousands of 6ft. lengths of fuse with stop-watches, and on no occasion has anything approaching a "running fuse" been found. We feel convinced that accidents so often attributed to "running fuses" must be in reality attributed to other causes, which it would be quite out of place to discuss in this reply. Mr. Pam and others have touched on the economic aspect of electric blasting, and we must confess that we are weak on this point, because when the paper was written few people had any practical experience of electric blasting in the metalliferous mines of the Rand. We were hopeful that at least one of these few would come forward with a paper on the economics of electric blasting, which would have rounded off the subject properly. Although up to the present we have been disappointed, we still hope that this will be done.

Since the paper was written it has been found by practical experience that iron wires can be used instead of copper wires for the leads, and this, while effecting a small economy, also removes any possible chance of metallurgical complications at the bullion end. While on this subject, we might mention that we have been officially informed that the use of copper leads has not caused any trouble in the gold recovery process at the same mine where electric blasting is employed.

We agree that with ordinary safety fuse a special team of expert blasters could perform the same function as the special team for the electric system, but the great point is, as Mr. Moynihan has pointed out, to have the men out of the mine before blasting takes place. We do not like to make any positive assertions about blasting the "cut" and the "round" together—a most important point—but we think that we now see a way out of the difficulty, although it is too early to speak with certainty on this point. We must reiterate, however, what we indicated in the paper, viz., that it is very bad practice to attempt to use instantaneous and delay action fuses in the same circuit. Recent investigations which we have made at the South Nourse Mine have convinced us that our theory is correct.

We now come to Mr. Weston, and we may say that we will take his criticisms seriously. We feel that we cannot do better than to refer to Mr. Moynihan's sparkling and inimitable contribution which so effectively dealt with Mr. Weston's "practical men." We are accused of having no practical mining knowledge; in order to prove, however, that we had some little knowledge of all the various subjects covered by our paper, we might be allowed to say that one of us has had a lifelong experience in practical underground mining operations in very many well-known mining countries, including Great Britain, Australia, South Africa, India and the Far East, and we will say no more on this subject. Mr. Weston can take it from us that no unrecorded incidents of any consequence took place during the attempted introduction of electric blasting on the Rand. Although, up to the present, electric blasting has not made much headway in metalliferous mines in the Transvaal, this system is very largely practised in the coal mines, and also to a lesser extent in the diamond mines. In coal mines there is no special reason why electric blasting should be used other than on purely economic grounds, seeing that the Transvaal collieries have no gas troubles to contend against.

With regard to the question of the smoke of the fuse holding fine dust in suspension, surely it is obvious that smoke given off by safety fuse, which is made up, in the main, of small tarry globules, cannot help holding dust in suspension.

Mr. Weston made the following statement:—"When an accident occurs with fuse in a stope or drive, one hole may go, but with electricity there would be a bombardment, from which no one would have a chance to escape." We cannot conceive how any such accident as portrayed by Mr. Weston could occur with electric blasting, unless either murder or suicide were contemplated. Perhaps Mr. Weston contemplated carelessness on the part of the operators. Well, if a man carelessly stepped into the shaft instead of stepping into the cage or skip, he would probably lose his life. We thought we had made it quite clear that all workers should be either out at the shaft station at least, or preferably out of the mine altogether, before any electrical current was switched into the firing line.

In spite of Mr. Weston's somewhat caustic remarks about the blasting of the "cut" with instantaneous electric detonators, we wish to say that our dicta are based on experience gained in a great many mines, and we see no sound reason for withdrawing from the position which we have taken up. We admit that if a special blasting gang were employed where safety fuse is used, there would also be more time available for drilling holes, but the whole object of our paper was to advocate the diminution of blasting with safety fuse as much as possible. To finally sum up Mr. Weston's criticisms, we consider that most of his points are of a dialectic nature, and we will pass on.

Mr. Veasey states that manufacturers should supply tested, or even guaranteed fuses. Our experience goes to show that electric delay action fuses are about as free from defects which would cause misfires as it is possible to make them. Further on, he asked us to revise our relative costs, in view of the fact that in a wet shaft, the junction of the time fuse and detonator has to be carefully waterproofed by greasing. It is difficult to assess an item of this sort, but it is only fair to point out that with delay action fuses as opposed to instantaneous, greasing would still have to be done.

Mr. Moynihan's criticism is somewhat eulogistic, and calls for no particular comment, but we thank him for having taken up the cudgels on our behalf.

We much appreciate Mr. Chilton's thoughtful contribution to the discussion. We do not assert for one moment that electric blasting would eliminate all the accidents which he quoted (p. 376, February Journal), but we certainly think it would reduce their number. We agree with him in his remarks on specialisation, and we hope that the discussion on this paper will enable mining men to look on the subject of blasting generally in a new light, and we truly believe that the time is not far distant when, to quote Mr. Chilton, "The breaking of rock will be in the hands of a few carefully chosen miners on each mine, who will commence to load up the holes after the drilling shift is done."

THE PREVENTION OF DUST IN UNDERGROUND WORKINGS.

(Read at January Meeting, 1914).

By B. C. Gullachsen, M.I.M.E. (Member).

DISCUSSION.

The President: I hope any member who has any suggestions to offer regarding this important subject of dust prevention underground will put them before us. Mr. Gullachsen has given a useful summary in his paper, and mentioned several directions in which improvements might be made. I think, however, when he has had an opportunity of going more fully into what has been done in this respect, he will find that most, if not all, of his suggestions have already been put into practice, as well as a great many more to which he has not referred.

RELATION OF FALLS OF ROOF IN COLLIERIES ON THE MIDDELBURG COAL FIELD TO WEATHER CHANGES.

(Read at March Meeting, 1914.)

By Chas. J. Gray, A.R.S.M., F.G.S., M.I.M.E. (Member).

DISCUSSION.

Mr. T. N. Dewar (*Member*): The paper just read is of special interest to those members who are engaged in coal mining. Doubtless many examples can be cited and one from Natal will be briefly referred to.

At a colliery there the development heading was driven for a distance of 500 yards before a ventilation connection was made. The section of coal exposed was about 3 ft. thick with an overlying shale band 9 in. thick. Above that the roof was composed of hard close grained sandstone. The parting between the shale and the coal was very clean. During the operations described the shale gave no trouble in breaking away from the sandstone roof. As the mine was only 60 ft. deep the temperature and humidity of the air current were affected by weather changes. It was noticed that a deposition of water and drying action such as is described by the author occurred on the section of the drive near the downcast shaft. Some time elapsed and it was found there was a strong tendency for the shale to break away from the roof. It became so marked that ultimately it was necessary to remove the shale entirely as it was falling between the timber sets which were placed about 4 ft. apart. No special attention was paid to these changes of weather with regard to these falls, but the writer is satisfied that the changes had something to do with it as the deposition of moisture and the drying action were so marked. The sandstone bed contained very few joints and the surface contour was so steep that water would not lodge but would run off quickly into the spruits leading from that area. It is not likely therefore that there was percolation to any great extent.

The author writes that the "atmospheric conditions can seldom cause deposit of moisture on the roof near the working face as that can occur only in hot moist weather and when the face is near the mine entrance. Generally such condensation as there may be, will be on the intake airways." It seems to the writer, in view of these facts, that in discussing the statistics of accidents from falls of roof that the location of these accidents should be considered. One would like to know whether the falls have occurred in main intake airways which are generally the main haulage drives or in the neighbourhood of the face itself where the conditions are as a rule unaffected by weather changes. If the accident figures were analysed in that manner, the information would be of greater value to the discussion.

The question of the degree of saturation of mine air is of interest. If the ventilation of a mine is increased, the moistening or drying effect on the surface of the drives, etc., becomes greater. On the proposition put forward in the paper it appears likely that with the improved ventilation which modern mining practice demands in all mines, the accidents will increase unless supervision meets the conditions.

When the question of the relative degree of humidity of the air is further discussed it is evident that the passing of a little steam into the main intake airways will meet the difficulty outlined in the paper. The temperature will be raised and the saturation completed. One would not care to say that it is a practical mining scheme, but as a suggestion it is of passing interest.

The writer is not informed as to the conditions prevailing in the Middelburg coal field, but possibly it is similar to other fields where the coal seams occur at shallow depths. In workings of that nature the means of entry and exit are generally fairly easy. In the winter time there may be difficulty in getting the miners early to work. Cases may have occurred where boys have reached thir work before the examination of the roof has been made. Is it not possible that this hypothesis may affect the accident rate at Middelburg? The writer's intention is merely to point out a possible factor in considering the statistics before us.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

LEAD CYANIDE.—"According to Rammelsberg (D.R.-P. 139456), the compound $Pb(CN)_2$ is formed by precipitating a solution of a lead salt with aqueous hydrocyanic acid or a soluble cyanide, whereas Kugler (*Annalen*, 1848, *66*, 63) states that a basic salt, $Pb(CN)_2.2PbO,H_2O$, is formed by precipitation from an ammoniacal solution. It appears, however, to be generally recognised that the precipitate formed by the interaction of cyanides and lead salts in aqueous solution varies in composition with the concentration of the solutions employed, a fact which the author has confirmed.

Lead cyanide was decomposed by hydrogen sulphide, and the hydrogen cyanide was lead into water through a U-tube containing lead cyanide. To remove any traces of hydrogen sulphide from the aqueous hydrocyanic acid, some lead cyanide was added to it, and the solution shaken. As no trace of hydrogen sulphide was present no lead sulphide was formed ; but after filtering this solution it was found that it contained a considerable quantity of lead. It was evident that lead cyanide, which is not appreciably soluble in cold water, is soluble in aqueous hydrocyanic acid, and it was expected that this solution would, on evaporation, deposit pure lead cyanide free from oxide.

About 250 cc. of a 5% solution of hydrocyanic acid were heated to boiling with a small quantity of precipitated lead cyanide under reflux. After about half an hour the solution was filtered and allowed to evaporate slowly in a desiccator. The deep yellow, needle-shaped crystals which separated were collected and dried in a vacuum.

Lead cyanide, even when powdered, appears to be unattacked by concentrated nitric and sulphuric acids in the cold. On adding water to a crystal, the insoluble oxycyanide is formed, and the water becomes cloudy.

The crystals on analysis were found to be lead cyanide having the formula $Pb(CN)_2$."—Nalini Mohan Gupta, *Proc. Chem. Soc.*, Vol. 29, No. 421, p. 361. (C. T.)

The Composition of Coal.—"An account was given of an examination of the liquid products of distillation of coal in a vacuum at low temperatures, from which conclusions were drawn regarding the composition of the 'resinous substances' that form part of the coal conglomerate.

A description was given of the various compounds contained in the oils obtained, which consisted largely of unsaturated (ethylenic) hydrocarbons, naphthenes, paraffins, phenols (chiefly cresols and xylenols), and homologues of naphthalene. Benzene, anthracene, and solid aromatic hydrocarbons were stated to be absent.

The presence or absence of free hydrocarbons in any quantity in coal was discussed, and an hypothesis put forward to account for the rapid formation of paraffins, naphthenes, etc., on distilling coal at low temperatures. It was suggested that such hydrocarbons must be present in the coal substance in such a manner that whilst, in a sense, structurally complete, some change in their state, such as can be produced by moderate heating, must take place before they can be set free.

This hypothesis, particularised for the case of the paraffins, assumes their existence in coal as alkyl groups attached chemically to another non-alkyl group, R·H, the paraffin being in what may be termed a 'bound' condition, as a component part of a molecule represented by the general formula $RH - C_nH_{2n+1}$. The rapid distillation of 'free' paraffins from these 'bound' molecules when coal is decomposed thermally was then explained according to the following scheme:

$$RH - C_nH_{2n+1} \longrightarrow R + C_nH_{2n+2},$$

or

$$RH - C_nH_{2n+1} \longrightarrow R + C_{n_1}H_{2n_1+2} + C_{n_2}H_{2n_2}.$$

With certain modifications the hypothesis was used to explain the appearance of naphthenes, olefines, and napthalene homologues in coal distillates."—David Trevor Jones and Richard Vernon Wheeler.—*Proc. Chem. Soc.*, Vol. 29, No. 422, Dec. 30, 1913, p. 376. (C. T.)

Solubility of Calcium Carbonate and Its Bearing on Water Purification.—"The lime softening process does not remove the whole of the calcium carbonate from water. Water free from dissolved carbon dioxide dissolves calcium carbonate to the extent of 1·5 parts per 100,000. The complete separation of the precipitated calcium carbonate requires a much longer time than is usually supposed; even after the precipitate has settled, the clear water continues to deposit crystalline calcium carbonate for some days. The more perfectly the water is softened the greater the necessity for giving it a long period for subsidence. When it is desired to use the water as soon as possible after the precipitate has settled, it may be treated with a small quantity of carbon dioxide to prevent deposition of calcium carbonate in the mains (compare Archbutt and Deeley's process: *J. S.C.I.*, 1891, 511); as little as 0·3 part of carbon dioxide per 100,000 of water is sufficient. In the method described by Houston (*J.S.C.I.*, 1912, 508) for sterilising water by the addition of an excess of lime and subsequent removal of the excess by carbon dioxide or sodium bicarbonate, it would be necessary, owing to the slow deposition of the last portions of the calcium carbonate, to take precautions to prevent the precipitation of calcium carbonate in the water mains."—W. T. Burgess, (Inst. Water Engineers).—*Journal of the Society of Chemical Industry*, Feb. 16, 1914, p. 158. (J. W.)

Determination of Titanium by Titration with Methylene Blue.—"Brief accounts of all published methods for Ti are given. Knecht and Hibbert's methylene blue method gives accurate results and is a satisfactory, easy method if carried out as follows: 1 gm. of the sample is fused with KOH and Na_2O_2 and the melt dissolved in concentrated HCl. After filtering an aliquot is taken for the Ti determination. If H_2SO_4 or HNO_3 results from the decomposition of the sample, the Ti must be precipitated with NH_4OH and redissolved in HCl. To the clear solution 30 cc. of concentrated HCl is added and about 2 gm. Zn dust with some pieces of Zn. Reduction takes 15–20 minutes and then the solution is filtered into an Erlenmeyer flask into which a current of CO_2 is passed. 20 cc. of concentrated HCl is added and a piece of Zn is suspended in the solution by a Pt wire passing through a Bunsen valve. After heating for a few minutes, the Zn is drawn up out of the solution and the methylene blue solution is added from a burette whose outlet is thrust through the Bunsen valve. The best results are with a solution of methylene blue containing 3·9–7·8 gm per litre. This solution is standardized against a solution of Ti of known strength."—B. Neumann and R. K. Murphy (*Z. angew. Chem.*, 26, *Aufsatz*, 613-6). *Chem. Abs.*, Vol. 8, No. 2, January 20, 1914. p. 308. (J. G.)

Detecting Platinum Metals in Beads from Cupellation.—"The presence of platinum metals in cupellation beads can be detected by examining the surface appearance of the cool bead under a low power objective, such as a 1 in. or a ¾ in. Photographs showing characteristic structures are given. *Platinum.* 1·6% Pt in silver produces changes visible to the naked eye, and 0·3% can be detected with certainty under the microscope. The boundaries of the crystals become distorted, and a banded structure develops. The effect of platinum on gold is similar but less marked. Beads containing equal quantities of silver and gold have the 'frosted surface' appearance caused by over 2% Pt. *Iridium* makes the bead more spherical. Under the microscope, the crystal faces appear strongly marked with lines crossing one another after the manner of slip bands. The marks are ascribed to internal stresses resulting from occluded oxygen. The effect due to 0·4% Ir is clearly visible. *Rhodium.* Traces of rhodium, such as 0·004%, in silver beads, cause a distinct crystallisation visible to the naked eye. With 0·03% Rh, and upwards, the beads spit persistently, and assume a blue-gray colour. Gold with 0·2% Rh has a ruby colour, and with 0·8% is covered with a black film. *Ruthenium* produces a black crystalline deposit at the edge of the bead near the bottom, and gives a 'single-sided herring-bone structure' to the surface of the bead. The deposit with 0·004% Ru is small but distinctly visible to the naked eye. No increased tendency to spit was observed. *Palladium.* The effect is similar to that of platinum, and palladium may be detected by a colouration of the parting acid, even 0·0002 gm. Pd producing a faintly yellow colour. *Osmium* up to 2% gives no characteristic appearance to the silver bead, and *osmiridium* gives similar results to those obtained with iridium alone."—C. O. BANNISTER and G. PATCHIN, *Journal Soc. of Chem. Ind.* (*Inst. Min. and Met.*) Jan. 15, 1914, p. 29. (J. W.)

METALLURGY.

TREATMENT OF GOLD BEARING SCHEELITE IN NEW ZEALAND.—"The scheelite deposits of New Zealand are important sources of tungsten. One of the mines, the Golden Point, has been operated for twenty-five years, and is the principal producer.

The ore is sorted in the mine into first and second grades. On reaching the mill, the first grade is again hand-sorted, yielding from 1 to 2 tons per month of a product containing from 60% to 65% WO_3, which is marketed. The reject from the hand-sorting is crushed and jigged, producing a concentrate of 60% WO_3 and a tailing which is added to low-grade ore. The low grade portion is crushed in stamps through 30 mesh screens, and run over amalgamated plates to recover some gold. The value of the ore usually averages $3.60 in gold, and of this $1.55 is caught on the plates. After amalgamation the pulp is classified and concentrated on Wilfley tables. The products are (1) a low-grade tungsten concentrate containing 30% WO_3 and 60% FeS_2; (2) slime tailing which is discharged, and (3) sand tailing which is leached with cyanide solution. The recovery in the cyanide plant is $1.30 per ton of ore. The low-grade tungsten concentrate amounts to only ½% to 1% of the ore milled, and contains about 2½ oz. gold per ton. This material is then dried and roasted and subjected to magnetic separation for the removal of pyrite. The tungsten product runs 60% WO_3; the pyrite contains from 5 oz. to 6 oz. gold per ton. The costs of milling 1.998 tons of ore, in cents per ton, were: Breaking and jigging, 3·8; stamping, 55·8; concentrating, 8·0; general, 7·6; cyaniding, 26·6; magnetic separation, 36.0; total, $1.378."—C. W. GUDGEON, *Australian Mining Standard*, Nov. 13, 1913.—*Metallurgical and Chemical Engineering*, Jan., 1914, p. 53. (J. A. W.)

MOLYBDENUM IN CYANIDE SOLUTIONS.—"*Synopsis.—Tellurium was found in an ore brought from Central America. Tests on a caustic-soda extract showed the tellurium, but also some other element which was for a time supposed to be something new, but which eventually proved to be molybdenum.*

Action in Cyanide Solution. The sulphide was found to be extremely soluble in cyanide, forming a yellow solution, which gives a reddish turbidity on acidulating with hydrochloric acid, and a dark-red precipitate on continued boiling. The form in which the molybdenum occurs in the original cyanide solution has not yet been definitely ascertained. This ready solubility, however, indicated that it may act powerfully as a cyanicide and the diminished cyanide consumption noted after preliminary extraction with caustic soda may be largely due to the elimination of molybdenum by the latter.

In order to observe the effect of pure molybdenum compounds on cyanide, the following experiments were made: A solution of sodium cyanide was prepared, which, tested in the ordinary way, showed 1·242% KCN and 0·028% CaO. (1) 25 cc. of the NaCN solution, with addition of KI required 56·05 cc. standard $AgNO_3$ = 2·242% KCN.

(2) 25 cc. NaCN solution + 0·1 gm. molybdic acid, + KI required 48·55 cc. $AgNO_3$ = 1·492% KCN, indicating a consumption of 0·75 gm. KCN for every gram of molybdic acid added. The solution smelt strongly of HCN and after addition of $AgNO_3$, as above, was no longer alkaline to phenolphthalein.

(3) 25 cc. NaCN solution + 0·5 gm. ammonium molybdate + KI required 36·45 cc. $AgNO_3$ = 1·458% KCN indicating a consumption of 0·392 gm. KCN for every gram of ammonium molybdate added. This salt also acts as an acid toward cyanide as the solution smelled strongly of HCN and after addition of $AgNO_3$ was not alkaline to phenolphthalein.

(4) 65 cc. NaCN solution + 0·1 gm. molybdic acid, + a few cc. of strong NaOH solution, and KI required 55·95 cc. $AgNO_3$ = 2·238 = KCN.

(5) 25 cc. NaCN solution + 0·5 gm. ammonium molybdate + a few cc. strong NaOH and KI required 55·95 cc. $AgNO_3$ = 2·238% KCN.

These tests show that the compounds tried act as acids toward cyanide, but have no decomposing effect in presence of sufficient protective alkali. It may therefore be supposed that molybdenum in the solutions is present, not as a cyanide, but as a sodium salt in which molybdenum forms part of the acid radical."—J. E. CLENNELL.—*Engineering and Mining Journal*, Feb. 14, 1914, p. 363. (A. R.)

SLIME AGITATION FOR CYANIDING.—"The first efforts were made by introducing compressed air into shallow, round or rectangular vats containing the slime pulp, that is, the colloid material mixed with varying proportions of water. A small pipe connected by a hose to an air-main was introduced into the vat and an operator kept it moving from place to place continuously, thus keeping the material stirred up. The object was to maintain the solids in suspension, allowing the solution to act upon them. This process, while expensive, was a success because it gave a profit over its cost, and because it showed that there was no difficulty in dissolving the gold and silver from the slime, the only trouble being the mechanical one of keeping the solids in suspension, and after the dissolving process was complete, in separating the solids from the solution. The percentage of extraction, that is, the percentage of metal dissolved by cyanide solution, was considerably higher than had ever been obtained through leaching the granulated portion, or sand.

At the mill of the El Oro Mining and Railway Co., at El Oro, Mexico, the early slime-agitation vats were flat-bottomed, rectangular steel vats with rounded corners. In these, slime pulp was kept in agitation by a ½ in. pipe, drawn to a point, connected by hose to an air-main. A Mexican was employed to keep this hose moving about from place to place and in this way succeeded in keeping the solids in suspension successfully and rather cheaply. A good many other plants followed this system and there are plenty of examples where round, flat-bottomed wooden vats were used in the same way. An interesting feature of the use of the air pipe in steel or iron vats was that where a little sand was mixed in with slime and the air pipes allowed to remain in one place for a considerable length of time, the result was a neat, round hole drilled in the bottom of the vat.

A variation of the movable-pipe jet of compressed air was the installation of fixed pipes in the vat, the pipes having a number of holes drilled in them which discharged the compressed air at a number of different points throughout the vat. This worked very well the first time, but the pipes soon got plugged up and the result was that there was very little agitation taking place. These stationary pipes were used in a great many installations, but were not satisfactory in any. The entrance of lime stopped up the pipe so that the air came out at only a very few points, and these were likely to be in scattered sections of a vat, the result being that about four-fifths of the material was without agitation.

Mechanical Stirring of Pulp.—The next forward step was to institute agitation by mechanical means. This was done by using a round, flat-bottomed vat in which a central vertical shaft was installed, capped with a crown-wheel and moved by a pinion fixed to a horizontal shaft passing over a series of vats. The vertical shaft was equipped, near the bottom of the vat, with arms reaching out nearly to the sides, and these arms, revolving at medium speed, kept the contents of the vat in agitation. A little later development of this same idea was that the arms were placed about half way of the depth of the vat, and depending from them were a number of swinging iron rods or pieces of wire rope weighted by chunks of iron at the bottom. The object of this improvement was to avoid the necessity of digging out a vat whenever a long stoppage occurred. In such cases, when the arms were near the bottom of the vat, they become immersed in the settled slime and held so solidly that it was impossible to move them in the regular way without breaking something. By having the arms about half way up the vat and having these swinging iron rods or pieces of wire rope reaching down into the slime, the arms were not so likely to be held firmly but could move freely when the power was applied. The iron rods or wire-rope pieces would pull out of the slime very easily and drag over the top of it, mixing it little by little as agitation progressed. By this method they would dig down into the slime and soon have it all in circulation.

Mechanical agitation in this form was usually assisted by centrifugal pumps, which changed the comparative location of the solid particles. It will be readily seen that a floating particle in cyanide solution making simply a number of revolutions in a vat would not come into contact with any other solution except that which immediately surrounded it. There would be no up or down motion to the particle to any great extent, nor would the comparative location of any particular particle and its surrounding solution be changed. By pumping from the bottom of the vat with a centrifugal pump and throwing the pulp back into the top, a total change of location was assured. By pumping from near the outside of the vat and returning the pulp near its centre, a thorough change of location could be obtained.

Mechanical agitation did not continue popular because operators believed that its costs were too high compared to the work it did. It has been supplanted in most cases by a variety of simple and complex devices, all destined to remove the troubles that agitation is heir to. At the last report, however, we find that a recent cyanide plant, one of the most modern in the country, has built its new agitation vats on the mechanical system. They have passed by the newer and more fashionable devices and returned to the round, flat-bottomed vat with agitating arms, and the whole assisted by centrifugal pumps. The designers of this mill claim that while mechanical agitation, as they use it, may not be absolutely perfect, neither is any other system perfect, and no one of them can show any substantially lower costs than are obtained by this system. While this statement may require modification, it probably is sufficiently true to give rise to serious thought on the part of modern cyanide-plant designers.

Varied Forms of Mechanical Agitation.—Mechanical agitation, during the time it has been used by American metallurgists, has taken a great many different forms. The simplest is that which has already been described—horizontal arms revolving within a flat-bottomed round vat. A form which has been largely used in Mexico, particularly in Guanajuato, where it was devised and first installed, is one in which the central vertical shaft and the horizontal arm are both hollow, compressed air being introduced through the arm to various outlet points, so that while the mechanical agitation was taking place, the air would be forced out through the nozzles into the pulp in the vat. By this means it was attempted to obtain aëration in addition to mechanical agitation, the air assisting in the agitation as well as adding its oxygen to the charge.

Another system more or less well known is that devised by Hendryx in which the vat is usually provided with a cone bottom and in its centre is a rather large tube extending from near the bottom to near the top of the vat. Within this tube is placed one or more turbines or propeller wheels, which are revolved upon a central shaft. The movement set up takes the material from the bottom of the vat, forces it through the tube and discharges it at the top, thus obtaining a circulation from bottom to top throughout the vat. There is very little doubt that this gives good agitation, the principal objection which has been raised to it and one which has apparently been maintained is that its cost is more than is incurred by any one of several other systems. In fact, only recently a large installation has dispensed with its Hendryx agitators because of their high cost in competition with other machines. On a small scale, or in laboratory work, however, this agitator is particularly good as it gives an efficient direct agitation which is very sure to keep all the solids in suspension.

The Air-Lift Agitators.—Probably the most notable development in agitation devices is the Brown or Pachuca vat, which was introduced eight or nine years ago. It was invented and first used in New Zealand where it proved an economical success and was promptly transferred to Mexico, where its use has become widespread. The Pachuca agitator is nothing more or less than an extremely tall cylinder with a cone bottom fitted with an air lift in its centre. The transferring of pulp from the bottom to the top is, in effect, very much like the operation of the Hendryx vat, but it accomplished by compressed air in place of mechanical movement. The vat is made tall and narrow and with a cone bottom in order that the influence of the air lift may extend to all parts of it, which would not obtain if its area were large in comparison to the size of the air lift, and also to prevent any settling upon the bottom. For this purpose, the cone bottom required is exceedingly steep as the tendency of slime, particularly of the light and flocculent variety, is to settle on any surface flat enough to hold it. Even with the steepest construction possible, however, there is very likely to be some settlement in the vat, and often there is a great deal of it.

The hold which the Pachuca vat took upon the metallurgical profession immediately after its introduction, was remarkable. It spread rapidly into plants, going to a large majority of the plants which were planned or under construction, and also into a large number of them which had already been built, displacing agitators of other types Unfortunately it seems that enough study had not been given to the matter to decide absolutely just what its advantages were. It is, moreover, quite probable that only by the practical experience following the continued use of this device could its disadvantages have been discovered. The principal drawback is that it takes more power by far than was originally claimed for it, which means that the cost runs higher than it should to procure satisfactory agitation. Another difficulty is that it is almost impossible to avoid settling of solids. If there is any sandy material in the pulp, it will settle on the bottom and build up on the cone sides no matter how steep they are, and leave only a small channel to one side through which the circulation takes place. This not only reduces the capacity of the vat, but misleads the operator as to agitation. At the top of the vat no notice of this reduced agitation can be obtained because the surface of the vat is usually covered with pulp and agitation is apparently going on as usual If one, however, probed the vat with a long stick he would be likely to find a very solid deposit of sandy material in the vat which occupies a goodly proportion of it.

Additional Drawbacks of Air Agitation. Some operators have claimed that the carbon monoxide generated by the compression of air through the lubricating oil used and heat developed, is deleterious to cyanide solutions, also that compressed air cooled the solution to a great extent. All of these objections may have some weight, but the two principal ones, its cost and the tendency toward settling, have been decisive in putting a rather sudden halt to the installation of Pachuca vats. It is quite likely that fewer of them will be installed in the future.

Other machines have been installed to accomplish the same result, and the first success of the Pachuca encouraged many inventors to proceed with that device as a basis. Among the well known instances of this may be mentioned one of the hydraulic agitators which is a good deal like the Pachuca, in fact almost identical, except that the compressed air in the elevating central tube was replaced by a hydraulic jet. The vat has some advantages, but probably does not overcome either the items of cost or the propensity to settling. Some of these vats are used but they are not extremely popular at the present time.

The Parral vat has had considerable application and may be regarded as one of the standard appliances. It is simply the installation of several air lifts in a flat-bottomed vat, the discharges being set tangentially so that a rotary flow of pulp is secured. The vat is well known among operators.

A machine which has been widely used and which was original in its ideas, is the Trent agitator, about which much has been said for and against. Discussion about this machine has been particularly acrimonious, those in its favour becoming violent partisans, and those against it being with equal heat, opposed to its use under any circumstances.

Combination Devices. — The Trent agitator is essentially an arm agitator corresponding to the mechanical devices now in use, but the arms are hollow and the pulp is circulated through them. The great and essential difference of this machine is that the arms are not moved mechanically by any direct means, but are moved by the pulp itself, which is forced out of them by means of a centrifugal pump. The intake of the centrifugal pump is near the top of the same vat, where it can have practically clear solution, or at least solution which contains very little coarse, granular, or sandy pulp, while its discharge is through the bottom of the vat by the use of a special gland. The hollow arms, which are arranged to discharge the pulp all in one direction, act after the manner of a garden-lawn sprinkler, the reactive force of which causes the revolution of the arm. The pump is connected with a small air compressor for supplying oxygen to the solution, or a small valve is inserted in the suction through which it is allowed to take air from the atmosphere. The advantages claimed for the Trent agitator are, that it does not make expensive installations; that it gives satisfactory agitation at a low cost; and that it thoroughly aërates the pulp by adding air in minute particles to the whole pulp charge. It is also claimed that somewhat better extraction is obtained by the use of this machine. As far as these claims are concerned, that referring to the distribution of finely divided air to the charge is decidely true. The Trent agitator in operation shows clearly that there is a very large volume of air mixed in with the charge. If one starts a vat which has not been under agitation with a Trent agitator, within a short time after it has been started, the volume of the pulp will be seen to have increased appreciably. This is due to the air which it gives to the charge in such finely divided particles that it is not rapidly liberated,

The latest applicant for honours in the field of slime agitation is the Dorr machine. This device is an adaptation of the original Dorr thickener. It has a central shaft carrying two arms, which are set at an incline toward the centre of the bottom of the vat, these two arms carrying rakes which move the settled material toward the centre of the bottom of the vat. In the agitator the arms are arranged identically, but the central shaft is inclosed by an air-lift tube, or the tube itself may constitute the shaft. The tube terminates just below the upper level of the vat and discharges into a launder, which again distributes the pulp over the surface of the vat area. The launder is, of course, optional and the device can be operated without it. When used, the launder has discharge openings at various points along its length so that the pulp will be dropped at different points in the vat. The air pipe for operating the lift is usually put in through the bottom of the vat, this being a simple operation, as there is no step bearing for the central shaft, its weight and that of the agitating mechanism being carried on a truss above the vat. In this machine there is mechanically no possibility of settling of solids on the bottom of the vat and the agitation is positive. The slime discharged at the top of the vat settles through the solution, to a large extent, and is scraped to the centre of the vat again, and again elevated through the air-lift tube. The advantages of this vat are that the air-lift distance is low, the vat being of large area and low height, cheap to construct, and requiring little air at low pressure for the agitation. The arms are operated at low speed, and rake the slime gently toward a central tube. The arms probably do not take more than ¼ h.p. in a large vat, while the air-lift works under practically ideal conditions, requiring little power, the amount depending upon the depth of the vat. These vats have been installed in a number of mills, and reports of them up to the present time are favourable.

A recent patent shows the use of a principle somewhat similar to the Trent, or garden-sprinkler idea, the difference being that air alone is pumped through the arms, and escaping through the suitably-arranged arm openings, causes revolution. The whole mechanism is supported in a step bearing in the vat, the air entry being from above. The machine is known as the Hafer agitator.

Objects of Agitation.—It would serve no useful purpose at this time to enumerate the great number of agitating devices which have been tried from time to time, even those which may have been more or less successful, as those which have already been mentioned include the most important and are the ones which have received widest application among the operating plants of the world. It may, however, be of interest to discuss the principles of agitation from a general point of view.

Almost from the time of the inception of the cyanide process, oxygen has been believed to be indispensable in the reaction of dissolving gold and silver in cyanide solution. The truth of this having been demonstrated theoretically, and to a large extent, practically, operators went to extreme lengths to find means of furnishing this oxygen to their working solutions so that dissolution might be rapid and complete. Various chemicals have been used to supply oxygen, but these have all, or nearly all, been set aside as more expensive than the additional benefit derived through their use. Not having secured good results from chemicals, the next turn was to use air as a cheap source of oxygen. Air has been used for a number of years and great benefits have been claimed from it However that may be, it is by no means certain that air is a good thing to use for oxidizing slime pulp under agitation. In many cases it can be said that the agitation itself is all that is necessary, and agitation is only necessary in order that the particles of ore in an extremely fine state may be free so that solution may act upon them at all times. When slime is settled, it forms an impervious mass through which solution cannot pass, and therefore, solution is not at liberty to act upon the covered particles. The object of agitation is to keep these solids in suspension, so that the solution has an opportunity to act upon each particle continuously. Of course, if additional advantages could be obtained while this movement is going on and without additional expense, it would be undoubtedly advisable to make an effort to obtain them. For this reason, oxygen has been so widely used together with agitating processes.

It is somewhat unreasonable to suppose that by simply passing air into solution any great quantity of oxygen will be absorbed. It is true that some will be, but the extent of the change is not great, and no dependence can be placed upon it as a hastener of dissolution.

Recently, Morris Green* has shown that indiscriminate oxidation is not an advisable operation. However that may be, it is true that our ideas of oxidation will have to be somewhat revised, and in the meantime agitation had better be confined to the cheapest method of keeping fine particles of ore in suspension in solutions so that they can be acted upon with the greatest rapidity.

Cost of Agitation.—Agitation, like a good many other details of the cyanide process, defies reduction into concrete cost figures capable of reasonable comparison. Like crushing and grinding, agitation depends a good deal on the character of ore handled, and in addition, upon the specific gravity of the pulp handled, that is, the degree of dilution. It is clear that high cost at any one plant does not necessarily indicate poor work, but all data must be considered to make a fair conclusion.

At the Goldfield Consolidated mill, it is said that 75 cu. ft. of free air per minute is required to agitate a vat containing 85 dry tons. The installation consists of Pachuca vats. The cost is stated to be 2c. per ton agitated for power and under 1c. for repairs and maintenance, a total of less than 3c. per ton, not including labour.

At the Liberty Bell mill, with Hendryx agitators, the cost was 5·5c., including the cost of settling in Dorr thickeners. At the Hollinger mill, cost is said to be 2·9c. per ton with Dorr agitators. Experiments at the Liberty Bell mill showed that the new Dorr vats, holding 35 dry tons of ore, required 0·4 to 0·5 h.p for operating the arms and 40 cu. ft. free air, compressed to 8 or 10 lb., for the lift. At the West End mill, Tonopah, it is stated that a vat holding 88 dry tons of ore at a pulp sp. gr. of 1·25, required 6·5 h.p. to operate the pump, and about 1·5 h.p. additional for air for aëration. The McNamara mill agitates 97 tons at 1·26 sp. gr., with an expenditure of 9·5 h.p.

Costs are exceedingly elusive data with which to make comparisons, and cannot be depended upon unless all the factors leading to them are available for consideration. For comparing agitation systems, such items as cost of power, labour and all other factors; specific gravity of the dry slime, specific gravity of the pulp, percentage of sand in the slime, etc., are required. Knowing all of these, a fair inference may be drawn."—H. A. MEGRAW, *Engineering and Mining Journal*, Dec. 20, 1913, p. 1,161. (H. A. W.)

PROGRESS IN METALLURGY. — "Substantial improvement was made last year in methods of recovering gold and silver, but few striking innovations were brought forward. As has been the case in recent years, the cyanide process was the most important in point of quantity and value of metals recovered. Its use was extended to the treatment of a greater variety of ores; costs were reduced and extractions increased. Nothing of importance has occurred in the fields of amalgamation or smelting, and no data have been forthcoming from several attempts at chloridising. The electro-cyanamide process, originally installed at the Ajax mill, Cripple Creek, was finally abandoned in favour of combined concentration and bromo-cyaniding.

A year ago we commented on the filter litigation and the successful defence of the Moore patents. At the same time we suggested that the end of the litigation was not yet in sight. The prediction was verified late in 1913, when Mr. Charles Butters filed a suit against the Golden Cycle Mining Company, Colorado, for damages for infringement of filter patents. This outcome was the more striking by reason of the fact that the Golden Cycle was one of the first to settle with the Moore Company after the courts had sustained Moore's case against Butters. Mr. Butters has been active in other ways as well. In our department devoted to patents will be found a note on a new method of filter construction and operation, invented by Mr. Butters. His idea seems to be to avoid the Moore method of discharging the cake by a difference of pressure between the inside and outside of the filter leaf, and thus escape infringement of the Moore process. Another feature of the filter situation was the consolidation of the Oliver and Moore interests, possibly strengthening the position of both. On the other hand, the makers of the Portland filter, after investigating the whole matter of filter litigation, continued the manufacture and sale of their machines independently. While a number of large companies effected settlement with the Moore concern, many small users of vacuum filtration plants operated them without regard to the legal status of the business.

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* *Journ. Chem., Min., Met. Soc., S. A.*, February, 1913; and *Eng. and Min. Journ.*, June 21, 1913.

Consideration of the filter question naturally calls to mind the status of continuous counter-current decantation, as this process offers some measure of relief from the use of filters. Last year witnessed the greatest advances yet made in the use of this method of treating slime. Three companies abandoned filters in its favour: The Gold Road and Tom Reed, in Arizona, and the Lluvia de Oro, in Mexico; and a number of new plants adopted this method. The Gold Road Company reports decreased loss both in dissolved and undissolved gold, partly due to further dissolution of metal in passing through the series of thickeners, although it was thought formerly that the plant had ample agitating capacity. Operating costs have been lowered. The quantity of cyanide used, covering both chemical consumption and mechanical loss, has been increased by not more than 1-10 lb. per ton, although the ore is treated in a 2 lb. solution.

At the Globe & Phœnix, Rhodesia, South Africa, a 350 ton continuous counter-current decantation plant was installed last year. Loss of dissolved gold in the tailing is reported to be only 3c. per ton. A similar system was adopted by the Liberty Bell in Colorada, for the creatment of concentrate formerly sold to smelters. The new process affords a material saving to the company. In this case, in order to simplify treatment and reduce dissolved metal loss to a minimum, the tailing has been discharged into the regular mill circuit, ultimately passing out through the Moore filter. This gives the tailing additional agitation treatment and permits addition of all water at one place, *i.e.*, the filters. No additional labour has been required in the mill, and the total cyanide used is said not to be increased.

A late report from the Porcupine Crown mill, where the continuous counter-current decantation system was adopted, states that an extraction of 96% is being obtained. This result is ioteresting in view of the fact that concentration is not practiced in this mill as in some others in the same district.

The tall form of air agitator for slime, commonly known as the Pachuca, seems to have fallen in the estimation of American metallurgists, for most of the new installations have made use of combination air-mechanical agitators, such as the Dorr and Trent. The former, first described in our issue last January, has been adopted in new mills in Nevada, Colorado, Canada and Mexico, and some older forms of agitators have been displaced in old plants by it. Experience with this form of agitator indicates low cost for power, continuity of operation and mechanical efficiency. Sand and concentrate are readily handled, and the treatment is adapted to the chemical requirements of the ore, thereby giving high efficiency in dissolution.

The use of aluminium as a precipitant of gold and silver from cyanide solution received increased attention last year. This method is not new, having been proposed and patented in 1893 by Carl Moldenhauer; but the idea was never developed on a practical scale. Various experimenters have tried Moldenhauer's scheme, but it was not successfully used until Kirkpatrick developed the application of aluminium dust in 1908.

Dissatisfaction with zinc precipitation at the Nipissing mill, led to the investigation of the merits of aluminium, with the result that it was finally adopted. A study of conditions at this plant showed that precipitated solutions had lost their normal dissolving power, and the cause was finally attributed to the presence of zinc and arsenic in solution. Since the adoption of aluminium, no further difficulty of this sort has been experienced.

There are a number of advantages in using aluminium for precipitation. Although it is more expensive than zinc, yet much less of it is used per ounce of metal precipitated. Aluminium does not combine with alkali cyanide as does zinc, hence does not cause consumption of cyanide. When the precious metals are precipitated, all of the cyanide combined with them is regenerated. This necessitates the presence of free caustic alkali in the solution. One of the obstacles to the successful use of aluminium dust has been the difficulty of wetting it and getting a thorough contact with the solution. This is overcome by the use of some form of vortex mixer.

Another new proposal in precipitation made last year was that of MacArthur, who suggested the use of zinc wafers or small sheets. He reported very satisfactory results to the Chemical Metallurgical and Mining Society of South Africa, but subsequent investigators in different parts of the world were unable to confirm his figures.

The old scheme of volatilizing gold and silver from sulphide ores by roasting with salt was revived last year as a new invention in Australia. As pointed out in our columns at the time, the process of volatilization had been exploited in this country over ten years ago, but for various reasons was not successful. The Australian venture also is said to have been dropped.

The development of low-grade gold mines in Alaska has resulted in plans for milling on a huge scale. At the Alaska Gastineau the developments has been so rapid that each succeeding report of progress has far outdone its predecessor in reporting available ore, and the tonnage now in sight is beyond mental conception. Prospective mill capacities also have increased by leaps and bounds, and following the last visit of the promoters to the property the prospective mill capacity jumped from 6,000 to 20,000 tons per day. But with ore now said to run into the hundreds of millions of tons, the present operators can project mills of almost any old capacity without depriving their descendants of an opportunity to work the property and derive some of the enormous aggregate profit now in prospect. Actual milling operations will be awaited with interest."—Anon.—*Met. and Chem. Eng.*, Jan. 1914, p. 3. (H. A. W.)

MINING.

Sensitiveness of Explosives.—"In his report, Mr. E. A. Mann points out that some of the explosives imported into Australia exhibit a tendency, as a result of storage, to lose their sensitiveness to ignition by detonator to such an extent as to be unfit for mining purposes, and nearly all the explosives show some falling off in this respect. A recommendation has accordingly been submitted to the Government to the effect that regulations should be framed specifying a minimum velocity of detonation for all explosives imported, the same varying for the different classes."—No. 252, Oct., 1913, p. 146.—(T. D.)

IMPORTANT IMPROVEMENT IN BLASTING.—"The detonator was an important item in connection with blasting explosives, for it served to fire and develop the full force of the 'high' explosive class. For many years fulminate was the chief ingredient of the detonator charge, but now trinitrotoluol, or picric acid, or tetranitromethylaniline was used as the chief ingredient, with a small priming charge of fulminate, and more recently lead azide and also tetranitroaniline had been tried.

Increased power could be obtained by the use of a detonating fuse of trinitrotoluol inserted alongside of the charge of explosive, especially in deep holes, as the trinitrotoluol detonated with greater velocity than the blasting charge, and caused it to explode more simultaneously than when ordinarily fired by a detonator placed in the top cartridge."—Explosives by W. MACNAB, Institute of Chemistry Lecture, *Chemical News*, February 26, 1914. (J. M.)

MISCELLANEOUS.

ACETYLENE.—"Before the production of acetylene from carbide the acetylene was made by the incomplete combustion of coal-gas, and in this process more or less carbon monoxide was present in the acetylene obtained. What wonder, then, that there was some degree of toxic action of the acetylene examined! It contained carbon monoxide, the poison of common illuminating gas.

With the discovery of carbide and its use for the production of acetylene, all of this has been changed. It is found that carbide acetylene does not contain carbon monoxide and that it does not have the property of fixing haemoglobin and that it does not rob the blood of its capacity to take up oxygen from the air and carry it into the tissue. Hence the old allegation that acetylene is a poison because it deprives the blood of its oxygen-carrying capacity is no longer justified.

Another poisonous product sometimes present in the acetylene made by the combustion process was hydro-cyanic acid. Never in large quantities, it yet is so toxic that we can fully appreciate its effect. It is not present in the carbide acetylene, and so may be dismissed from consideration.

Another charge that is no longer justified is that acetylene is a poison because of the presence of phosphine as an impurity. This forms when carbide is made from limestone containing phosphate, the action of the coke reducing the phosphate. The selection of limestone free from phosphate has practically obviated this impurity, and any poisonous action of the acetylene consequent thereto.

Indeed, the present day product may be said to avoid the pitfalls of impurities so that its effect is determined by the action of acetylene itself. We may consider then whether acetylene as such is or is not a direct poison.

My present observations have been directed to the inquiry whether it produced noticeable effect on human subjects when present in increasing amounts up to 2½% during a period of 2½ hours. To this end, four men, including myself, were enclosed in a room of about 800 cubic feet capacity and at the beginning and four times subsequent at intervals of a half hour, acetylene was liberated into the room by throwing 450 gm. of carbide into an open tub of water, this corresponding to the liberation of 4 cub. ft. of acetylene, each time; that is, 20 cub. ft. in all, 2½% of the capacity of the room.

To eliminate, as far as possible, the mental effect of the environment, the subjects were engaged in playing a game of cards. They were interrupted only long enough to take readings of their blood pressures, at half hour intervals. The results of the experiment were quite negative. The game was continued throughout the time, excepting as noted. The blood pressure remained constant with one subject and was very slightly lowered from the inactivity with two and, of course, in the card game two men were defeated and two won, but there was absolutely no effect noted that could be ascribed to any poisonous or other action of the acetylene. It was without effect.

This same result has been obtained in experiments on animals. In such amounts as used in the above experiments there is no effect. Indeed, acetylene may be increased up to 20% and, if the mixture is so made as not to reduce the amount of oxygen, animals may be left in the atmosphere for some time, an hour or more, and will only become drowsy, from which they quickly recover when removed into ordinary air.

With very large quantities or with 20% admixtures acting for a longer time, the degree of drowsiness is increased. That is to say, the effect of acetylene in large doses is that of a narcotic, producing loss of consciousness in proportion to its degree of action. When this action is pushed to a fatal termination, the final action is upon the breathing centre, inhibiting its action and so producing death.

It thus appears that carbide acetylene is not poisonous in the sense that common illuminating gas is and that in large quantities, acting for some time, it produces a narcotic action. In respect to its toxicity, it presents no problem of adjustment under ordinary conditions. It, of course, may not replace in large degree the atmosphere we breathe, but otherwise it need not be anticipated that it produce any poisonous action." E. E. SMITH, *Mining Science*, January, 1914, p. 39. (H. A. W.)

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Abstracts of Patent Applications.

(C). 260/13. Robert Blane (1), Jacob Donald Silcock (2). Improvements in rollers for haulage ropes and the like. 6.6.13.

This application refers to the bearings of rollers for winding and hauling ropes, such as are commonly used in incline shafts and on rope haulages.

The improvement claimed is the provision of a removable sliding piece in the side of the grease box nearest the roller, so as to allow of the removal of the roller and spindle without removing the main casting from its place; the sliding piece is kept in place by a projection on the hinged lid of the box, and can only be removed when the lid is wide open, and in a vertical position.

(C.) 516/13. William Thomas Anderson (1), Edward Henry Johnson (2), Percy Ralph Nethersole (3). Improvements in treating mine dumps or other similar depositing sites for allaying dust. 25.10.13.

This specification refers to the application to the surface of mine dumps, or other similar depositing sites, of an emulsion of clayey or similar earth or soil for the purpose of allaying dust or preventing comminuted minerals or mine residues from being blown off the surface of such dumps or sites, and dispersed or dissipated in the atmosphere, substantially as described.

(C.) 49/14. Martin Deacon (1), William Gore. (2). Improvements in or relating to apparatus for filtering water or other liquids. 30.1.14.

This application relates to improvements in filters wherein the sand or other similar filtering medium is transferred, during operation, from one part to another of the filter, and is washed during transference.

The chief feature of the invention consists in dividing the filtering area into various units, and transferring the sand from one to another through a washing apparatus in the centre of these units.

(C.) 66/14. John Frederick Isbell. Concentrator. 12.2.14.

This patent refers to a concentrator or table somewhat on the lines of the Deister, the Ferraris and other similar tables.

It differs from other concentrating tables, principally in the way the riffles are placed, in the addition of separators having walls continuous with the riffles, ore outlets leading through the table, a cut-off extending across the table comprising means forming a chamber leading to the ore outlet, means forming a second chamber above the first chamber, including a water inlet and sundry other devices meant to perform successfully the operation for which the table has been designed.

(C.) 85/14. William Henry Aghan. Improved process and means for separating precious metals from their ores. 25.2.15.

This application is for a method of cleansing, sand tailings, etc., with a view to rendering the gold contents amenable to mercurial amalgamation. The applicant proposes to digest the sand in a solution of cyanide, sulphuric acid, corrosive sublimate and common salt prior to amalgamation. He also claims (claim 3) the neutralisation of the tailings by lime as a preliminary to treatment with the above solution.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 18/14. Ernst Hjalmar Lagarstrom. Improvements in rock drilling machines operated by compressed air or the like. 8.1.14.

(C.) 21/14. Heinrich Flottmann. Improvements in motors for gig-conveyors. 8.1.14.

(C.) 22/14. Henry Leitner. Improvements in means for controlling electric circuits. 9.1.14.

(C.) 24/14. Paul John Simmen. Improvements in and relating to systems and apparatus for the automatic control of the speed of moving vehicles. 9.1.14.

(C.) 25/14. Frank Clayton Herbert. Device for preventing the fraudulent refilling of bottles. 12.1.14.

(P.) 26/14. James Miners Holman (1), John Leonard Holman (2). Improvements in feeding fluid to hollow rock drills. 12.1.14.

(C.) 27/14. George Hutchison. Improvements in and relating to fencing. 14.1.14.

(P.) 29/14. Andre Leon Felicien Levy-Strauss. Improvements in and relating to rock drilling machines and supporting means therefor. 15.1.14.

(P.) 30/14. James Wilshir (1), The Denver Rock Drill and Machinery Co., Ltd. (2). Improvements in and relating to supports for rock drilling machines. 16.1.14.

(C.) 32/14. Explosions Turbine-studien gesellschaft (1), Robert Kutschinski (2). Improved fuel for internal combustion engines. 17.1.14.

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(P.) 42/14. Nicholas John Kempthorne. An improved design of two stroke internal combustion engine. 26.1.14.

(C.) 44/14. Edward Price Killen. Improvements in or relating to wheels. 27.1.14.

(P.) 45/14. Kenneth Bingham Quinan. Improvements in explosives. 27.1.14.

(C.) 46/14. Mathias Pier. Improvements in or relating to synthetic processes for the production of ammonia from its elements. 29.1.14.

(P.) 47/14. Samuel William Bradbury. Antifriction ball bearings. 29.1.14.

(P.) 48/14. Frank Hill. Improvements in rollers for mine shafts, haulage tracks and the like for supporting, hoisting, hauling or other ropes or cables. 30.1.14.

(C.) 49/14. Martin Deacon (1), William Gore (2). Improvements in or relating to apparatus for filtering water and other liquids. 30 1.14.

(P.) 50/14. Joseph Patrick Cunniff. Safety clutch for vertical shafts. 2.2.14.

(C). 52/14 William Wilson Hamill. Improved liquid fuel spraying devices. 5.2.14.

(C.) 54/14. Arnold Heinrich Wegener. Improvements in nut locks. 5.2.14.

(P.) 55/14. James Miners Holman (1), John Leonard Holman (2). Improvements in coal cutter mountings. 7.2.14.

(P.) 56/14. Arthur Joseph Leguin Fisher (1), William Wilson Lawrie (2), Thomas Herbert Bayldon (3). Improvements in connection with feeding water to hollow rock drills and the like. 7.2 14.

(P.) 57/14. Tom Wrathall. Self coupling trucks. 7.2.14.

(C.) 58/14. Alfred Butter. Process for sterilising milk, cream beverages and other alimentary substances. 9.2.14.

(P.) 59/14. William Noel Bickford Smith. Improvements in electric fuses for blasting purposes. 9.2 14.

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(P.) 67/14. Frank Ondra. Apparatus for concentrating and classifying pulverised substances. 12.2.14.

(C.) 68/14. Dr. Gotthold Issler. Process of and apparatus for the manufacture of explosive charges of fusible explosives. 12.2.14.

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Vol. XIV. MAY, 1914. No. 11

Proceedings
AT
Ordinary General Meeting, May 16th, 1914.

The Ordinary General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, May 16th, 1914, Mr. Alex. Richardson (President) in the chair. There were also present:

27 Members: Prof. G. H. Stanley, Messrs. J. E. Thomas, H. A. White, J. Gray, E. Pam, F. W. Watson, W. R. Dowling (Members of Council), A. S. Allam, R. A. Barry, R. W. Bennett, J. Q. Braidwood, J. M. Dixon, H. N. Dungey, H. W. Gill, W. H. Jane, W. E. John, G. Melvill, H. H. Morrell, P. T. B. Morrisby, E. A. Osterloh, W. S. V. Price, E. Roberts, D. F. Scott, A. L. Spoor, A. N. Stuart, and W. E. Thorpe.

10 Associates and Students: Messrs. C. L. Dewar, F. D. Long, P. C. de Maine, E. J. Moynihan, S. E. Willows Munro, C. H. Olssen, F. J. Pooler, H. Rusden, H. Stadler, and A. Willcox.

6 Visitors and Fred Rowland (Secretary).

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

NEW MEMBERS.

Messrs. J. Gray and F. W. Watson were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman declared that the candidate for membership had been elected as follows:—

Filley, Oliver Dwight, B S A. Co.'s Mines Development Co., Ltd., P. O. Box 627, Bulawayo.

The Secretary announced that since the last meeting of the Society the following Associates and Students had been admitted by the Council:

As Associates—

Davidson, Edmund Gordon, Inca Mining and Rubber Co., Santa Domingo, *via* Tirapata, Peru. General Manager.

Reed, William, El Oro Club, Estado de Mexico, Mexico. Metallurgist.

As Student—

Ruggles-Brice, Harold Ralph, A.R.S.M., Ferreira Deep, Ltd:, P. O. Box 5977, Johannesburg. Post Graduate Student.

General Business.

ANNUAL ELECTION OF OFFICERS AND COUNCIL.

The President: I now have to ask you to appoint scrutineers for the election of officers for the forthcoming year.

Mr. W. R. Dowling proposed, Mr. H. A. White seconded, and it was resolved that the following gentlemen act as scrutineers:—

Messrs. P. T. Morrisby, F. D. Phillips, Sam Newton, F. J. Pooler, E. A. Osterloh, and J. M. Dixon.

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

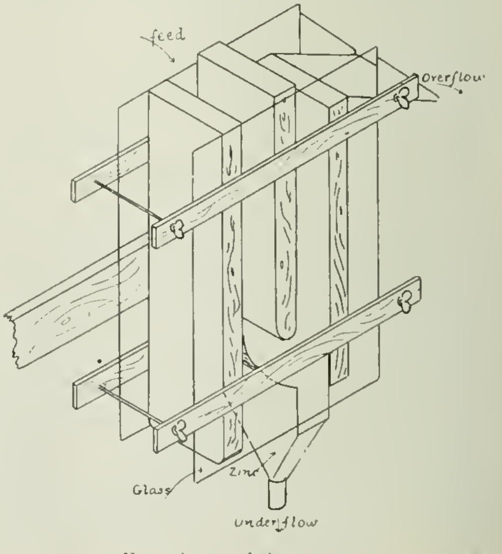
By G. H. Stanley, A.R.S.M., M.Inst.M.M.
(Vice-President).

In connection with this subject it is scarcely necessary to state that a very large amount of experimental work has been done in the past in order to determine such points as the relative rates of fall of various minerals in water, the sizes of equal falling particles, and so on, which is all, of course, perfectly well known to those interested in the subject. Nevertheless, at least with "free-falling" classifiers, the practical results obtained in the working of hydraulic classifiers are nearly always so far from perfect, even if by "perfect" be understood the best obtainable results in the laboratory, that the data experimentally obtained might almost as well be non-existent, so little has it been possible in practice to improve the classified products. That this is so is scarcely surprising when the difference in conditions between the usual laboratory experiment and practical work is considered. In the former case the ratio of water to solid is nearly always much greater than obtains in the latter, and free-falling conditions are generally observed. In many cases in experimental work the pulp is even fed dry, and generally with very little feed water indeed, which constitutes a most important difference from working conditions. The form of the apparatus is usually very different, such points as head available, size of pipes, ratio of hydraulic water to water in pulp feed, etc., not being of moment in a small size laboratory apparatus, and it can be arranged in such a manner that the stream of water is of approximately equal velocity all over its cross section. In practice it seems to be impossible to secure this uniform current, and this is largely responsible for the bad results. On the working scale, from a single classifier there is always in the coarse product a considerable percentage of material that, at the given average rate of flow of the water, should have gone over with the fine, and, similarly, in the fine product there is a variable amount of coarse particles which should have been separated with the coarse. This imperfect classification has obvious disadvantages, particularly when it is desired to separate sand particles of the same kind, i.e., specific gravity, into groups of different sizes. Experiments were therefore undertaken with the object of ascertaining more definitely the causes of this discrepancy between theory and practice, and, if possible, to overcome it: it is hoped that a description of these will be of interest, and although to many who have been concerned with a similar problem there may appear to be nothing novel brought forward, there was little on record when the experiments were commenced.

At first a glass U tube was taken and pulp fed to one limb, the other being shortened: sand particles of all sizes immediately commenced to settle in the bottom of the bend, and continued till the area of cross-section there became so constricted that the consequently increased velocity prevented further accumulation, although most of the sand seemed to settle there, but to be swept along by the current. The result was that instead of a uniform stream of pulp moving round the bend there appeared an upper portion of nearly clear water, and a lower of thick pulp merging into a stationary bed of sand. On passing into the rising limb heavier particles fell back to the bottom, to be again caught by the current and lifted, but obviously if an outlet were to be provided the outflow would consist partly of grains of all sizes which have settled immediately, mixed with larger grains falling from the rising limb; this resulting in the outflow containing a larger percentage of large particles than was present in the pulp fed, or, in other words, being coarser on the average. Thus an imperfect classification was effected and the action of most classifiers typified.

From this it seemed that if means could be devised to prevent what may be termed this "first settlement" of all grades of sand, or to keep all particles in suspension till, in the rising current, the larger began to fall out, a step forward would have been attained. On the other hand, if it is attempted to prevent this by causing a sharp change in the direction of flow of the pulp from downwards to upwards, eddy currents are set up which also cause deposition of all sizes of particles from coarse to fine. It therefore appeared to be preferable to prevent a sudden change of direction, but to obviate in some manner the discharge of the mixed "first settled" particles, and to arrange matters so that only those sorted out in the rising current should be discharged.

In order to be able to observe what took



Experimental Apparatus.

place in the apparatus, it was constructed of two sheets of plate-glass, between which pieces of paraffined wood and strips of zinc were clamped to form the outline of the classifier. Plasticine was used to make all joints watertight. The clamping was very simply effected by means of two wooden bars on each side projecting beyond the glass and held together by bolts as shown in the figure, and similarly fastened to a horizontal bar as a support. With this apparatus many different arrangements were investigated.

The arrangement, Fig. 1, was one of the earliest, but it was found that the sand which settled in the bottom of the U simply accumulated till it crept over the edge at (a), and, being then in stationary water, fell into the receiver at once. To avoid this, a pipe supplying hydraulic water was connected at (b): this proved to be only a palliative, as it was impossible to obtain a uniform upward stream, and much fine sand still fell over at (a).

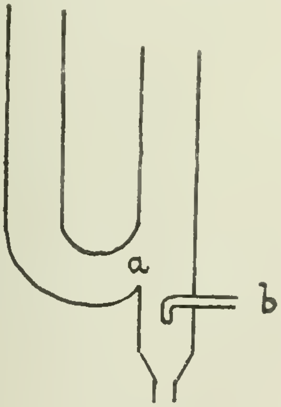


Figure 1.

Three bars, *c*, *d*, *e*, Fig. 2, were therefore placed across, at the foot of the rising column, so proportioned that the velocity of the current would be the same in the slots thus formed as in the rising limb, with the object of more nearly equalising the flow. However, the settled sand completely filled up the two outer slots and fell unhindered through them, while the water passed up through those in the middle.

Following this, forms shown in Figs. 3 and 4 were arranged in which a partition was provided to guide the descending "first settled" sand past the outlet and into the rising stream; the same difficulty occurred in both cases, *i.e.*, flowing over of the settled sand at the end of the guide, and more particularly settlement of ungraded sand on the slope *a*, *b*, which flowed back as a thick pulp towards the outlet slot, hugging the surface all the way, and escaped without being subjected to the washing action of the hydraulic water coming through the slot, as the stuff falling over the edge was, every particle being to some extent detached from its neighbour as it went over and having some opportunity of being lifted if small enough.

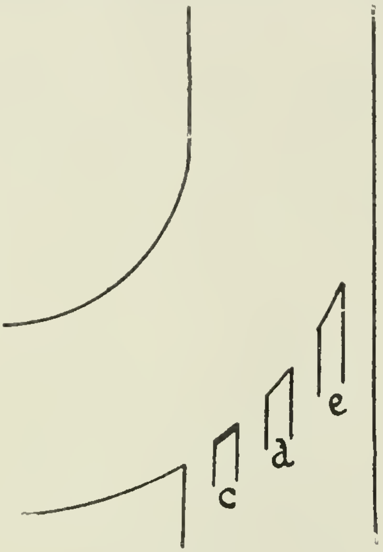


Figure 2.

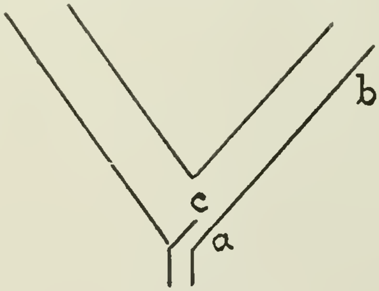


Figure 3.

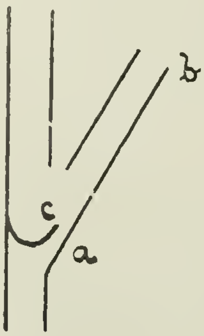


Figure 4.

The discharge slot was then modified into the shape shown in Fig. 5, which seemed to obviate the passage of thick pulp. It is, however, apparent that once a particle passes through the slot into the lower chamber, it is subject to the action of a far lower water velocity than when passing through the slot, and consequently a small particle, once through, is not rejected, and the classification is still very imperfect. Anotner modification consisted in extending the edges of the slot downwards by vertical plates as shown by dotted lines at (*b*), Fig. 5, so as to form a short vertical sorting column. This did not improve matters, as shown by the grading analyses, thick pulp tending to hug the sides and water tending to the middle, and was therefore discarded.

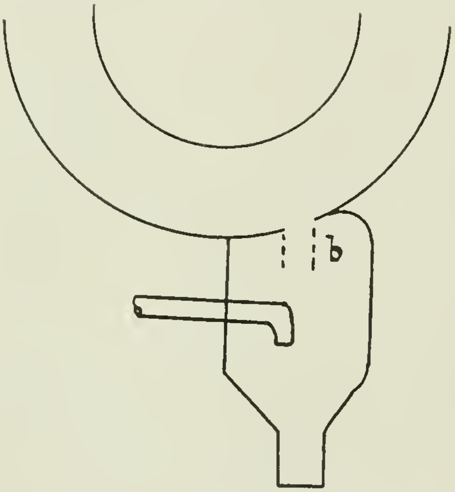


FIGURE 5.

It may be mentioned here that in the Richards vortex classifier this difficulty, in a cylindrical sorting column, is largely obviated by giving the hydraulic water a rapid swirling or rotatory movement, which sweeps it away from the sides.

Up to this point the ratio of water to solid was very large, and subsequently an attempt was made to watch the action when the ratio was brought more nearly into accord with practical conditions, the particular ratio aimed at being 10 : 1, though this is admittedly large (but includes hydraulic water), and the standard of work being as before the separation of the solids in the pulp into + 60 and —60 particles. It was found that on increasing the amount of sand fed, the settled sand accumulated above the discharge slot considerably faster than it could drop freely through it, till at length an accumulation existed in a state of more or less violent agitation, the sand falling through in a state of thick pulp in one place and water coming up in another, and often what may be described as "clots" of sand fell through the slot, first at one end, then at the other. No matter what form of slot was used, or what means were adopted to try and prevent this accumulation and manner of passing through the slot, with any reasonable ratio of water : solid, it occurred sooner or later.

Now, on account of the reduction of size of the water passage surrounding each particle, resulting from the approximation of the particles, the upward velocity should have been increased and a given amount of water should drop a much coarser product: on the other hand, a large proportion of the particles of all sizes, under the circumstances, actually agglomerate weakly together and drop to some extent as a whole, carrying down enclosed small particles, so that the actual classification effected was not improved. Obviously, to ensure "free-falling" conditions through the slot it would need to be increased in size, this, again, would need more water to maintain the required upward velocity, thus disturbing the ratio water : ore. It seems plain, then, that there is a maximum weight of sand which can fall per unit area of cross-section of rising current, while free-falling conditions are maintained, and this appears to be so small that although the actual sorting or classification in the body of the classifier may be effected under such conditions, since the passage leading to the discharge opening is necessarily much smaller, the weight of sand per unit of water must necessarily (in ordinary cases) be much greater and free-falling no longer obtains. What happens in any such passage through which upward current is flowing is that most of the material slips down at the sides and a violent current of water passes up, mainly at the centre; but in order to obtain a satisfactory coarse product it is always necessary to employ an upward velocity greatly in excess of that theoretically required from a knowledge of the rates of fall in water of various sized particles. The quantity of water is also greater, although the cross-sectional area is restricted by the sand. The reason being, obviously, to secure sufficient velocity at the sides to prevent the fall of the undesired small particles. This action can be clearly seen in the model with only a comparatively few grains of sand charged, and the velocity such that they are readily carried upward by the full velocity: yet sooner or later they get to the side by settling and sliding down, and there pass through the slot. The Richards vortex classifier, and others employing a helical current, as already stated, certainly seem to secure better mixing and agitation of the pulp in the sorting column than in the more common forms. As the conditions were constantly changing, stable hindered-settling did not obtain, and, besides, the classifier was not constructed for this condition. Up to this point the observations refer to classifiers which are not "overloaded," and from which the overflow product is of comparatively satisfactory fineness. When overloading takes place, it follows that a larger quantity of coarse particles, able to settle under the given conditions, and too heavy to be carried forward, are fed to the classifier per unit time: these settle into a thick pulp, restricting by its presence the area of water passage, and therefore increasing the water velocity. In consequence, the sand settling in the classifier is of a coarser grade on the average, but this is accompanied necessarily by a large excess of coarse material in the overflow. Still, in general, it may be stated that, within limits, the greater the quantity fed to a classifier, the better will be the character of the underflow up to the point at which, owing to the sand depositing faster than it can be discharged, either by the underflow or with the overflow, the classifier chokes up. It was therefore seen to be impossible in practice to obtain by means of ordinary hydraulic classifiers—using the term to include both those with and without separate "hydraulic" water supply—a satisfactory separation of coarse and fine particles by one operation. If the underflow is free, or nearly so, from fine particles, then the overflow will contain too much coarse, and *vice versa*, if the overflow is free from coarse particles the underflow will contain an altogether unsatisfactory proportion of fine.

If the action of some of the known types of classifiers be examined in the light of this knowledge of the behaviour of the pulp, the method of action and reasons for the character of the product become apparent.

In an ordinary launder there is always a layer of sand of greater or less thickness travelling along the bottom more slowly than the stream as a whole. When the stream enters an ordinary side-fed pyramidal spitzkasten, this layer largely continues to travel down the side to the bottom; other particles pursue paths which are resultants of the horizontal velocity they possessed at the time of entering and the vertical velocity due to the acceleration of gravity, and also, of course, affected by the motion of the carrying water. Some of them deposit below the outlet, on to the side, down which they slide to the discharge; others fall so little, being small, that they remain in the stream and are carried out with the outflow. The horizontal velocity of the stream is diminished by increasing the area of cross section by widening the stream, thus helping settlement; but the velocity always tends to be greater in the centre, and various eddies form which also cause deposition of considerable fine stuff with the coarse. Once this fine stuff settles it becomes entangled with the coarse and is carried out with it. The use of hydraulic water in any reasonable amount is of little benefit except for removing the very finest or slimy portion of the underflow, for the area of cross section increases so rapidly upwards that even if the pulp is washed free from fine in the centre, where most of the water will pass, it rapidly settles again on the sides. If a vertical baffle is introduced across the stream, converting the spitzkast into a crude spitzlutte, some improvement is effected, as the hitherto comparatively quiescent water at the bottom is kept in motion, and the settlement of fine at least hindered. The area of cross section, however, and consequent velocity is not constant, and a mixture of all sizes settles on the sides and runs down to the outlet. The same remarks with regard to hydraulic water apply as before. Even in a properly designed spitzlutte, where the area of cross section is approximately constant, and the separation is supposed to be effected in a rising current of water of known velocity and of the same velocity all over the section, the velocity is in reality greater at the centre than at the sides, and changes of direction cause eddies, resulting again in the deposition of fine with the coarse and a result, although improved, still very imperfect.

Passing over the various pocket classifiers simply with the remark that the same conditions hold in varying degree, the large cone classifiers so widely used at the present day in the metallurgy of gold, may be briefly examined. The feed to these is central, generally downward, through a short vertical pipe opening on to a horizontal baffle, and the outflow peripheral. The pulp flows out from the centre towards the periphery, dropping the coarser particles on the way. They are usually operated with a thick underflow, the cone being nearly full of settled sand and consequently the classification is effected entirely above the top of this, the stream from which settlement takes place passing in a direction more nearly horizontal than vertical and of no great depth. The conditions, therefore, resemble those in a launder in which the width is rapidly increased, and depth and velocity of stream therefore reduced. By suitably regulating the feed and discharge the velocity is so controlled that in passing over the edge it is insufficient to carry over particles above a desired size. But at the same time particles of all sizes settle on the top surface of the settled sand, larger ones of course predominating, and the finer ones so settling are largely covered and prevented from rising again by the larger, in consequence, quite a large proportion of fine remains and is discharged with the underflow, rendering the separation very imperfect. Obviously, when the pulp fed is the product of a modern coarse crushing battery, there is such a great divergence between the size of the largest particles, and that of the particles small enough for cyaniding (for the separation of which it is used), and such a large percentage of these large particles present, that the amount of fine remaining with the coarse is much less in proportion than would be the case if the battery were crushing say through a 1,600 screen, and consequently the grading analysis of the underflow does not appear so bad as it undoubtedly would be in the latter case. Under the conditions in which they are employed it would be difficult to specify a more suitable appliance, on other grounds as well, such as large capacity and low per cent. of water in underflow, but they would scarcely be suitable for close classification in a dressing plant. When used for the subsequent separation of sand and slime, it appears to be necessary to wash the first underflow to remove slime which has settled with it; or else to use hydraulic water in the cone, as for instance in the Brazier cone, and discard the thick underflow method of operation.

In all these cases it seems to be so difficult, if not impossible, to approximate to laboratory "free falling" conditions that it would seem advisable to aim rather at the adaptation of "hindered-settling" conditions. From the standpoint of the ratio water: ore alone, working conditions can be much more nearly like those under which hindered settling can be studied in the laboratory and, in other ways, the practical conditions can be brought more nearly to resemble those in the laboratory and, therefore, practical results might be expected also to be more in line. This anticipation would appear to be realised by the success of the Richards pulsator classifier, the products of which on the large scale are nearly, if not quite, as good as in the laboratory. On the other hand, the underflow product, on account of the "hindered settling" ratios being much wider than "free falling" ratios, may contain fine heavy mineral of a considerably smaller size than would be obtained under ideal "free falling" conditions accompanying the gangue mineral, this if subsequent concentration is to be employed, may be a disadvantage, as on theoretical grounds separation then becomes more difficult. When, however, the object is merely to classify by size only one mineral in place of sifting, the product is probably better than from any free falling classifier of expected similar performance. Sifting, of course, is capable of dividing sand closely into different sizes, though this method has fairly well-known practical drawbacks, especially with the finer sizes, the chief, apart from cost of upkeep and power, being the small capacity if a really good screening is to be effected, which seems to limit its use. This is largely due to the obvious fact that the particles which cannot pass the screen remain upon it until discharged in some way, thus hindering the passage of the finer stuff, tending in fact to prevent it reaching the screen at all and consequently allowing it to be discharged to a considerable extent along with the coarse. Very little consideration is required to perceive that it is therefore much easier to sift a little coarse from a lot of fine than *vice versa*, and it therefore occurred to the writer that it might be possible to remove the bulk of the coarse stuff from the pulp, by a hydraulic classifier worked in such a way as to yield a good coarse product and then to sift out from the overflow the coarse stuff which would necessarily escape with it. This could be done by existing types of screens, but their capacity is comparatively small for the reasons stated *i.e.*, clogging of the screen by the pulp itself, quite apart from choking due to particles of rock, etc., becoming wedged in the apertures; but it seemed feasible to actually attach the screen to the classifier outlet, so that all overflow would have to pass through it, and to arrange it either vertically or sloping backwards so that the larger particles which it stopped would fall off and drop again to the classifier underflow. This turned out to be a vain hope, the large particles soon accumulating on the back surface of the screen, being held there by the water pressure, and choking it completely. Various means were then tried of agitating the screen so as to keep it clear and it was finally ascertained that the most effective way was to give it a vertical reciprocating motion in its own plane, of small amplitude but high frequency, and to cause it to bump at the end of the down stroke. This was secured by mounting the screen in a frame running in guides and lifting by a cam. This arrangement was quite effective, but considerable mechanical difficulties had to be overcome in so mounting the screen as to prevent pulp passing round or underneath it: this was finally secured by attaching a sheet rubber edging all round the edge of the frame and also to the body of the classifier.

While experimenting with the screen some dozen or so modifications of the body form were also tried till finally the form shown in Fig. 6 was adopted. During these tests both feeds and products were weighed and graded, and water used, both feed and hydraulic, also determined, upwards of 150 trials being made in all, so that progress was readily judged. The feeds averaged 30 +60, and with the final form of apparatus the coarse product contained about 85% −60 and the overflow (through the screen) 100% —60. A new model was then constructed of galvanised iron but with a 50 mesh screen instead of 60, as it was thought that the rapid movement of the screen would prevent a particle which was just —50 from passing, while probably allowing —60 to pass. The products obtained from a similar feed averaged about 91% to 96% +60 in the underflow but with 4% to 8% −60 in the overflow.

It was then considered advisable to construct a full size experimental machine and accordingly this was erected, by permission of the Robinson G. M. Co. alongside their old sand dump. The construction was somewhat crude, but, nevertheless, a good classification was obtained till an unexpected difficulty appeared, *i.e.* the increased head of water inside the screen developed pressure sufficient to pack the sand particles so tightly on the back that the bump was insufficient to detach them and consequently the screen choked up. This was overcome by equalising the pressure back and front of the screen by enclosing the front by an iron plate attached to the side and bottom of the classifier—leaving the screen free to move—forming thus a casing with an adjustable outlet so that the fine pulp outflow was sufficiently restricted to keep this casing full to very nearly the same level in front of the screen as inside, behind it. This required a little attention, but quite obviated the difficulty. It is shown diagrammatically in Fig. 7. After a little practice it was found quite possible to run the classifier with a thick underflow, the products being very satisfactory and the capacity about 3 tons per hour with a screen (40

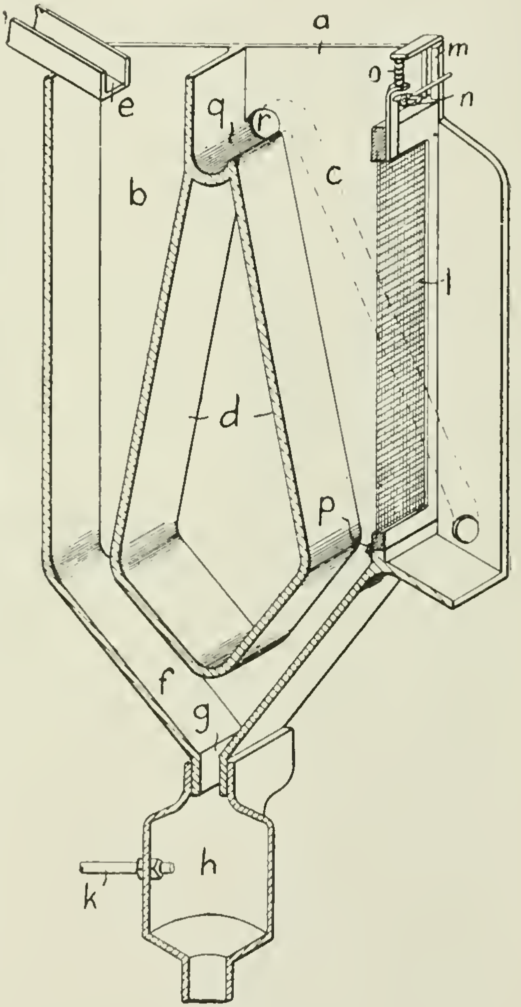


FIGURE 6.

linear mesh) of size about 4 ft. sq. Many variations of conditions were tried and it was found possible to secure from a feed containing about 30% to 35% + 60, an underflow with 77% to 80% + 60, and overflow with 2% to 6% + 60.

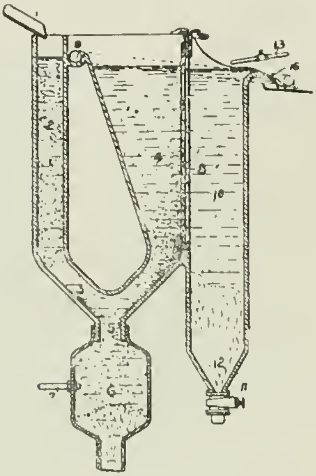


Figure 7.

Later, another form of the apparatus was constructed, embodying a suggestion of the United Engineering Company, which obviated the employment of a rubber edging to the screens. This was done by mounting the screens on each side of a wooden frame held in a vertical jigging frame of channel iron, so forming a deep flat box with a screen back and front, which was suspended in the pulp in the classifier and agitated as before by cams. The discharge from this box was effected through a short length of rubber hose which made connection between the shaking frame and the classifier side, the outlet being adjustable. This type unfortunately ran for a very short while on account of financial difficulties and extensive experimenting was not possible, but the results were encouraging, and doubtless with time and funds to correct faulty mechanical details there seems no reason why the laboratory results should not have been at least closely approached.

That there is a demand for good classification was amply demonstrated by the receipt of enquiries from several mining districts, some details having been given publicity in the mining press. Published gradings also indicate the same, and it is with the hope that interest in this subject may be revived and a useful discussion ensue, that these notes are presented.

In conclusion, it is obvious that this experimental work could not have been carried out single-handed. Mr. M. T. Murray was the chief co-operator, and I have to acknowledge with many thanks his valuable assistance. I have also to thank Messrs. Moon, Buckland, and Estill.

Mr. H. A. White *(Vice-President)*: I have to propose a very hearty vote of thanks to Prof. Stanley for his very interesting paper. It brings before us in a very clear way the origin of the difficulties we used to have in the old times in working spitzkasten; but it is obvious to most of us that, in practice, with the standard cones which are now so usual in the tube mill classifications, and also for separating sand and slime, a great many of these difficulties are removed. Of course, it is perfectly true the results we are now getting are not theoretically perfect, and I have no doubt experiments made on the lines of this paper, as indicated by Prof. Stanley, will enable us to improve the work we are actually doing: but I must say I do not think there is any very great margin for improvement at the present time. I would draw attention to the fact that on several mines the finest portion of the sand residue is now the richest product discarded, and this illustrates the effect of the tube mill classification and return system in giving preferential treatment to the pyritic portion of the ore, which requires and repays extra comminution. This paper contains a hint or two on this very point. I therefore have pleasure in proposing a very hearty vote of thanks to Prof. Stanley for his paper.

The vote of thanks was carried by acclamation.

The President: We are much indebted to Prof. Stanley for going into this subject of classification so thoroughly, and for so kindly putting all his data on record.

THE EFFECT OF CHARCOAL IN GOLD-BEARING CYANIDE SOLUTIONS WITH REFERENCE TO THE PRECIPITATION OF GOLD.

(Read at November Meeting.)

By Mr. Morris Green (Associate).

REPLY TO DISCUSSION.

Mr. M. Green *(Associate)*: Dr. A. L. Simon's demonstration confirms, in an interesting manner, the author's conclusion that graphite does not precipitate gold. The presence of the two tubes here (Johannesburg) would have added greatly to the interest of Dr. Simon's contribution.

Mr. H. K. Pickard appears to have misinterpreted the object of the investigation. The author nowhere in his paper proposed "to employ charcoal precipitation in a modern cyanide plant," or charcoal precipitation in any manner. The author is grateful to Mr. H. N. G. Cobbe for immediately having pointed out to Mr. Pickard in what directions the investigation was of importance. Mr. Paul T. Brühl's remarks indicate another instance in which it becomes important to determine the effect of carbonaceous matter. In fact, it was the seemingly contrary effect of "graphite" in a certain ore that led to the carrying out of the investigation.

Discussing further Mr. Pickard's remarks, the author admits omitting, in order to limit the length of the paper, certain experimental details. But this omission in no way affects the ultimate results given. If twice or three times the amount of charcoal be employed, *similar* results, in each series, are obtained, the *absolute* amounts of gold precipitated in the different series being different. It is unimportant to know the percentage of gold in the charcoal carrying 5 mgm. of precipitated gold, since different forms of charcoal are capable of precipitating, weight for weight, different amounts of gold. It should have been mentioned, however, that the charcoal precipitated gold from a solution in which there was excess of gold, or, in other words, the amount of charcoal employed was not sufficient to precipitate all the gold from solution. A solution poor in gold *can* be precipitated by charcoal. Mr. Pickard's deduction, therefore, appears to be incorrect, the author offering, instead, the explanation that the charcoal was no longer an effective precipitant, and that the cyanide solution redissolved the precipitated gold. That dissolution of the precipitated gold proceeds so slowly probably may be partially accounted for by the fact that the gold is precipitated in the metallic state.

Mr. Pickard's figure 1·7% appears to be incorrect. The charcoal finally carries about 1% of gold [39 of 0.57 = 0.2223. Weight of charcoal = 0·22 gm.; ∴ gold in charcoal = 1% approximately]. Lowles' figure (0.5) for the amount of gold carried by charcoal in practical precipitation work need not necessarily agree with the author's number. The amount of gold precipitated depends upon the nature of the charcoal employed, besides upon other conditions. Solutions in practice, too, are usually impure, whereas in the author's experiments pure solutions were used.

In reply to Mr. C. O. Bannister's remarks on the precipitation of gold chloride, the author would point out that the case instanced and the precipitation of aurocyanide are dissimilar in that the gold chlorides are very unstable, whereas the aurocyanide is characterised by its stability. Aurocyanide is not decomposed very rapidly in the presence of charcoal

Mr. H. N. G. Cobbe's remarks have been alluded to previously. It only remains to add that many old mines have tailing dumps that are at present unavailable for cyanide treatment on account of the presence of charcoal and other carbonaceous refuse.

Mr. Paul T. Brühl's remarks about his personal experience of a "graphitic" ore (inverted commas are used advisedly, since the term graphite is frequently used to describe carbonaceous matter even when no true graphite is present) are of extreme interest. That the form of "graphite" he refers to is a precipitating agent is quite likely, but the power to precipitate is probably due to occluded gases. Graphite, when no occluded reducing gases are present, does not precipitate gold; and ordinary mineral graphite does not occlude reducing gases. Mr. Brühl's concluding remarks may suggest to Mr. C. O. Bannister that aurocyanide is not rapidly decomposed.

With our present knowledge of the condition of the occluded gases in charcoal, it is impossible satisfactorily to answer Mr. A. F. Crosse's question: "If oxygen and carbon monoxide are present together in this active condition, should they act act on one another and form carbon dioxide?" Of the nature of "this active condition" the author confessed himself to be ignorant, and then threw out a suggestion for a fresh investigation that might elucidate the problem. In this connection, the author would further point out that in dealing with the three systems formed by the two phases, gas—solid, three chief cases call for mention*

1. The gas is not absorbed by the solid, but when the pressure reaches a certain value, combination of the two components can result.

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* "The Phase Rule and its Applications," by A. Findlay, p. 175, 1904 ed.

2. The gas may be absorbed and may also form a compound.
3. Absorption of gas occurs, but at a certain concentration the solid solution can separate into two immiscible solid solutions.

The above is such a well-known text-book classification that it was not referred to in the paper, but is inserted here since the conception of the possible modes of existence of the gases appears occasionally to be somewhat vague. An investigation on the above basis was what the author implied in his paper. As further comment on Mr. A. F. Crosse's question, the analogous case of the absorption of hydrogen by palladium may be quoted. Although this still presents certain unexplained phenomena, it is well known that the hydrogen when thus occluded is capable of bringing about certain chemical changes which ordinary hydrogen is unable to effect, *e.g.*, the reduction of iron from the ferric to the ferrous condition.

Mr. F. J. Pooler, near the commencement of his criticism, takes objection to a certain statement which the author never made. He says that the author regards "graphite" as "carbon" and proceeds thus: "The structures of graphite and carbon are so different that the explanation does not seem quite conclusive." To the first charge, the author pleads guilty. Does Mr. Pooler imply that graphite does *not* consist, chemically, of carbon? From his above remark that "the structures of graphite and carbon are so different" it would appear that he does *not* regard graphite as carbon! "Since liquefied gases are, on the whole, less chemically active than the gases themselves" and because "some text-books suggest the possibility of absorbed gases being liquefied in the pores" (the author quotes Mr. Pooler), Mr. Pooler seems to conclude* that the carbon itself is the active agent, and that it acts catalytically. In reply to these remarks, the author would refer Mr. Pooler to the classification outlined above relative to the possible modes of existence of occluded gases. Further, old charcoal, which exhibits exactly the same structure as freshly ignited charcoal, is a very poor precipitating agent. Charcoal, too, exhausted in the vacuum pump, the author found to be inefficient as a precipitating agent. Why do these forms of charcoal not act catalytically? Mr. Pooler's explanation "of a catalytic action of the 'fine' carbon," appears to the author to be adopted to conceal a candid admission that the reactions are not understood. To conclude that the carbon acts catalytically is certainly not the logical outcome of any of the experiments. That carbon, *per se*, does not precipitate gold from aurocyanide solutions was known long before the present investigation was undertaken.

Mr. Pooler's remarks about the transmutation of elements and the possibility of errors arising from the use of silica tubes in the experiments, seem to the author to be irrelevant. The analysis of the gas obtained on exhausting graphite was given, and showed no reducing gases. No reducing gases were, therefore, derived from the silica.

In reply to the question as to whether the precipitating power of charcoal had been tried on solutions other than aurocyanide solutions, the author would refer Mr. Pooler to the reference he gave in his paper, namely, "Gels in Relation to Ore Deposition," Trans. Inst. Min. & Met., xxi. In addition, other work has been performed on the decomposition of gold chloride solutions by charcoal; but so far as the author is aware, merely the effects were noted, while no attempt was made experimentally to determine the cause of the reaction. Besides, the two cases (gold chlorides and aurocyanide) are obviously different, since the gold chlorides are extremely unstable.

Mr. Pooler states that "the large percentages of nitrogen found in Tables IV. and V. do not confirm the usual text-book statements that oxygen is more easily absorbed than nitrogen, and quotes from Mellor, absorption co-efficients determined by Hunter. These remarks demonstrate that Mr. Pooler has rather failed to grasp the situation. The author's figures denote the proportions in which the gases were found in the charcoal after the distillation of the wood in closed retorts. Hunter's absorption number for any gas is the volume of gas absorbed by a given weight of charcoal after exhausting the charcoal and then allowing it to communicate with an excess of the particular gas. The absorption co-efficients are the relative figures for the series of gases. In his paper the author presented the results of his experiments, together with deductions he made therefrom. Mr. Pooler further contends that "it is evident that the numbers given in Tables II., III., and IV. are of little value in this research as the carbon lost little, if any, of its precipitating power."

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* The condition (activity or otherwise) of liquefied carbon monoxide at ordinary temperatures is unknown.

To the author this is not so evident, and he considers, as also did Mr. C. O. Bannister, that it is important to appreciate the difference in the precipitating power of the carbon monoxide which is loosely held by the charcoal and that which is not loosely held; or, in Mr. Bannister's words, "what was merely absorbed in the pores as against what was absorbed by the charcoal.

Since the gases evolved under different conditions (*e.g.*, time of exposure of the charcoal, pressure, temperature) from any one sample of charcoal are of different volumetric composition, nothing can be gained by comparing two equal volumes obtained under entirely different conditions. By a general comparison of the various tables of analyses given, it will be seen that the total oxygen content of a given volume is not constant. Mr. Pooler's particular example, therefore, can only comprise two cases showing an accidental agreement.

In concluding his reply, the author is pleased to be able to agree on one point with Mr. Pooler when the latter says that he has "an idea that the condition of the carbon has a lot to do with the reaction." The author would point out that he had already been at great pains to explain the difference in behaviour between graphite and carbon, and Mr. Pooler's conclusion was, therefore, anticipated.

ROCK TEMPERATURES.

By E. J. Moynihan (*Associate*).

Some months back, in a contribution to the discussion on Mr. Blenkinsop's paper, read here in September last, on "The Ventilation of the Mines of the Rand," I pointed out that the question of the determination of the temperature gradient in rock is in a very unsatisfactory state. I said that I did not question the good faith of the observations, but that certain most important considerations had been overlooked, and that I hoped to deal with them before this Society at an early date. I am later in dealing with the subject than I expected to be. Industrial and political water has flowed under the bridges since last December, in such a torrent that many of us have found it difficult to do the dispassionate reflection that is necessary in scientific work. That is my case, and that is my apology for not having presented these notes before now.

My main purpose is to show that temperatures, taken by means of drill-holes of moderate depth in stopes or development faces, are often far from the original rock temperatures; that therefore the temperatures so got often give rock temperature gradients that are far from the original; and some of the reasons why this is so. From the practical and historical standpoint, this paper is therefore largely of a critical character. I shall also attempt to forecast the minimum air-gradient to be expected without mechanical refrigeration.

The first contention is easily proved by a reference to the diagram of rock temperatures contained in Messrs. Whitehouse and Wotherspoon's paper read here in May, 1911.* The temperatures there given from drill-holes are such, that by choosing different pairs of them, 1,000 ft. or so apart vertically, it would be possible to deduce either a positive gradient (temperature increasing in depth) or a negative gradient. One can show a wide difference in temperature between points on the same level, or practically no difference in temperature between points at a vertical distance of 1,500 ft. apart—in short, almost any gradient can be shown if one chooses the two reference points suitably. By continuing the line obtained from different pairs of such temperatures till it cuts the surface level, one can deduce any surface temperatures from about 51° to 71° F.—a difference of 20°. This is more than the difference for 5,000 ft. vertical, according to the gradient deduced by Messrs. Whitehouse & Wotherspoon.

The actual temperature of the undisturbed rock within a few feet of the surface under normal conditions is, and must be, within a degree or so, the mean surface temperature at that place as taken over an average of a fair number of seasons. The subject of the variation of solar radiation is so new, and observations in that connection so few, that we do not know for how many seasons one must observe to get a fairly correct mean. Nor do we know whether the common custom of taking half the sum of the maximum and minimum air temperatures gives a very truthful "mean." It seems quite possible that a better and quicker way of getting the mean temperatures of any place would be to drill a borehole, say, 1,000 ft. in depth, in a dry spot, and in hard rock, to put an electrical resistance thermometer at the bottom, fill the hole with cement to within 500 ft. of the top, and then put in a second thermometer and cement the hole up to the top. On the assumption that the *mean* gradient is uniform, one arrives at a certain mean temperature for the surface, which is probably as near the truth as the result of any air temperatures yet taken. All rock temperature gradients should practically pass through this temperature at the surface. However, failing this, or some other reliable method, we must make the best of what we have, and that is the mean air temperature of the place. Any gradient based on the temperature of the air, or rock walls, in shafts and near the surface is unreliable. The shaft may be up-cast, when the temperature at the top is usually too high: or down-cast, when it is often too low. The most casual reference to the diagram in Messrs. Whitehouse & Wotherspoon's paper, or to the figures contained in the last Government Mining Engineers' report serves to prove this. The following are the official mean annual surface temperatures for 1910 for the Transvaal:—

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* See this *Journal*, Vol. XI., May, 1911, p. 535.

Observatory, Johannesburg ...	58·4° F
Joubert Park, Johannesburg ...	59·9° F
Sunnyside, Johannesburg ...	58·7° F
Zuurbekom	59·4° F
Vereeniging	60·8° F

We are therefore fairly safe in taking the mean temperature of the Rand, for the purpose of these calculations, at 60° F. It is perhaps a little too high, and therefore tends to flatten the gradient calculated from temperatures read at depths. Take, as an instance of what I have already said, the six temperatures from Marriott's mine for 500 ft. below the surface. The highest and lowest temperatures for the six are 68° F. and 56° F respectively, a difference of 12°. This is practically the same difference as that between the highest Marriott reading for a depth of 500 ft. and his reading at 3,300 ft. When one gets the same variation in drill-hole temperatures, all taken at one level, as is obtained between two holes 2,800 ft. vertically apart, it is clear that temperature-gradients plotted from such observations must be worthless guides to the original gradient of the rock. The errors of observation may be as great as the deduced difference of temperature in 2,800 ft. This is actually and obviously the case with the Marriott observations plotted by Messrs. Whitehouse & Wotherspoon.

Messrs. Whitehouse & Wotherspoon's own observations show errors of much the same kind. Two of their readings show a difference of 3° at practically the same level of 3,200 ft. This is a greater difference than is plotted between the hotter of those at 3,200 ft. and the temperature given at the 3,800 ft. level; and is also greater than the difference between the cooler of them and the lowest temperature in the Village Deep. The difference of temperature shown in the Village Deep and Turf shaft readings, for a difference of level of 1,600 ft., is little more than twice the plotted difference of temperature for two Village Deep readings taken at the same level. Let any one make a tracing of the diagram, omitting the gradient lines drawn by Messrs. Whitehouse and Wotherspoon, and ask himself, or any one, "What curve would show fairly the law connecting the points on the squared paper?" Any one used to plotting curves would at once say that the errors of observation were so big that he must "give it up." There is not even any evidence from the points plotted that the law is a straight line. The low gradient arrived at by Messrs. Whitehouse & Wotherspoon is obtained by assuming (by implication) a surface temperature about 5° too high, and then plotting the gradient so as to pass *below* most of the points plotted from their deeper observations, and through *none* of the points referred to in their text. They say, in the first column of their text, that the temperature of the rock varies from 65° F. at 200 ft. to 98° F. at 7,000 ft. The gradient line that they draw on their curve passes through the *surface* at *above* 65°, through 200 ft. at 66°, and through 7,000 ft. at less than 93°. They say it is apparent "from the chart" thus drawn, that the "estimated rate of mean temperature rise is now represented to be 1° for 253·9 ft." It is not "apparent" at all: the gradient on the chart might as justifiably be drawn in a dozen different ways: the figures given in the text, by the authors themselves, in no wise agree with the gradient line plotted on the chart: their text figures imply a gradient of 33° in 6,800 ft., or closely 1° for 206 ft.: and it is obvious therefore that the general inductions from the chart are of little scientific value.

If a careful examination of their other figures is made this conclusion is further borne out. The paper is accompanied by a schedule of "psychrometrical" observations showing dry and wet bulb temperatures, at various times and places, before and after the installation of the fan on the Village Deep. They deduce from these observations that the installation of the fan materially reduced the air temperature and humidity in the mine. The reductions in both cases may be allowed to be facts: but there is no evidence that they were due to the fan. The average reduction in the temperature according to the authors, was from 2° to 3° F. at distances of 1,000 ft. from the shaft stations; and the reduction in humidity was from 2% to 6%. The readings compared, in order to find out the effect of the fan, were taken in January and February (before the fan was installed) and in May (after the fan was installed). Now, the temperature and humidity in the mine depend, in a considerable degree, on the temperature and humidity of the air at the surface. The surface figures are printed in their paper. These show quite good cause for the reduction in temperature and humidity without the existence of the fan at all. The maximum and minimum surface dry-bulb temperatures, for January and February, were 80·2° F. and 57·2° F.; for May, 71·6° F. and 38·1° F. The surface humidities, for January and February, were never below 48.2%; in May, 15 out of 21 readings were materially below this figure. From the Transvaal Meteorological Report for 1910 it can be seen that the mean air temperature for January was about 64°, for February 63°, and for May 54°. This is quite enough to account for a corresponding seasonal and average difference of 2° to 3° in the Village Deep mine. The mean surface humidity in January (for six seasons) was 73%, in February 76%, and in May 52%. A difference of this order fully explains a diminution of 2% to 6% in underground humidity.

In dealing with the effect of surface temperatures on the temperature of a mine, it must be borne in mind that though surface seasonal variations in temperature from one month to another produce a considerable effect on the mine workings, large surface variations in two or three days produce very little effect, owing to the walls of the mine acting as a heat reservoir. Therefore comparisons of underground and overground temperatures at one particular instant in a large and deep mine are practically of no value whatever. Even in an ordinary office building it will take several days for the internal temperature to adjust itself to a sudden change of 10° in the mean external temperature. In all that follows, it is therefore to be understood that I am dealing with mean temperatures taken over a period of something like a month or more.

The authors, in taking the temperatures at the bottom of drill holes of moderate depth as the basis of the temperature gradient of the rock, followed precedent. How erroneous their general induction is can be seen from one of the statements in their own paper, in which it is pointed out that the average temperatures in these holes were 3·2° F. less *after* the fan was installed than *before* it was in operation. They say nothing as to the time of year in which these observations were taken. One naturally presumes that the temperatures of the drill holes (5 ft. deep) were taken about the same time as the other temperatures tabulated in their paper. If this is so, the seasonal difference is quite enough to account for the reduction claimed for the fan, because (according to the 1910 Report of the Transvaal Meteorological Department) the earth temperature varies, at Johannesburg, between January and May, about 5·5° F. at a depth of 4 ft. At a depth of 3 ft. 8½ ins. at Pretoria the corresponding seasonal difference in 1910 was nearly 10° F. Yet the authors plotted a temperature gradient of the rocks of the Witwatersrand from their diminished drill-hole temperatures. In common with their predecessors, they not only overlooked the facts set out above, but also failed to remember a much more important fact.

I have just shown how a mere seasonal difference of surface temperature may affect the temperature at the bottom of a 4 ft. hole 10° in a few months. The effect of the circulation of air from the surface on the rock walls of a mine is immensely greater. Even before the mine passes out of the development stage into its productive career, the main shafts and a large part of the drives have usually been open for at least two or three years. If a variation of 13° in the surface temperature at Arcadia can and does produce a 10° variation of drill-hole temperature in five months, it is clear that the constant circulation, for years, of air very much below the original temperature of the rock exposed will produce large temperature changes in the rock even at a great distance below its surface. This distance may well be 100 ft., or more. This phenomenon, in itself, is enough to account for the discrepancies in rock temperature gradients in mines, as read from drill-holes. The effect of the circulating air is very much added to if the mine be naturally wet, or sprayed for any purpose. If the quantity of water extracted as vapour daily from the mine be 200 tons—(202 tons having been taken out of the Village Deep in 24 hours)—the heat so abstracted from the mine is about 430,000,000 heat units. If the air and contained moisture leaving a mine has more total sensible heat than can be accounted for by the sum of its heat on entry plus the balance of the oxidising, mechanical heating, and mechanical refrigerating effects known to exist, then the whole of the latent heat of the moisture carried out of the mine by the air must, on balance, come from the original heat of the rock. In such a case the final temperature of the air is not lowered by the evaporation of the water.

It is known that the combined presence of working rock-drills and their crews lowers the temperature in working places in the mines of the Rand by several degrees. This is practically a local effect, and one which as far as the whole mine is concerned is usually negligible. The heating effect due to each man at work is about 60 watts, or, say, 3,000 heat units per day, or less than that due to the combustion of one-fourth of a lb. of coal per man per day. Candles may be taken at the equivalent of ½ lb. of coal per man, or 6,000 heat units per man per day. The heating effect of explosives may also be taken as equal, roughly, to about ½ lb. of coal, or 6,000 heat units per man per day. Assuming 3,000 men underground in a mine like the Village Deep, the total heating effect from these causes is certainly less than 15,000 H.U. per man per day. For a force of 3,000 underground men the total is 45,000,000 H.U. (corresponding to about 7° rise in temperature for 260,000 cub. ft. of air per min. for 23½ hours daily), so that roughly, 400,000,000 H.U. is daily abstracted from the rock. This latter is about the amount of heat obtained by the combustion of 15 tons of coal, or about 11 lb. for each of the 3,000 persons underground. The oxidation of timber and pyrite is clearly neligible as compared with any of the figures already given.

I will put the facts another way, which will show the effect more clearly still. Assume that 5,000,000 tons of rock have been taken from the workings, as one sheet 6 ft. thick. Then every ton mined has left about 4 sq. ft. of surface exposed. If the above quantity of heat is withdrawn evenly, then about 20 heat units are taken from each sq. ft. of surface per 24 hours by the evaporated water alone. Taking the specific heat of the rock at even 0·2, and the weight of a cubic foot at 170 lb., the cooling effect per sq. ft. of exposed surface would be sufficient to lower the temperature of a cub. ft. of rock about 4° in a week, or of 100 cub. ft. about 2° in a year.

In Professor Cadman's paper, an abstract of which appears in the February *Journal* of this Society,* attention is drawn to the possibility of a mine being warmed up above the original rock temperature by oxidation. He says that the effect produced will depend upon the rate of oxidation as compared with the rate of ventilation. It seems pretty clear that if a general and constant oxidation is taking place at all in any part of a given ventilating current, then the rate of oxidation is likely to increase faster than the rate of ventilation, as is indeed the case with an ordinary fire. As no one has ever yet mentioned a working mine, not visibly burning, in which ventilation increased the temperature of the workings generally, it is extremely doubtful whether such a mine has ever existed. Professor Cadman also says that both effects are seen in most coal-workings: the intake and part of the workings being cooled by the air current, whilst part of the workings and the return are warmed up. This has no necessary connection with an oxidation effect. A similar thing occurs in cases where there is no oxidising effect at all. The circulation of a current of water or air through a hole in an unequally heated solid, parts of which are below the final air or water temperature, must necessarily transfer heat from some of the hotter to some of the cooler parts of the solid. This happens in most deep mines, whether they are coal mines or not. The operation of the ventilating current tends, on the whole, to flatten the original temperature gradient of the rock, and to reduce it to the natural mean temperature gradient which the air has due to compression. This is about 1° in 180 ft.

The wet-bulb air temperature gradient will not tend to fall below this figure because the latent heat of the water-vapour will practically all come from the rock, most of the rock being hotter than the air. Unless refrigerating methods are used, we cannot therefore hope to reduce the air temperature gradient below this figure of 1 in 180, as a *mean* for the year and for the mine. In winter the gradient would be steeper, with slightly lower temperatures at the bottom of a very deep mine; and in summer slightly flatter with a slightly higher temperature down below. The humidity in deep mines, with all stopes and working places sprayed, would be nearly 100 in both cases. 85° F. wet-bulb mean annual temperature will therefore be met with on the Rand mines at a mean depth of about 5,000 ft. or less, no matter how strong the ventilating current is made. This temperature is judged by Prof. Cadman and others to be the limit for fair physical effort, even with good ventilation. If mining is to be carried on economically at greater depths it will be necessary to refrigerate air supplied to the working places. In England the corresponding depth would be about 6,500 ft.

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* See this *Journal*, Vol. XIV., February, 1914, p. 388.

It may be said that this is "speculative." Well, compare it with the facts. It is certainly one fact that there is no mining field in the world for which half as low a temperature gradient is claimed as for the Rand. Another is that even in the list of temperatures of 53 mines of the Rand, published in the Government Mining Engineers' report for 1912, there are but four mines returning their deepest temperatures at such figures as to give a flatter air gradient (reckoned from the mean temperature of 60° at the surface) than 1 in 180. One of them is the Village Deep, which returns a temperature of 70° at 4,245 ft.: thus giving the obviously absurd gradient of 1 in 424. Another is the Simmer deep, returning the low figure of 65° for a depth of 2,300 ft., or 1 in 460. Another is the Van Ryn Estates, giving 60° at 1,600 ft., or no gradient at all (reckoned from 60° at the surface). The fourth is the York, with 58° at the bottom of the mine, and a negative gradient! The temperatures of these four mines, if correctly given, must have been taken in specially cool spots, or on a very cold day. In the list there are 12 mines with air gradients between 1 in 180 and 1 in 150: 24 mines with gradients between 1 in 149 and 1 in 100: and 13 mines with gradients steeper than 1 in 100, the steepest given being that of the New Modderfontein, which is 1 in 61. With the exception of the four doubtful cases just mentioned, therefore, there are no mines in the world, as yet, claiming a flatter gradient than that I have set forth as the flattest probable. The majority even of the mines of the Rand have much steeper gradients, and outside the Rand the gradient is, on the average, about twice as steep.

Mr. E. Pam *(Member of Council)*: I have much pleasure in proposing a vote of thanks to Mr. Moynihan for his paper. The matter is of such vital importance to all of us that we should get accurate figures without delay as to what our rock temperatures, and temperatures underground, really are. It is quite obvious that unless we have scientific data to go on, it will be practically impossible to know what efforts must be made and what work is necessary to reduce temperatures to somewhere about the 85° maximum mentioned by Prof. Cadman. I have not the least intention of making any remarks on this paper without proper study of it, but I hope that Mr. Whitehouse and Mr. Wotherspoon being away, someone else on the Village Deep may go into the points raised by the author, and carry out further experiments, or look into their former experimental figures to see what reasons there are why the temperatures collected by them are lower than what one would expect from experiments on the rest of the Rand. I hope to read through this paper more carefully and possibly to make a few remarks at a later date.

The vote was carried by acclamation.

The President: Mr. Moynihan has given us a thoughtful contribution, and I hope the ensuing discussion will elicit such information as will enable us to put this very important question of rock temperatures on a proper scientific footing.

THE PREVENTION OF DUST IN UNDERGROUND WORKINGS.

(Read at January Meeting, 1914).

By B. C. Gullachsen, M.I.M.E. (Member).

DISCUSSION.

Mr. W. S. V. Price *(Member)*: I have brought with me a number of sprays of different kinds, in use on the Consolidated Main Reef Mine. This which I will call No. 1* can be adjusted to throw a very fine spray, or a coarser circular shower of water. The main objection to this spray is that Kafirs are apt to screw the front end in so tightly as to stop the spray altogether, or loosen it so as to destroy the effect as a spray, beside wasting a lot of water.

No 2, which is called the "Simplex," gives a very fine spray over a large area, it is used in the drives or to spray a compartment of the shaft; in the former case it may be used close to the main water column with a very short connection, and sending the water up vertically, or the connection may be long enough to reach the top of the drive, and the spray thrown down and along the drive. In the shaft, a pipe would be led right across the shaft, and one spray would be turned on to each compartment, which would give quite a fine spray of water right across the shaft, moistening the air thoroughly and arresting a large amount of dust.

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* "Ball Spray."

No. 3 is called "Price's Patent Spray," and is a very simple arrangement, throwing a very fine and broken stream; it is one of the best, as the drops of water are so small that they do not drench a man, as some of the coarser shower sprays do. It can be and is used in the same manner and under the same conditions as the Simplex.

No. 4 has a hole bored part of the way through, then a piece is cut out so that the water breaks on the bevelled part of the cut. It is an older type of spray, and is not being used to any extent now. In fact, it, as well as No. 1, the "Ball Spray," is being rejected in favour of the other sprays, which give better results for the amount of water used.

No. 5, "The Atomiser," is extravagant, as it takes about 25 cubic feet of air an hour to run it, but if it is properly used it will fill a drive with a fairly dense fog for the distance of a couple of hundred feet, which, after a blast, must catch and cause to fall to the ground a lot of dust, as well as dissolving any nitrous fumes it may come in contact with. I have found that the best result is obtained by turning on a small quantity of water only, but full air to thoroughly break up the drops of water. I found that when full air and full water were turned on, the drops of water were so large they soon fell to the ground; but with a little water it remained longer in suspension, and so had a better chance to be brought into contact with the dust.

I quite fail to see the force of Mr. Gullachsen's argument *re* the two water blasts, one at the face and one further along the drive, where he says that the gas and smoke would be pent up like a corked bottle. If the smoke *was* pent up or sent from air jet to air jet, it would soon fall to the ground by reason of the moisture it would collect, and that, I take it, is what we are aiming at. As to the gas, again even if it were pent up, it would be so diluted that it would be practically innocuous. But I do not think it would be; for we have a current of fresh air coming through a pipe in the dead end of the drive, and there is only one way of escape, *i.e.*, along the drive, and the force would be bound to drive gas, etc., past the next supply of fresh air, but it would have to pass through the water and so the dust would be caught.

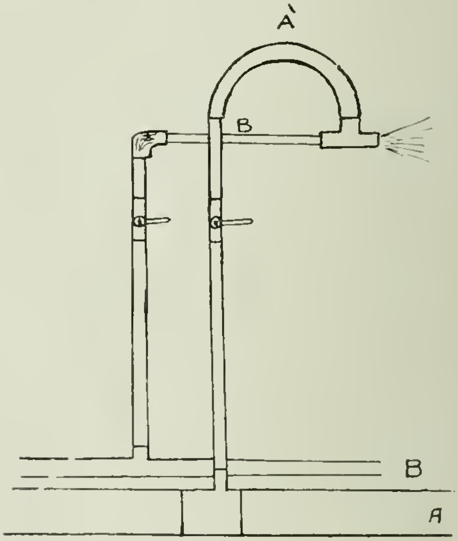
Mr. Gullachsen mentions the fact that tramming boys object to the cold douche of the fixed spray. He mentions the scheme for prevention of dust by mixing molasses with water. I say the water must be mixed with something, even if it is only with "brains," and if the spray is a very fine one, the boys would not touch it; but when it more resembles an open tap, or a very heavy shower bath, no wonder they object. The same thing applies to the direction in which sprays are turned. If the sprays face toward the station, and a man, probably wet through with perspiration, having just charged up and blasted his round, turns on the spray and gets a shower on his heated body, I am afraid he will fail to appreciate the amount of good the water will do, and only curse it for fear of rheumatism or pneumonia. This would be avoided if all the sprays were facing inwards, *i.e.*, towards drive ends, the handles of the taps up near the spray, so that they could be turned on without having to bend down: I do not mention this merely to save a man the trouble of bending his back, but sometimes the tap is stiff, in fact, they generally are if they have been in use any length of time, and the time lost is considerable when a man is trying to get away from the reports of a blast. In short, if we want the sprays to be used consistently, as they must be to be effective, we must have them arranged so as to be easy of access and to cause a minimum of discomfort to make them popular with the men. They are already saying that the cure is worse than the disease, that phthisis will be stamped out automatically, for the miners will all die of pneumonia or rheumatic fever! Still, if we mix a little "brains" with the water, I think we shall have no need to say that; but the men will give the thing a fair trial, which will undoubtedly almost entirely eliminate our old enemy phthisis, without the fear of others. Just one remark in concluding. I wish to speak of the drainage of levels. With all this water all over the place, more attention will have to be paid to draining the levels efficiently. Under each box-hole, where fine gets washed down the stopes and chokes the drain, a 3in. pipe could be laid just long enough to clear the ends of the box, so that even if it got covered, the water could go in at one end and out at the other. A bit of chain or rod iron, or even a bit of old ½in. pipe would be good enough to stop it from getting choked, and the old pipes we see lying about would be well utilised in such work, as it would prevent men getting their feet wet as soon as they go on shift, with consequent risk of colds, etc., to say nothing of the discomfort.

I quite agree with the author on the advisability of the pipe-fitters being underground an hour or so before the shift goes down, for I know by experience what it is to have to wait for water while the pipe-fitter repairs a broken or blasted water pipe somewhere between my working place and the station. I wish now to thank Mr. Gullachsen most heartily for his paper, and the opportunity for discussing this most important question from a miner's point of view.

Mr. E. M. Weston *(Member):* I have read Mr. Gullachsen's paper with interest, and must congratulate him on his careful review of the subject. The author evidently knows of what he writes, and has had personal experience of the weak spots that develop in practice in the most beautiful schemes whenever they are put in practice underground. As in the old days the air service of pipes for rock drills was put in anyhow without any idea or calculation with regard to the work it would have to do, so recently in many places they were put in on quite an inadequate scale, with the result that as the great demand for water is at the beginning of the shift (to wet down stopes, etc.), in practice the system failed hopelessly. To remedy this Mr. McGuire, on the Jupiter mine, put in 100 gallon tanks near all the principal working faces, so that local reservoirs were at hand to supply sudden needs. This is a system that needs adoption in some mines to-day. Another cause of trouble was that no proper attempts were made to regulate pressures; so that at the bottom levels the water pressure became so excessive as to rupture pipes and hoses, and as it is a heart-breaking task to try and induce any miner to take the slightest interest in economising water, and as the jets on water hoses are always either lost or cut off by the natives, the result was that there was an enormous wastage of water on the lower levels and a shortage on the upper levels. Why do natives cut off the iron pipe jets? For one reason because when the hose is not in use it spurts out more with the jet on and wets them more, and they prefer a quite full bore stream which is regulated when wanted by pinching the hose with their fingers. For another, because jets get choked, and for both miners and boys it is so much easier to cut them off than to blow them out by inserting the jet end in the spud of an air-pipe! Such are some of the problems of mining with irresponsible and careless white and black labour.

Water services should be planned as carefully as air services. They should draw their supplies from as many different sources and from such different elevations as may be necessary. I scarcely agree with the author when he states that the contract system breeds dust; as in developing there are generally wet holes that can be drilled while waiting for repairs to water services. As the author points out, the true remedy for many water troubles is to have a pipe-fitter working on night shift or between shifts. This practice I always try to follow, and it pays in many ways. As the author from his remarks on pipe-fitters evidently knows by experience, much money is lost through false economy of labour in this department and others. Dust is produced, rounds lost, benches are ruined and other drilling time lost. All this does not show up as due to its true causes on the "statistics" (a doubtful word as applied to most underground work!); but it looks well to reduce pipefitting costs £50 in one whole month. I think I was the first to draw this Society's attention to the question of increased humidity due to the large quantity of water necessary in deeper workings. I doubt, however, if it need directly affect health. We have, I think, something to hope for from rock drills using hollow steel and water, but these machines must be used intelligently. I do, however, from what I have seen in the mines I have visited lately, emphatically disagree with Dr. Aymard's publicly announced statement that there is enough dust at present being produced by rock-drilling alone to produce the terrible crop of phthisis cases. Men and natives will occasionally endanger their own lives and lives of others by making dust; but during the last few years the Inspectors of the Mines Department have done much good work in this direction. Mine officials have in most cases realised the importance of the subject, and *that it is for their own lives* as well as for those of others that they need take heed, and I believe conditions have vastly improved. Could we dispense with the practice of returning after blasting cuts, or could we be sure that in all cases the water blast was used and a reasonable time allowed before returning after blasting, we might feel fairly satisfied. The mining regulation that seeks to make the shift boss responsible for this is ridiculous. The eight-hour law has, I believe, brought almost more evils in its train than it has brought benefits. More rush, more hurry, more accidents, more recklessness; but that is another story!

Dr. Aymard's remedy for all our troubles is steam! Excellent! We poor miners were threatened with three deaths—dust and fumes if we work dry, rheumatism, etc., if we use water, pneumonia if we instal swift ventilating air currents. Dr. Aymard now proposes to endeavour to scald us alive as we work through the shift. For we note that, according to him, we make dust during all the shift and only steam can allay this. Another scientist, known chiefly for his scorn for the practical man and all his ways, proposes to introduce this steam at the close of the shift by sending the steam down the air pipes. Three thousand feet of wet shaft column—ten, fifty, one hundred thousand running feet of air pipes in water, in sprays, and in cold draughts—steam! Let us, however, return to the paths of sanity. We may take it that we have removed, or almost removed, the danger from dust in drilling or shovelling rock. The dust after blasting remains. In a moderate current of air a water-blast will, in about half an hour, settle practically all dust, fine or coarse. Hence, the problem in dead ends seems to be solved if everyone will do his duty. In a strong current this very fine dust will, however, pass water blasts and sprays. Therefore, we endeavour to remove as many men as possible before or shortly after blasting, and I would recommend that, in addition, we should make the main ventilating currents at that time *as strong as possible*, so that you may sweep the finest dust *right out of the mine*. What can be settled should be settled so that it will fall on to the wet floor of the drive and remain there, or be washed away. This is where we are, I believe, weak in practice to-day. Too much fine and semi-coarse dust settles in stopes and on the walls and sides of drives and main cross-cuts, haulages, etc., and is liable to be sent on its course again at times when it may do damage. The molasses method may have its benefits, but I believe that the real solution is the placing of a sufficient number of properly constructed water-blasts at all strategic points of the mines. I emphasise "properly constructed," as the great objection to the use of water-blasts on main pipe lines is that they are liable to flood the whole pipe system during or between shifts, and to cause much inconvenience and loss. To this I attribute their somewhat limited employment. The accompanying sketch explains itself. The upright pipes may be 1in. diam. and a half-inch diameter. I would, however, advise that the water tap be made with a plug bored with an eighth to quarter inch hole, and the air tap with a plug bored a quarter inch to half inch, in order to avoid waste of air and water. Owing to the bend on the air pipe A, it will be at once seen that water cannot enter the air pipes A, as the water from B should the air pressure fail, simply discharges into the drive direct. I would earnestly commend this plan to those interested. Owing to having a comparatively few of these on any level, it becomes easy to make the turning of them on at or near blasting time one of the definite duties of day trammers or lashers. They can also be turned on for a few minutes during the shift, and will wet the walls of drives for two or three hundred feet back along the drives, and cool and moisten the atmosphere.



Sketch of Water Blast.

Personally, I think sprays are of little use. If, as the author says, an attempt is made the Kafir will very naturally cover them up or he will annex the brass tops. I arrange my sprays two or three together on the top of a pipe arched over the drive to form a spray screen right across the air current. These are turned on occasionally during the shift, and the rock driller is instructed to turn them on after blasting. It is much more easy to check the turning on of the "strategic" water-blasts.

With the author I agree that the matter of prevention is largely one of education. "He knoweth not that it is for his life," is a text from which a sermon with many applications might be preached to mine employees. I have been surprised at the ignorance of apparently intelligent white miners, and the natives must be in a worse state. I would suggest that the bioscope be brought to our aid in spreading the knowledge of the terrible disease of miners' phthisis. Make the film as gruesome as may be needed, show the causes of the disease and results of neglect and carelessness. With concise and simple letterpress explain the danger and how to guard against it, and I am sure much good would be done. I very much regret this paper has not brought out more discussion and show of interest from members, and must again thank the author for endeavouring to do his "bit" towards bettering conditions.

THE NATURAL SODA DEPOSITS OF AFRICA: WITH SOME NOTES ON THE ALKALI TRADE.

(Read at November Meeting, 1913.)

By John Watson, F.I.C. (Member of Council).

REPLY TO DISCUSSION.

Mr. John Watson *(Member)*: The author expresses his thanks to the three members who have taken part in this discussion. To test the point raised by Mr. Whitby, three samples of settled and sun-dried slime were taken from three different points of the City and Suburban slime dam, *i.e.*, the dam into which treated slime residues are pumped.

Protective Alkali.—A preliminary test for alkalinity, carried out on weighed quantities, with phenol-phthalein, showed on No. 1 sample barely faint pink, No. 2 faint pink, and No. 3 distinctly pink.

Samples Nos. 1 and 2 were then tested for "protective alkali," after addition of $AgNO_3$ solution, 5 cc. of $N/10\ K_4Fe(CN)_6$ and a few drops of KI solution, according to which routine test neither sample showed any protective alkali. In other words, any calcium hydrate which had been present had now become carbonated or sulphated.

Comparative Coagulation.—Equal weights of No. 1 slime sample and tap-water (R.W.B.) were then made up, so as to fill two graduated, cylindrical, stoppered litre glass-jars. The author found that 700 gm. of the above slime and 700 cc. water gave exactly 1,000 cc. of the consistency of "collected slime."

To the first jar, 2.436 gm. of pulverised "Northern Lime" (—30 mesh) were added; to the second, the same weight of S.A. Alkali Co.'s crude calcined soda (also —30 mesh) was added. This quantity was figured out on the basis of 6 lb. lime per ton of slime, given on p. 211 of "Rand Metallurgical Practice," Vol. I. The two jars were then stoppered, thoroughly agitated, stoppers withdrawn, and allowed to settle side by side.

The following figures show the rate of settling:—

Slime Sample No. 1.			Slime Sample No. 2.		
After	Lime.	Soda.	After	Lime.	Soda.
hr. min.	cc.	cc.	hr. min.	cc.	cc.
5	950	982	12	980	982
10	915	970	30	912	945
15	870	952	45	860	918
20	830	936	1 0	801	890
25	800	927	1 10	756	867
30	765	912	1 30	680	838
40	685	885	1 40	640	806
1 0	623	822	2 10	633	730
1 10	613	795	2 35	633	730
1 20	612	760	4 0	633	637
1 30	612	733	5 0	632	629
2 0	612	656	5 30	632	629
2 30	610	642	6 30	632	626
4 30	610	622	24 0	632	622
5 0	610	620			
6 0	610	618			
24 0	610	617			
	Supernatant water clear.	Supernatant water rather milky.		Supernatant water beautifully clear.	Supernatant water rather milky. Next morning, supernatant water much clearer; but, still less clear than the other jar.

Fezzan.—The author is aware that Dr. Lunge described, somewhat fully, these natron lakes, in his second edition, published in 1895 (Vol. II., pp. 61 and 62). In his third edition in 1909, the doctor contents himself with a brief description and a reference to the previous edition.

Magadi.—The author is indebted to Professor Wilkinson for the reference to "The State," which article he had not seen, but has since read with interest. Also, the analysis by the respected hon. treasurer of the Institute of Chemistry is highly interesting, as showing an article of great purity after calcination. There is no information quoted, however, as to how much this sample represents. It might be representative of the bulk of the 200 million tons which Lake Magadi is said to contain; on the other hand, it might represent merely a picked small sample of, say, half an ounce of extra pure natural crystal.

Zoutpan.—Zoutpan *does* mean saltpan. As the author said, on p. 237, two well-known firms have tried to work the pan for its sodium chloride content; the present company regards it, however, as a soda-pan, and calls it such.

His critic says: "This is an impure deposit of soda, about 27 ft. thick. . . . . ." Such a statement is somewhat lacking in fairness, as Drs. Hatch and Corstorphine say (on p. 185): "The presence of this deposit of carbonate and chloride of sodium to a depth of 200 ft. . . ." It may be news to his critic that the dark coloured impregnated clay, on exposure to sunlight and the atmosphere, for a few days, loses a large proportion of moisture and becomes almost white in colour. Analyses of this material made by Mr. W. B. Marshall, Ph.C. (Lond.) showed fairly high percentages of sodium carbonate.

The analyses in "The Geology of South Africa" (loc. cit.) are practically complete, except the 2′-4″ deposit, in which case the the analysis totals up to only 90·96.

The author did not have the same facilities for sampling the various layers as the above-mentioned geologists; but he cannot see why Professor Wilkinson should desire to have such work repeated. The crude crystal has to be dried and calcined; also, it is certain that the layers vary in different parts of the pan and at different seasons of the year.

Then, why worry as to how much water of crystallisation the natural carbonate or sesquicarbonate contains? "Economic principles" forbid the carriage of tons of water through the bushveld, 14 miles by ox-wagon, and over the S.A. Railways, a distance of 73 miles from Haman's Kraal to Johannesburg, where water is more plentiful than in that arid region.

The analysis quoted by Professor Wilkinson, done by Clarke, from some unnamed source, is stated in a very unusual form. The author has yet to learn that British, or any other, alkali-makers state their analyses, or sell their products, according to the percentage of metallic sodium.

Soda Crystals.—The pure article contains:—

Soda (Na_2O)	21·68
Carbon dioxide	15·39
Water	62·93
	100·00

The sample of locally made crystal was taken from shelves where the material had been laid to dry and effloresce. The author was told that Johannesburg buyers prefer it in this condition. It has been said that water at this altitude evaporates as rapidly as alcohol does at sea-level. This fact would account for the increase in the percentage of real soda to 22·46%, as compared with the above 21·68%. The author regrets that a hitherto undetected printer's error gives the real soda as 24.46%. This should read 22.46

Professor Wilkinson says a large portion of the paper can be found in text-books, even of an elementary character.

This assertion is unwarranted, and the author would feel obliged if his critics could let him know the titles and authors, or publishers, of such elementary text-books. In almost every case where an author is quoted the reference to his publication is given as a matter of literary etiquette. Does the critic do this in his contribution? He mentions Hoffmann, Cohen, and Clarke, with no reference to their published papers. The author doubts whether 1% of our members know where to find these references.

Had his critic done him the honour of being present to hear the paper, he would not write as he does *re* caustic soda and soap manufacture.

The audience was told that the increased import of caustic soda, and the decreased import of soap for 1912, were due to the starting of the Congella Soap Works.

He also says it is obvious that the author has not kept abreast of the later developments of alkali production. He does not, himself, specifiy what those later developments are!

But besides corresponding with old colleagues, the author has subscribed for and studied at least three or four of the leading technical journals now being published bearing on alkali manufacture.

THE UNION PATENTS BILL.

(Read at April Meeting.)

By A. L. Spoor and W. E. John (Members).

DISCUSSION.

Mr. E. J. Moynihan *(Associate)*: The Patent Law referred to by the authors of this paper is probably still-born, and will never appear on the Statute-book. Nevertheless, I think it advisable to say a few words on this subject from an entirely unpopular point of view.

The law which has been drafted is based on generally accepted assumptions which are quite untenable. The first of these is that the patent laws of the world have encouraged invention and inventors. The only argument in favour of this proposition is that patentable inventions of importance have greatly increased during the existence of the modern patent laws. It does not follow that they have done so because of these laws. It is highly likely that the increase is due rather to the spread of scientific knowledge, of education, and of the factory system; and also to the spread of scientific discovery, which itself has been encouraged by no patent laws at all. In proportion to the number of people possessed of scientific knowledge and education, I think it is unquestionable that the number of patentable inventions taken to the patent offices has very much diminished during the last fifty years. In a vague and instinctive kind of way the inventors are beginning to find the patent laws out.

What they are beginning to see is that the nations of the world, under the pretence—I admit the unconscious pretence—of giving the inventor bread, give him a stone. The inventor, in his capacity as inventor, is not a business or commercial man. As far as head work is concerned the real inventor invents because he likes it, and he will continue to invent, whether there is any financial reward or not, as long as he can keep going, and is not starved to death. The principle of all the patent laws of the world is to give the inventor, as a reward for his genius, a commercial monopoly. In the first place, this is of little use to inventors as a class, because they neither know how nor have the time to turn it to proper benefit; and in the second place the patent commercial monopoly granted is so fraught with legal pitfalls that those who have the commercial instinct—merely another name for greed—find the inventor an easy prey. In my judgment, inventors, as a class, would be much better off if there were no patent laws at all, than is now the case. There would then be no burking of discovery nor shelving of improvements by big corporations, such as is now common. Free competition would lead to much more rapid progress and improvement of industry. The controllers of manufacture would be forced to employ men with an inventive type of mind, in order to keep pace with their competitors; and as the number of men possessing originality is quite limited, the competition for their services would send their wages up to at least a thousand times what it is at present. Their present remuneration, on the average, and for inventing, is much less than that of an unskilled child labourer. Their salaries, with the patent laws abolished, would probably average ten thousand pounds a year. Of course, I am speaking of those who can really invent useful things, and not of the 90 per cent. of patentees whose inventions are of no value whatever to the community.

To bear out this statement, it is only necessary to point to the case of those who are able to exercise invention of a non-patentable kind in the preparation of clever advertising matter. These men are not protected in their calling by any legislation at all. It is open to anybody to do what the advertisement writers do—if he can. Nevertheless, able writers of advertisements command high salaries.

However, it is quite hopeless to expect that my view of the patent law will prevail generally for a great many years to come. It is therefore necessary to suggest ways in which the present patent law can be so modified as to be less a hindrance to the inventor than it now is. In this connection, the first thing to be borne in mind is that the inventor is nearly always a poor man, to whom delayed or expensive justice is in effect no justice whatever; and to whom uncertain rights are no rights at all. It is better to give him a moderate amount of protection which is certain, than an exclusive monopoly which is at the mercy of everyone financially powerful enough to steal it. I, therefore, suggest that a patent once granted should have some of the characteristics of a registered mining title. First, it should be unassailable as to validity. Before being granted, the fact of the forthcoming grant should be advertised, and all objectors heard by a simple procedure in the Patent Office. Second, no inventor should be allowed to be sued for any infringement provided he is working in accordance with a patent which has been granted to him. Third, all technical patent suits should be heard by technicians in the Patent Office, and not by a civil judge, who is technically incapable. No "costs" should be allowed.

These three provisions should make a patent of some real and negotiable value to the possessor, and would greatly improve the existing laws. The inventor would not get his due, but he would really get something. At present he gets nothing.

THE WITWATERSRAND EARTH TREMORS.

(Read at April Meeting, 1914.)

By H. E. Wood, M.Sc., F.R.Met.S. (of the Union Observatory).

DISCUSSION.

Prof. G. H. Stanley *(Vice-President)*: In the *Mining Journal* for 1909 a gentleman describes experiences at Oaxaca, in Mexico. He says the district is particularly liable to earthquakes, which, although they have, in many instances, excited persons on the surface, have not been felt underground. He gives two instances. In one case two earthquakes lasting, respectively, 45 seconds and 20 seconds with a 5½ hours interval between, did considerable damage on the surface and were not felt at all underground, and the men, on coming up, were quite surprised to find what had happened. He also noticed that in these mines the quantity and temperature of the water increased immediately after an earthquake.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Effect of Tungsten on the Ammonium-Molybdate Assay of Lead.—"At a previous meeting of the Broken Hill Branch of the Institute the writer reported the occurrence of two minerals in the sulphide zone of Block 14 Mine, both of them new to Broken Hill. They were wolfram and scheelite. The former—wolfram—a tungstate of iron and manganese, gave on assay 42% tungstic acid. The latter—scheelite—a tungstate of lime, gave 35% tungstic acid. Both these samples contained zinc and lead sulphides, and clean specimens of wolfram gave 64% WO_3; scheelite gave 70% WO_3.

An estimation of the lead in these samples revealed the fact that the presence of tungsten in the ore seriously interfered with the ammonium molybdate assay for lead, giving low results according to the amount of tungsten present in the ore. The sample under review, when assayed under ordinary conditions, gave a negative value for lead, but on removal of the tungsten showed 9·8% Pb. The reason for this interference is shown on an examination of the reactions which occur in the assay. The ore is reduced with suitable acids, and the lead obtained in the form of sulphate, which is dissolved in ammonium acetate, thus:—

$$\text{A.}\quad \underset{\text{Lead sulphate.}}{PbSO_4} + \underset{\text{Ammon. acetate.}}{2NH_4C_2H_3O_2} = \underset{\text{Lead acetate.}}{Pb(C_2H_3O_2)_2} + \underset{\text{Ammon. sulphate.}}{(NH_4)_2SO_4}$$

The lead acetate thus formed is now titrated with a standard solution of ammonium molybdate as follows:—

$$\text{B.}$$

$$\underset{\text{Lead acetate.}}{Pb(C_2H_3O_2)_2} + \underset{\text{Ammon. molybdate.}}{(NH_4)_2MoO_4} = \underset{\text{Lead molybdate.}}{PbMoO_4} + \underset{\text{Ammon. acetate.}}{2NH_4C_2H_3O_2}$$

The above reactions take place under ordinary conditions, but when the ore contains tungsten, this is converted into the trioxide by the action of the acids, and during solution by ammonium acetate it readily combines with the lead, forming lead tungstate as per equation (C.):—

$$\text{C.}\quad \underset{\text{Lead acetate.}}{Pb(C_2H_3O_2)_2} + \underset{\text{Ammon. tungstate.}}{(NH_4)_2WO_4} = \underset{\text{Lead tungstate.}}{PbWO_4} + \underset{\text{Ammon. acetate.}}{2NH_4C_2H_3O_2}$$

So that if sufficient tungsten be present in the ore to combine with all the lead present, the addition of the standard solution gives negative results.

It therefore becomes necessary to remove the tungsten before proceeding with the lead assay, and this is done in the following way:—

The ore is digested in 10 cc. HCl, 10 cc. HNO_3 added, and evaporated to small bulk. This is taken up with water, boiled, allowed to settle, and filtered. By this means the tungsten is converted into tungstic acid, and removed with the silica. To the filtrate is added H_2SO_4 taken to white fumes (this is important, otherwise gives 0·2 high in 14% lead, and the assay proceeded with in the usual manner.

The reaction occurring in equation (C) suggested a rapid volumetric method for the estimation of tungsten.

The tungstic acid obtained on the filter-paper in the previous operation is dissolved in ammonium hydroxide, neutralised with acetic acid, and an excess of standardised solution of lead acetate added. This combines with the WO_3 to form lead tungstate as per equation (C). The excess of lead acetate solution is now titrated with ammonium-molybdate solution, as in the ordinary lead assay and shown by equation (B). From the formula $PbWO_4$ the factor for the tungstic acid, WO_3, can be obtained.

The presence of tungstic acid has no effect on the zinc assay. A laboratory experiment in flotation was carried out on some sulphide ore containing scheelite in order to see if this method could be used in separation. The following results were obtained:

	Lead.	Silver.	Zinc.	WO_3.
No. 1 concentrate ...	52·4	23·4	13·2	—
Residue	2·8	15·5	1·0	20·4
No. 2 concentrate ...	53·6	26·2	12·6	
Residue	3·5	14·2	1·0	17·4

Note the high silver in the residue, which latter readily lends itself to further concentration by tabling.

The above has a local interest from the fact that lead concentrate made from a parcel of Block 14 ore showed the presence of 5·5% WO_3. The concentrate contained 63% lead (approx.), and the assay value shown by buyer and seller differed by upwards of 3 %. The umpire, who evidently recognised and removed the tungsten, agreed with the higher of the two assays.

The method shown above for the elimination of the tungsten in the determination of the lead was embodied in an agreement for the purchase of crude ore from Block 14 Mine, and has also been adopted in the lead assay for the purchase of certain lead concentrates."—HERBERT LAVERS. *Australian Institute of Mining Engineers*, 1913, No. 10; *Chemical News*, Feb. 27, 1914, p. 97. (C. T.)

THE SPITTING OF SILVER. — "The amount of oxygen absorbed by melted silver is so large that it seems difficult to explain on the hypothesis of a mere solution of such a gas in such a liquid. It seemed conceivable that there might be formed an oxide of silver which was stable at high temperatures, but decomposed on allowing the temperature to fall; a case analogous to the well-known behaviour of silicon trichloride. An attempt has been made to apply a test for the presence in the molten metal of an oxide of silver. It is found that if silver oxide is dropped into melted boron trioxide silver borate is formed, although the temperature is much above the decomposition point of the silver oxide. Accordingly, some highly purified silver was melted in a stream of oxygen, a part only of the surface of the metal being covered with boron trioxide. This procedure was rendered easy by the convexity of the surface of the liquid metal, a ring of the trioxide surrounding the exposed surface. In these circumstances silver borate is formed in large quantity. If, however, the metal is covered entirely, even by a thin film of the trioxide, no silver borate is produced. It cannot be claimed that an absolute proof of the existence of an oxide of silver has been obtained, because there is the possibility of the reaction of three substances together, no two of which will otherwise unite. It seemed worth while, however, to put the experiments on record."—H. BRERETON BAKER.—*Proc. Chem. Soc.*, Vol. 30, No. 425, p. 56. (C. T.)

METALLURGY.

TUBE MILL PRACTICE.—"In dry grinding the 5 ft × 22 ft. mill may, perhaps be the most suitable size but in the kinds of wet grinding usually required in the reduction of ores the conditions vary so tremendously that, correctly speaking, each particular installation should be studied and if tube mills are selected for the work, their length and diameter should be carefully proportioned so as to produce the desired results with as small an expenditure of power as possible.

Some years ago I was managing a mine where we had a stamp battery of twenty head, which required about 37 h.p. and this would only reduce about 35 tons of the ore, which was very hard, to the fineness required for its subsequent cyanide treatment. It was desired to increase the tonnage and a 4 ft. × 16 ft. tube mill, which consumed 25 h.p., was installed. After getting this into proper running condition it was found that 70 tons could now be reduced. Of course, it may not be quite correct to credit the extra 35 tons to the 25 h.p. tube mill, but it shows that the tube mill is not as wasteful on power as some think.

The name 'tube mill' naturally suggests a long cylinder of small diameter and we find that most early installations were essentially of this description and often of a length far in excess of what was necessary to reduce the ore to the required fineness. Consequently a large percentage of slime was produced, which soon gave the tube mills the reputation of being suitable only for sliming, *i.e.*, very fine grinding. The conical mill now came along at the psychological moment and mill men were astonished to find that a very short cylinder terminating with a cone would do as much grinding as a long tube mill, and with very much less power, without producing the large percentage of slime. This mill 'caught on' and for awhile bid fair to drive the old tube mills out of commission. Of course, it was the cone that did it! But it did not seem to strike tube mill operators that if they were to cut their long tube mills in halves and make two short ones out of one long one, they would, in many cases, pretty well double their grinding for about the same power, and not produce as much slime.

Manufacturers of tube mills are now awakening to the fact that it is not necessary to lay too much stress on the 'tube' idea and are turning out mills of a length a little in excess of the diameter. These are giving astonishingly satisfactory results where the object is to make a fairly fine but granular product.

In considering tube mill installations it is obvious that the length and diameter of the mill are very important factors, and these should be proportioned to the class of work to be performed. It follows from this that the number of mills to be installed for a given tonnage must be carefully considered. I have known cases where apparently for the sake of economy in first cost, one huge mill was put in, using a tremendous amount of power and pebbles. Whereas, two smaller mills, using far less power and pebbles, would have done more and better work.

In addition to length and diameter, other important factors are:

(*a*) Size of pebbles.
(*b*) Load of pebbles.
(*c*) Speed of mill.
(*d*) Percentage of moisture in the feed.
(*e*) Type of lining in the mill.
(*f*) Diameter of discharge opening.

I will give a few notes on each of these points.

a.—Size of pebbles. These should bear some ratio to the coarseness and hardness of the material to be ground. Large pebbles for coarse material and small pebbles for fine material which has to be ground finer. For charging a new mill it is well to use an assortment of large and small pebbles. Where a mill has been in operation for some time it contains a good percentage of small pebbles, so the pebbles that are fed to the mill to keep up the charge can be larger than would be correct for the whole charge. For grinding ordinary battery pulp crushed through screens varying from 4 to 8 or 10 holes to the linear inch, pebbles varying from 2 in. to 3 in. diameter answer well, while in some short, large diameter, coarse grinding mills, now in use, pebbles up to 5 in. diameter are suitable.

b. Load and Pebbles. The usual practice is to keep the load up to about the axis of rotation of the mill. When the 'Komata' type of liners are used the load should be a little below the axis of rotation. In South Africa the pebble load is about 3 in. above the axis of rotation in a 5 ft. 6 in. diameter mill, (Ball 'Economies of Tube Milling.' Mining and Scientific Press, Sep. 23. 1911).

c.—Speed of Mill. It is very important to have the mill running at correct speed, but it often is difficult to ascertain what this speed should be without making actual tests. The usual speeds given for mills of 4 ft. and 5 ft. diameter are 32 and 28 revolutions per minute, respectively, and these give peripheral speeds of 412 and 439 feet. In all calculations relating to the speed of mills it is important to base them on the diameter inside the liners and not the diameter inside the shell, as the manufacturers usually do, as some types of thick liners, such as silex blocks, very materially reduce the working diameter of the tube mill.

d.—Percentage of Moisture of Feed. This is an important factor and in the early days of tube milling the feed, in most cases, was too wet, as the usual method of dewatering or thickening the feed was a simple pointed box or cone. The millman has now the Dorr classifier and improved cone classifiers and can set the feed as thick as desired. With most ores the percentage of moisture should be about 38% to 45% depending on the specific gravity and the coarseness of the material being ground.

e.—Lining of the Mill. Modern tube mill practice is gradually discarding the old silex block and smooth metal plate liners and is replacing them by two different types of ribbed liners, one of which, like the 'El Oro,' is designed to hold pebbles between projecting ribs and virtually make a lining of pebbles; the other, the 'Komata,' is designed by its special spacing and shape of the bars or ribs, to produce a cascading of the pebbles which greatly increases the grinding efficiency. This spacing of the bars also protects the plates from wear and plates of manganese steel, a little over one inch thick at their centres, where they are thinnest, will last two years with continuous service. The bars are made sectional and the wear shoes, with which they are fitted, are easily renewed and last nine to twelve months. The Komata liners have been adopted by many of the leading mines in the United States and the United States Reclamation Service is now trying them for sand-cement grinding.

The use of these liners makes it necessary to reduce the speed of the mill and also the height of the pebble load. The former should be about 350 ft. per minute, peripheral speed for mills of 4 ft. to 5 ft. diameter grinding ordinary mill pulp, but should be increased if very coarse, thick, material has to be ground. The pebble load should be 4 in. to 6 in. below the axis of rotation in mills 4 ft. to 5 ft. diameter.

f.—Diameter of Discharge Opening. This should be as small as possible consistent with the free discharge of the material being ground in the mill. Usually a 4 ft. diameter mill has a discharge opening 6 in diameter and a 5 ft. mill one of 8 in. diameter. I recently came across some mills 4 ft. 6 in. diameter made by a large machinery concern which had a discharge opening 18 in diameter. The result of the large opening was that the power consumed was 30% above normal, and the pebble consumption was twice what it should have been. When the discharge openings were reduced to about 6 in. both the power and the pebble consumption came down to normal.

For preparing ores for cyaniding the tube mill, in my opinion, excels all other grinding machines, as it is possible with the tube mill to produce a very large percentage of very fine material. On the other hand, by using short tube mills of large diameter and having a system of rapidly classifying out and returning coarse particles to the mill, the tube mill can be used to produce a fairly fine and granular product with but a small percentage of slime.

The great superiority of the tube mill over most other grinding machines is the small amount of attention it requires and now that it has been found that its work is not restricted to fine grinding or sliming only, its use will be more and more extended."—F. C. Brown.—*Mexican Mining Journal*, Feb. 1914, p. 56. (H. A. W.)

MISCELLANEOUS.

Storage of Coal.—"Coals which when heated to 150 degrees in a stream of oxygen show no tendency to increase the temperature are absolutely safe in storage. Those which show a slight tendency to heat up locally, but fail to ignite inside of an hour, are safe enough for storing and transporting on board ship. Those which ignite within one hour and below 150 degrees are relatively dangerous coals. The temperature of self-ignition and the time of heating are necessary to determine the relative degree of danger. In general, the heat value of the coal is not an index as to its tendency to self-ignition. Brown coals are more dangerous in this respect than soft coals. The coals tested showed that the greater the oxygen content the greater the liability of self-ignition."—Demstedt and Schafer.—*The Mining and Engineering Review; Gas World*, 58, 697; March 5, 1914, p. 174. (J. E. T.)

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified at once of non-receipt of Journals and Notices.

Anderson, J. S., *to* Compania Minera Las Vacas, via Caimanes, Ferrocarril Longitudinal, Chile, S. America.

Bluett, H. A. N., *l/o* England; M. M. Kethoen, Lebong Soelit, Benkoelen, Sumatra, D. E. Indies.

Currie, W., *l/o* Rhodesia; c/o Standard Bank, 10, Clements Lane, London. E.C.

Dyason, E. C. E., *to* Equitable Buildings, Melbourne, Australia.

Hardy, J. G., *l/o* New York; Cloudcroft, New Mexico.

James, J. H. D., *l/o* Rietfontein; New Primrose G. M. Co., P. O. Box 193, Germiston.

Mackay, A. N., *l/o* S. America; Trevose, West Byfleet, Surrey, England.

Newbery, J. W., *to* Strode Lodge, Weybridge, England.

Osborne, H. G., *l/o* Cleveland; Golden Kopje Mine, Lomagundi, Rhodesia.

Price, W. S. V., *l/o* Johannesburg; P. O. Maraisburg.

Robinson, V. E., *l/o* Germiston; Goldon Kopje Mine, Lomagundi, Rhodesia.

Rose, A. F., *to* Veldsyde, Ridge Road, Parktown.

Scott, D. F., *l/o* Langlaagte: P. O. Box 15, Rietfontein Mines.

Sim, A. Malcolm, *l/o* Johannesburg; 27, Sussex Square, Brighton, Sussex, England.

Treloar, A., *to* Wheal Kitty and Penhalls, St. Agnes, Cornwall, England.

Wunsch, A. F. W., *to* Gouv. Mynwesen Sassak, Paloppo, Celebes, D. E. Indies,

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OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

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Vol. XIV. JUNE, 1914. No. 12.

Proceedings
AT
Annual General Meeting, June 20th, 1914.

The Eighteenth Annual General Meeting of the Society was held in the South African School of Mines Building, Johannesburg, on Saturday, 20th June, 1914, Mr. A. Richardson (President) in the Chair. There were also present :—

34 Members : Prof. G. H. Stanley, Messrs. H. A. White, J. Gray, H. Meyer, A. Salkinson, F. W. Watson, A. Whitby, R. G. Bevington, W. R. Dowling, A. McA. Johnston, J. R. Williams (Members of Council), A. S. Allam, W. Beaver, H. D. Bell, J. G. Dealy, R. Dures, L. L. Ellis, H. W. Gill, C. J. Gray, B. J. Hastings, W. E. John, W. A. Ledingham, A. Macdonald, J. T. Milligan, P. J. Morrisby, S. Newton, E. A. Osterloh, W. S. V. Price, J. F. Pyles, D. F. Scott, G. H. Smith, A. L. Spoor, W. E. Thorpe, and E. M. Weston.

14 Associates and Students : Messrs. H. S. Arnott, Dr. J. L. Aymard, J. M. Cronin, O. A. Gerber, J. Gibson, R. W. Irwin, F. D. Long, G. F. Mathews, F. J .Pooler, H. E. Ruggles-Brice, H. Rusden, H. Stadler I. R. Trollip, and A. Willcox.

10 Visitors and Fred Rowland (Secretary)

MINUTES.

The minutes of the last Ordinary General Meeting were confirmed.

The President: I will now ask the Secretary to read the Annual Report of the Council and Financial Statement for the past year.

ANNUAL REPORT OF COUNCIL.

Your Council has much pleasure in submitting its report on the work accomplished by the Society during the past year. The Society's work has been greatly hampered during the year by the industrial troubles. The July meeting could not be held owing to the upheaval in that month, while the Martial Law restrictions prevented the holding of the January meeting. There is no doubt at all that the decreased revenue from subscriptions, the decrease in new members, and the smaller attendance at the monthly meetings may be ascribed largely to those disturbances. Your Council hopes that with the better relations which may now be looked for between employers and employees, affairs on the mines, at least, may be more stable, and a greater sense of both security of tenure and responsibility will prevail.

Early in 1915 the Society will celebrate the 21st anniversary of its formation, the first meeting having been held on the 24th March, 1894. Your Council trusts that, failing any other celebration of this coming of age, every member will co-operate in making the coming year memorable in the Society's annals, both in increasing the membership to a larger number than that yet recorded (1,229 in June, 1912), in the subject matter of papers presented, and in increasing its sphere of usefulness to the Industry and the community at large.

Membership.—33 Members have been elected or re-admitted during the year, while 30 Associates and 15 Students have been admitted. The resignations have numbered 66 (38 Members and 28 Associates, a number of whom have relinquished work on the mines to take up farming, in which branch of industry the Council trusts they will meet with success). 21 Students have also either been transferred to a higher grade or relinquished their connection with the Society.

The Council regrets to record the following deaths during the year : Mr. J. P. Fuller, Dr. Johnstone Brown, and Mr. W. D. Williamson (Members) and Mr. C. A. Damant (Associate).

For the non-payment of subscriptions for the past two years, 60 Members and 46 Associates have been struck off the roll. Every endeavour has been made to recover the amount of subscriptions outstanding; this year no fewer than six special notices, besides the usual reminders in the monthly Agenda papers having been issued to those in arrears. A large number of the foregoing cannot be traced owing to their changing their address without notifying the Secretary. The total membership of the Society is 951, as against a total of 1,070 reported last year, a most regrettable decrease of 119. The usual comparative statement will be found here.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.	June, 1909.	June, 1910.	June, 1911.	June, 1912.	June, 1913.	June, 1914.
Hon members	5	5	5	8	14	9	13	18	18	16	17	18	21	22	20	23	25	25
Life members		1	1	1	1	1	1	2	3	4	7	7	8	9	9	9	10	10
Members ...	49	81	101	197	281	291	441	639	680	680	683	618	630	731	711	733	618	550
Associates ...	—	—	—	180	238	243	281	338	321	314	342	357	342	402	380	442	396	351
Students ...	—	—	—	—	—	—	—	4	8	4	7	18	20	60	57	22	21	15
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224	1,177	1,229	1,070	951

Finance.— It is much regretted that it is not possible to place before you the usual statement of accounts at this meeting. The Exhibition being held during the last fortnight of May, and the clearing-up work in connection therewith, coupled with the fact that the Secretary has, most unfortunately, been ill practically ever since, has prevented him from completing the making up of the books. It is therefore proposed to place the statement before you at the next meeting.

Journal and Papers.—The Journal continued during the year fully to maintain its position as an efficient vehicle for the distribution of technical information among members; and several important original papers, notes, and a generous supply of abstracts, culled from the best technical literature of the day, enabled them to keep their knowledge abreast of modern developments in the several departments in which the Society specializes. The Council considers that money expended on the Journal, provided that the resources of the Society are not unduly taxed, is well laid out; inasmuch as the Journal is the only means of bringing many of those members who are unable to be present at the monthly meetings into touch with the latest advances of technology. The Council hopes that members will increase the utility of the Journal by contributing to it papers or notes dealing with any point of technical practice with which their duties render them specially familiar; even a few lines describing some new or useful device may prove of the greatest service to members.

The following is a list of the papers and notes read before the Society or published in the Journal during the year:—

July 1913. "The Determination of Gold in the Presence of Iridium and Allied Metals in Materials such as Black Sand," by (a) Mr. Jas. Gray, and (b) Mr. Chris. Toombs.

August, 1913. "The President's (Mr. Alex. Richardson) Inaugural Address."

"Mining Copper Ores at Messina," by Mr. J. A. Woodburn.

September, 1913. "Improved Taps for Burettes," by Prof. J. A. Wilkinson. (Note.)

"The Determination of the Acidity or Alkalinity of Waters—a Study in Indicators," by Dr. J. Moir.

"The Sandfilling of Mines," by Messrs. W. A. Caldecott and O. P. Powell.

"A System of Keeping Mine and Mill Accounts, Costs and Metallurgical Records," by Mr. M. W. Maclachlan.

"Ventilation of the Mines of the Rand: the Problem of Obtaining Healthier Conditions," by Mr. G. H. Blenkinsop.

October, 1913. "Notes on a Proposed Method of Assay of Tin Ores with some remarks on the Electrical Conductivity of Minerals," by Mr. A. Whitby. (Note.)

"Electric Blasting," by Messrs. Wm. Cullen, Thos. Donaldson, and W. Waters.

November, 1913. "Notes on Waste Rock Values," by Mr. Robert Dures. (Note.)

"The Natural Soda Deposits of Africa, with some notes on the Alkali Trade," by Mr. John Watson.

"Explosives in Hand and Machine Labour Stopes," by Mr. T. D. Delprat.

"The Effect of Charcoal in Gold-bearing Cyanide Solutions, with Reference to the Precipitation of Gold," by Mr. Morris Green.

December, 1913. "The Solubility of Iodine in Sodium Iodide Solution," by Mr. Harold W. Gill.

"The Union Patents Bill," by Messrs. A. L. Spoor and W. E. John.

January, 1914. "The Prevention of Dust in Underground Workings," by Mr. B. C. Gullachsen.

February. 1914. "Primer Clip for Explosives," by Mr. Wm. Cullen. (Note.)

"Description ot au Electrical Parting Apparatus," by Mr. F. Wartenweiler. (Note.)

"A Method of Assaying Concentrates and Battery Chips for Gold and Platinum Metals," by Mr. Andrew F. Crosse.

March, 1914. "Fine Screening," by Mr. A. McA. Johnston. (Note.)

"Relation of Falls of Roof in Collieries on the Middelburg Coalfields to Weather Changes," by Mr. C. J. Gray.

April, 1914. "The Witwatersrand Earth Tremors," by Mr. H. E. Wood.

May, 1914. "Notes on Hydraulic Classifiers and Classification," by Prof. G. H. Stanley.

"Rock Temperatures," by Mr. E. J. Moynihan.

June, 1914. "Analysis of a Cyanide Solution from the Meyer and Charlton Mine," by Mr. H. E. Williams.

"The Detection of the Platinum Metals in Cupellation Beads by Means of the Microscope," by Messrs. C. O. Bannister and G. Patchin.

Mining Exhibition.—The Third Mining Exhibition was held during the latter half of last month, and by the courtesy of the Defence Force Authorities the Volunteer Drill Hall was again placed at our disposal. The Exhibition Committee, which consisted of Mr. Cullen (Chairman), Prof. G. H. Stanley, Mr. J. E. Thomas, Mr. Andrew F. Crosse, Dr. J. Moir, with your Secretary, Mr. Fred Rowland, worked well to ensure its success, and it was even more successful than the previous exhibitions. Greater interest was evinced in it this year by reason of the kindly co-operation of the "Safety First" Committee of the Rand Mutual Assurance Co., and the South African Red Cross Society, both of which offered valuable prizes and medals for the best exhibits in Safety First and Ambulance Appliances. As a result the number of exhibits by the small inventor on the mines and elsewhere was practically doubled, as compared with those of former years while the number of commercial firms exhibiting was practically the same as before. The attendance also was most encouraging, and the help and assistance of the South African press, most freely given, largely contributed to the success of the Exhibition. The Exhibition was opened on the 16th May by Mr. Evelyn A. Wallers, President of the Chamber of Mines, which body again assisted the Society in its work by a financial grant. It is not possible at this moment to state the financial results of the Exhibition, but it is certain that the expenses will be fully met by the receipts.

A feature of the Exhibition this year was the lectures given by Mr. A. M. Archbold (Deputy Inspector of Mines), Mr. Wm. Cullen, and Dr. L. G. Irvine, all of which were large attended and highly appreciated by those present. The Council would like to take this opportunity of thanking Major Jones, D.S.O. (of the Defence Force), Mr. H. Neuhaus, and Mr. J. F. Bilbrough (of the Safety First Committee), Prof. J. A. Wilkinson, and Dr. L. G. Irvine (of the S.A. Red Cross Society); the lecturers and the Exhibition Committee for much invaluable assistance in contributing to the success of the Exhibition.

Awards.—The Society again awarded prizes to successful evening class students at the School of Mines and Technology in Chemistry and Metallurgy. The results obtained at the last examinations, which were notified in the December Journal, attained a higher level of excellence than those previously attained, and the Council is gratified at the results. The prizes offered by the Society to its Student Members elicited no response this year.

Visits and Lectures.—Visits were made to the Union Observatory in October, and the Crown Mines in March, and were well attended by the members, who greatly appreciated the hospitality extended to them, as well as the opportunity afforded of increasing their technical knowledge.

Dr. G. Lindsay Johnson very kindly gave a most interesting lecture to the Society on Colour Photography on the 14th March.

The Annual Dinner, which for the first time was attended by ladies, was held on the 22nd November, and was highly successful.

The S.A. Red Cross Society kindly undertook the arrangements for the Ambulance

Shields Competition in December last, when teams from the Luipaardsvlei Estate and G.M. Co. were successful in securing both the Underground and Surface Shields. Owing to the lack of entries, it was not found possible to hold the competition for the Colliery Shield, which was most regrettable. It is hoped that during the coming year the competitions for all three Shields will be the means of increasing the interest and efficiency in Ambulance Work, which is so greatly to be desired.

The Society has received invitations to attend the Sixth International Mining Congress in London in July, 1915, and the International Engineering Congress in San Francisco in September, 1915, which invitations have been accepted by the Council.

Mr. R. S. G. Stokes very kindly represented the Society at the International Geological Congress held last year in Canada.

The Draft Union Patents Bill was brought to the notice of the Society, and the Council has taken joint action with other technical Societies in laying their views before the Minister.

The usual table of attendances at the eleven Council meetings held during the year, at which the average attendance was 13, is given hereunder:—

A. Richardson	9	F. Wartenweiler	2
G. H. Stanley	10	F. W. Watson	9
J. E. Thomas	10	John Watson	9
H. A. White	7	A. Whitby	7
J. Littlejohn	7	J. A. Wilkinson	10
T. Donaldson	5	R. G. Bevington	1
K. L. Graham	4	W. A. Caldecott	1
J. Gray	10	A. F. Crosse	1
H. S. Meyer	7	W. Cullen	4
E. Pam	8	W. R. Dowling	6
A. Salkinson	4	A. McA. Johnston	2
C. Toombs	9	J. Moir	1
J. Yates	1		

The various Committees of the Council have met frequently during the year, and your heartiest thanks are due to the members of them, who have done so much hard work in the interests of the Socitey.

The Society is again indebted to the Corresponding Members of Council, particularly to Mr. G. T. Holloway, for his many kindnesses to members visiting London, for their help and assistance during the year.

The thanks of the Society are due to the local Press for reporting its proceedings, and also to the Council and Staff of the School of Mines and Technology for their ready assistance in many ways.

The President: Before proposing the formal adoption of the Report, there are a few remarks I would like to make. You will notice that the membership has dropped off during the year to the number of 110. This is really due to the fact that we are passing through rather bad times, and the Society suffers with every other public and private body. Also, we are now rather more strict with regard to those members who are in arrears with their subscriptions. We give them two years grace in which to pay, and I think, after that time, it is not much use keeping them on the books.

There is one point with regard to the papers and *Journals*. We should indeed very much like some of the younger members to come forward and contribute a few papers, or short simple notes, on any little device or on any departure from current practice which has come to their notice in the course of their daily duties. These short contributions are often of the greatest service to members, and, in addition, it gives the younger members an opportunity of coming forward and bringing themselves to the notice of the general body of the Society; because we are anxious that we should have a large number of runners-up for vacancies on the Council. We cannot expect those who have served on the Council for a number of years, and who are busy men, and who have freely given us of their time, to do so for ever, and when they drop out we should like to know what material we have available to draw upon to fill their places; and by contributing papers and joining in discussions they will enable us to judge who are the most likely men to propose for future Councils.

The papers that have been read during the year have been very interesting, and some of them have provoked a vigorous discussion, which is a thing we rather enjoy. The visits and lectures have also been very interesting; and, generally, the year has gone through quite smoothly and well. I now formally move the adoption of the Annual Report.

Mr. J. R. Williams *(Past President)*: It gives me very great pleasure to second the adoption of the Report. Unfortunately, I have been so much away from Johannesburg that I have not been able to give that attention to the Society that I used to in the past, when, believe me, I always attended if possible. I may say that the Society is doing what we started out about 20 years ago to do. One of our biggest ideas, Mr President, at that time was to impart knowledge to others. I well remember Ruskin's words: "The greatest thing a man can do in this world is to see something and, having once seen it, to put it in the clearest, simplest language for the benefit of his kind." During my visits abroad, whenever I came across technical men, they always told me of the pleasure with which they read our Journals—the original papers and discussions; and many of them particularly drew attention to the very great number of extracts that we have from other scientific journals, and so on. So there is no doubt in my own mind that the Society is continuing, as it has done in the past, to do good work. Sometimes, when I come here I am sorry to see that the attendance at the meetings is not up to what it was in the old days in the Chamber of Mines Building, when we used to think we were not doing very well if we had not an attendance of 100; but if everyone studies their Journal I feel that we are doing the bit we set out to do.

The adoption of the Report was carried unanimously.

SCRUTINEERS' REPORT AND DECLARATION OF ELECTION OF OFFICERS AND COUNCIL FOR ENSUING YEAR.

The President: The next business is the Scrutineers' Report and declaration of election of Officers and Council for the incoming year.

Johannesburg,
20th June, 1914.

The President,
Annual General Meeting of Members.

Dear Sir,

We have to report that we have this day examined 142 ballot papers, handed to us by the Secretary.

Our scrutiny enables us to declare the result of the ballot as follows:—

President:
Prof. G. H. Stanley (unopposed).

Vice-Presidents:
Mr. E. Pam.
Prof. J. A. Wilkinson.
Mr. J. E. Thomas.

Hon. Treasurer:
Mr. James Littlejohn (unopposed.)

Members of the Council:
Mr. H. A. White. Mr. F. Wartenweiler.
Mr. F. L. Bosqui. Mr. R. A. Barry.
Mr. F. W. Watson. Mr. G. Hildick Smith
Mr. A. Whitby Mr C. Toombs.
Mr. J. Gray. Mr. M. T. Murray.
Mr. John Watson. Mr. H. Meyer.

Yours faithfully,
P. T. MORRISBY.
S. NEWTON.
E. A. OSTERLOH.
F. J. POOLER.

The President: I must congratulate the Society on having elected Professor Stanley as its President for the coming year. He is a gentleman possessing high scientific attainments, and, in addition to that, he is a very hard worker; the amount of work which he has done for the Society—which only we of the Council know—has been very great indeed during the past year, as well as during previous years, and I think you are extremely fortunate in obtaining his services as President. He will be ably supported by the Vice-Presidents and Members of the Council you have elected to assist him. I feel sure that the year just about to start will, under such direction, prove to be one of the most successful that the Society has ever experienced.

Prof. G. H. Stanley *(Vice-President):* Mr. President and gentlemen, I can only say I thank you very much indeed for the very kind way in which you have received this announcement, and I say also that I very greatly appreciate the honour which the Society has conferred upon me. I think, Mr. President, you have rather overrated the amount of work I have been able to do in the past. But whatever work I have been able to do has certainly been of a very pleasurable character, and it has been a pleasure to me to do it, and if, in the incoming year and the years to follow I can further the interests of the Society by doing a little more work on its behalf, I shall consider it a privilege to do so. I am sorry to see that none of our newly-elected Vice-Presidents is here to-night. I very much regret the absence of the name of our friend Mr. White, from the list of Vice-Presidents for the year; but I believe it is largely his own wish that he should drop out this year. As I have said, the newly-elected Vice-Presidents are not here, and I therefore take pleasure in thanking you, on their behalf, for their being elected also.

Mr. G. Hildick Smith *(Member):* As a newly-elected member of the Council I want to thank you all, on behalf of the newly-elected members, for the honour which you have done us in electing us as members of your Council. I am sure we shall all try to do our best for the Society during the coming year. The most important thing, I think, for us to do is to try and get new members. We should all make a special effort in this direction. There is one point referred to in the retiring Council's report, and it is a very serious one, to my mind—and that is that there appears to have been no papers at all written by Students, for the prizes which are offered by this Society. Students I think fail to understand what they can learn by trying to write a paper. If they only try and start a paper it will help to get their ideas into line, and whether they read it before the Society or not they will find that it has helped. I think Students are very slack if they do not write papers; and they should certainly do so if they want to get on in their work. I think we should look round now to get members to promise to write papers for the different sections, whether Chemical, Metallurgical or Mining. Personally, I would like to start now and promise some small contribution on mining for the September meeting. I think if each of us now promises to write something for our respective sections we shall make a good start for the new year.

NEW MEMBERS.

The President: The next business is to elect scrutineers for the ballot for new members.

Messrs. W. R. Dowling and A. McArthur Johnston were elected scrutineers, and after their scrutiny of the ballot papers, the President declared that the candidates for membership had been elected, as follows:—

Campbell, John Alexander, Dynamite Factory, Modderfontein. Chemist.
Friel, Alfred R., 25, Kapteyn Street, Hospital Hill, Johannesburg. Medical Practitioner.
Lind, John Conrad, Highland Street, P. O. Denver. Manufacturer.
Reid, George, P. O. Box 26, Randfontein. Surveyor.

The Secretary announced that since the last meeting of the Society the following Associates had been admitted by the Council:

Allan, David Lyall, Dynamite Factory, Modderfontein. Chemist.
Bond, Harold, Luipaardsvlei Estate & G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Head Sampler.
Lundholm, Gösta, Dynamite Factory, Modderfontein. Chemist.
Milliken, Andrew, Dynamite Factory, Modderfontein. Chemist.
Wright, Lawrence Alexander, Grenfell Camp, Messina. Mill Foreman

General Business.

VISIT TO DE BEERS.

The President drew the attention of Members to the forthcoming visit, by the kind invitation of the General Manager of De Beers Consolidated Mines, Mr. A. F. Williams, to Kimberley, during the week July 6th to 11th.

ELECTION OF AUDITOR.

Mr. McArthur Johnston *(Past President):* One of the pleasantest duties that can devolve upon any member is to propose the election of our honorary auditors. These have been so long part and parcel of our Society that I think we must congratulate ourselves that we are able to elect these gentlemen from year to year. It gives me great pleasure to propose that Messrs. Aiken and Carter be re-elected our honorary auditors for the ensuing year.

Mr. W. R. Dowling *(Past President)* seconded, and the motion was carried by acclamation.

VOTE OF THANKS TO SCRUTINEERS.

Prof. G. H. Stanley *(Vice-President):* I do not think we should pass on without passing a vote of thanks to the Scrutineers who had the election of Officers and Councillors in hand this morning. They did their duty admirably. The Scrutineers were Messrs. P. T. Morrisby, S. Newton, E. A. Osterloh, and F. J. Pooler.

Mr. F. J. Pooler *(Associate):* On behalf of the scrutineers, I thank you for the vote of thanks. It is rather a dry job, but the Secretary did his best to "liquidate" it. I have been a scrutineer for some years, and I do not think there have ever been fewer spoilt papers.

THE SECRETARY.

Mr. Hildick Smith *(Member):* I should like to propose a vote of thanks to our Secretary, Mr. Fred Rowland. I believe he has been very unwell lately, and for that reason he has not been able to get out the statement of accounts in time for this meeting. Mr. Rowland has had a very busy year with Red Cross work and work in connection with the Mining Exhibition. We can only hope that the short holiday he is going to take now will restore him to perfect health again and fit him for work during the coming year.

The Secretary: I thank you very much, gentlemen.

EXHIBITION COMMITTEE.

Mr. H. A. White *(Vice-President)*: I should like to take this opportunity of moving, on behalf of the Society, a hearty vote of thanks to the Committee who were responsible for the very successful Exhibition that has recently been held, and to others who rendered such valuable assistance as mentioned in the Council's report. I was only able to get in to see the Exhibition once, and that was during the dinner hour, and there were quite a number of people there, including three inventors, each of whom tried to convince me that his little thing was the *clou* of the Exhibition! I think we must not omit a special vote of thanks to the Secretary for the very successful way in which he performed his very arduous work.

The Secretary: I wish to thank you, gentlemen, for the way in which you have received this vote of thanks. You will learn with regret that Mr. Cullen, who was Chairman of the Exhibition Committee, and who did a great deal of work—how much only I know—is lying ill in Capetown. I had a letter from him this morning say that he was in a Nursing Home. He regrets that he does not know when he will be able to get back. I think if we could send him a note of thanks for what he has done it would be some small recognition of his hard work on your behalf in bringing the Exhibition to a successful issue.

PRESENTATION OF THE BRONZE MEDAL OF THE SOCIETY.

The President: I now have a very pleasant duty to perform in presenting the Bronze Medal of the Society to Mr. E. Rosenberg for Stage III. in Practical Inorganic Chemistry at the Evening Class Examinations at the School of Mines and Technology. Mr Rosenberg has already received the prize of £5 worth of technical books, and I now ask him to come up and receive the Society's Bronze Medal.

Mr. Rosenberg received the medal amidst applause.

JULY MEETING.

Mr. F. W. Watson *(Member of Council)*: At the last Council Meeting the question as to whether we should hold our Meeting in July or not was discussed, and the Council felt that the matter ought to be left to the decision of the Annual General Meeting. Last year this question was brought up, and there was no meeting in July owing to the Industrial disturbances, and the Council felt this year that, owing to the illness of the Secretary and his going away for a holiday, there was sufficient justification for allowing the July meeting to lapse. We would like an expression of opinion from the members present at this Annual Meeting. I therefore beg to move, on behalf of the Council, that the meeting in July be abandoned.

The President: I would like to point out in connection with this that the visit to Kimberley in July may involve quite a lot of work and arrangement. There is another thing; the July Meeting, as a rule, is very poorly attended; it is just about the time that people are getting away out of the cold winds and dust-storms down to Durban and other coast resorts, and I do not think that we should be doing any harm by dropping the meeting, because the Journal will appear as usual, and any papers that have come forward for reading will appear in it. There is another point I may mention, it gives the incoming President time to prepare his Inaugural Address. I know what that means myself, and, if the meeting had taken place in July last year, I should have had the greatest difficulty in getting it ready; so, on the whole, I think it would be a very good plan to drop the July meeting, at any rate for this year, as an experiment; if it is found not to work very well then, of course, we can go back to the old arrangement next year.

Mr. E. M. Weston *(Member)*: I beg to second that.

Mr. W. S. V. Price *(Member)*: Is this to be taken as a precedent?

The President: No. We just want to try it this year and see how it goes, and if it does not answer we will return to the old arrangement next year.

Prof. G. H. Stanley *(Vice-President)*: There are a good many points in favour of this proposal. Some members, however, may have a grievance if this meeting is dropped, and if any member has anything to urge against dropping the meeting and in favour of holding it as usual, then, I think, he might say so now.

On being put to the meeting it was declared carried unanimously that the July meeting this year be dropped.

ANALYSIS OF A CYANIDE SOLUTION FROM THE MEYER AND CHARLTON GOLD MINE.

Mr. H. E. Williams *(Associate)*: The analysis given below is that of the strong circulating liquor, and it will be observed that the total cyanogen expressed as sodium cyanide is small, notwithstanding the fact that this liquor is (at least so we are told) one of the strongest on the Rand.

It certainly seems to me from a study of this liquor that the double zinc cyanide that is formed must for the greater part be brought into play as a solvent for gold, the zinc being removed from solution and alkaline cyanide liberated by some reaction, or reactions, that take place during the operation of gold extraction, and there are many such possible reactions, otherwise I cannot account for the comparatively low amount of this compound which is present in the liquor.

And, again, I think that the total free cyanide returned by the usual Liebig test, in the presence of so many other double cyanides, is much too high as an expression of the actual free cyanide present, even when potassium iodide indicator is used. The figure I have obtained by this test was considerably in excess of that which I actually found to be the case.

It is, of course, quite impossible to speak with any certainty on this point until a very large number of these solutions have been examined in this manner.

ANALYSIS :

	Per cent.	Parts per 100.000
Sodium cyanide	0·03880	38·80
Sodium ferrocyanide ...	0·00570	5·70
Sodium zinc cyanide ...	0·04080	40·80
Sodium cuprocyanide ...	0·00660	6·60
Sodium nickel cyanide ...	0·00419	4·19
Sodium aurous cyanide .	0.00091	0.91
Sodium cobalticyanide .	0.00290	2.90
Sodium cyanate...	0·00780	7·80
Sodium thiocyanate ...	0·01458	14·58
Sodium sulphate	0·16630	166·30
Sodium chloride	0·04614	46·14
Calcium sulphate	0.06600	66.00
Calcium hydroxide	0·02440	24·40
Free ammonia	0·00136	1·36
	0·42648	426·48

THE SOLUBILITY OF IODINE IN SODIUM IODIDE SOLUTION.

(Read at December Meeting, 1913.)

By Harold W. Gill, B.Sc.(Lond.), A.I.C. (Member).

REPLY TO DISCUSSION.

Mr. H. W. Gill *(Member):* The discussion on this paper can be divided into two distinct parts. Firstly, a discussion as to what was meant by the term concentration, and, secondly, as to the suggestion which was thrown out as to the existence of a definite compound in iodine-sodium iodide solutions. The first point was one on which there could be no valid discussion, as it was quite recognisable that any doubtful point of that kind could have been cleared up in the reply. The vigorous discussion of the second point, namely, a mere suggestion which was thrown out as a result of the work done, and which had not in any way been followed up or investigated, seems almost waste of time unless chemical evidence could be brought to bear upon it.

Dealing with the first point, a considerable number of guesses as to what was the real meaning were made, and it was shown that each assumption enabled results different from those given (in the original paper) to be made. Notwithstanding the fact that this point could be cleared up, it was even stated that two well-known gentlemen could not possibly make use of the figures given —a point which was really without foundation, and erroneous. Without considering each assumption in detail, it may be pointed out that a considerable amount of space in our valuable journal could have been saved if this criticism had been replaced by a question. The solutions were prepared by dissolving x gm. of sodium iodide in water and making up to 100 cc. at 20° ($x=5$, 10, 15, etc., gm). This is not a very extraordinary way of making up a solution, indeed it would occur to some that it is what might be referred to as the usual method. Before the experiments were carried out it was carefully considered what would be the most convenient means of expressing concentration. On careful consideration it will be seen that the one adopted is probably as practical as any. Suppose an x% solution of sodium iodide solution be prepared by any method, i.e., x gm. of NaI + 100 gm. of water, or x gm. of NaI + 100 cc. of water at t°C. etc.; then if iodine be dissolved in that solution the volume will almost certainly be changed, and hence the solution will no longer contain x % of NaI but $(x \pm dx)$, depending on whether the solution contracts or expands on taking up iodine. Variations of a similar kind will take place at any and all temperatures, so that, practically speaking, it would be needless and useless to attempt to prepare such solutions, and consequently much of the criticism based on this point was ill-considered. Finally, in this connection it may be pointed out that this problem differs very greatly from one in which the solubility of a substance is determined in a solvent such as water or alcohol, etc. Here the solvent itself is a solution, and there seems to be no general plan for expressing the results obtained. Thus in one well-known text-book two tables are given, one in which the amounts of "solvent-salt" and solute present in 1 litre of solution are given, and in the next table the amounts of solute which dissolve in 1 litre of solution prepared by dissolving certain amounts of "solvent-salt" per 1,000 gm. of solvent at 25° C. are given. In two other places in the same book the solubility has been determined and the concentration expressed in exactly the same manner as was done in the paper under discussion. These facts show that there is no glaring error in the method adopted. No doubt a very useful result would be obtained if all workers used the same method so that their results would be strictly comparable, as in the case of the determination of the solubility of a substance in a simple solvent. Here there is, in modern determinations complete co-ordination, the results being expressed in grams of anhydrous solute per 100 gm. of solvent [Landolt-Bornstein-Meyerhoffer's Tabellen; Kaye and Laby, Physical and Chemical Constants; Ostwald's Physico-Chemical Measurements]. From this, too, it is seen that the authors of these well-known works of reference do not agree with the criticism on this paper.

Passing on now to the criticism of the suggestion that NaI_2 $2H_2O$ exists in solution, had more time been spent in reading the original paper before criticising, the remarks would probably have been a little less caustic. The author has always compared a chemical theory to an Aunt Sally, which one chemist puts up for the other to throw *facts* at. That theory is best which requires the most knocking down, but it must be noticed that it is of no use to look on from a distance and say how easily it can be bowled over. Before dealing with any details of the criticism, it may be well to state the facts which were known before the present paper was written. Moissan, in his "Traité de Chimie Minerale." III., p. 299, summarises the facts as follows: "La solution aqueuse d'iodure de sodium dissout facilement une quantité notable d'iode, mais cette solution se différencie de la solution d'iodure de potassium en ce qu'elle laisse dégager l'iode avec une grande facilité." In addition to this, almost every text-book mentions the fact that a solution of sodium iodide at normal temperatures contains NaI $2H_2O$, and that at some higher temperature this suddenly changes to NaI, this being indicated by a sudden change in the physical properties of the solution, and by the fact that at normal temperatures NaI $2H_2O$ crystals are obtained, whilst above the "critical" temperature NaI anhydrous is deposited. In the paper under discussion it is stated that "*It would seem* that a compound of NaI_2 $2H_2O$ exists in solution," and later on "*It is probable* that the compound, etc." It is clearly evident even to the casual reader that a very carefully guarded and cautious statement was being made. Anyone should have seen that there was insufficient evidence given to draw any definite conclusions. No experiments were done to prove or disprove the the presence of any compound, because that was not the object of the paper. Before leaving this subject it will be well to point out a few of the errors and unjustifiable assumptions made during the discussion.

Firstly, it is stated that "very conclusive proof will be required before we accept the statement that a compound with two molecules of water exists in aqueous solution." Any elementary text-book on general and physical chemistry will furnish the proof required. Herewith are a few references:

1. Senter. Outlines of Physical Chemistry, p. 93.
2. Findlay. The Phase Rule and Its Applications, p. 131.
3. Nernst. Theoretical Chemistry, Book IV., Chap. II.

Secondly, the author was well aware of the work of Jakowkin on potassium iodide, and it was an unjustifiable assumption to suppose the contrary.

Thirdly, it is even necessary to point out the danger of "analogy." It is a good servant, but a dangerous master, and really no amount of evidence on potassium compounds yields that proof about sodium compounds which is so righteously and earnestly sought after in other parts of the criticism, although of course no one would deny the value of analogy when other evidence is lacking.

The fourth point raised is the validity of the method of analysis should the determination of the formula of any compound present be the ultimate object of the experiments. Consider the example taken in the discussion

$$KI_3 \longleftrightarrow KI + I_2$$

The solution, therefore, contains dissociated and undissociated KI_3. Now, provided the dissociated products do not leave the solution, the ratio of I_2/KI is not affected, and hence the results are valid, in spite of a certain amount of dissociation.

Again when ammonium acetate is dissolved in water it is partially hydrolysed. $CH_3COONH_4 + Aq \longleftrightarrow CH_3COOH + NH_4OH$, but this does not prevent the formula for the compound being deduced from the percentages of NH_4OH and CH_3COOH found in the solution.

Fifthly, in a paragraph dealing with results at 25° C., and only 25° C., it is suggested that $NaI_2\ 2H_2O$ is present. It is now humbly submitted that no amount of evidence taken at some other temperature affects in the slightest degree the results obtained at 25° C.

The sixth assumption is that the author is ignorant of the method of calculation of the formula of a substance from its percentage composition. The author was perfectly well aware at the time of writing that the figures given in his paper worked out to $NaI_{2\cdot2}$ *(circa)*, and in view of the known facts it was thought that it was not unduly stretching the evidence to call this NaI_2. Whether this was correct or not was, and still is, an open question.

The seventh assumption is that iodine-sodium iodide solutions contain a mixture of NaI_3 and NaI_5. For this statement there is no jot of evidence, unless it be based on analogy with the corresponding potassium compounds.

The eighth point that "concentration" is a technical term meaning "gram molecules per litre of solution" is erroneous. In the first place concentration is a non-technical as well as a technical term (see Webster). It seems that molar concentration has been confused with concentration, pure and simple; indeed, such an extremely narrow definition of such a general term would almost necessitate the coining of a new term, for of course, "strength" cannot be used in view of its "ionic" meaning.

With the solubility of iodine in water the paper had no concern, and surely "half and half" is a most unscientific term.

The author has now answered some of the more relevant points which appear in the criticisms, but considers it necessary to enter a strong protest against the discourteous manner in which he considers the criticisms were made. Quite apart from the merits or demerits of the paper, the fact that the Publication Committee accepted it should, he thinks, bind the Council to protect themselves and their authors from misstatements of fact, groundless assumptions, and personalities such as he considers are found in the discussion of the present paper. It seems to the author to be truly lamentable that these contributions were not more carefully censored by the Editorial Committee before publication. Our aim is to be a scientific Society—a group of men gathered together for scientific intercourse far from the centre of learning—a group endeavouring to form an oasis where the scientific traveller may find congenial surroundings for a while in his wanderings, and thus it seems indeed a pity that the words Dr. Moir used in 1903, on page 275 of our *Journal* for that year, could not be carried out more earnestly. Dr. Moir there deplores literary violence, and speaking of a certain contribution, hopes that it will "possibly help to redeem its (the *Journal's*) character from the effects of recent discussions, where there has perhaps been more of a desire to make personalities than to get at facts."

THE DETECTION OF THE PLATINUM METALS IN CUPELLATION BEADS BY MEANS OF THE MICROSCOPE.

By C. O. Bannister, *Assoc.*R.S.M., F.I.C., and G. Patchin, *Assoc.*R.S.M.

Presented by Prof. G. H. Stanley, Assoc.R.S.M., F.I.C.

At a recent Meeting of the Institution of Mining and Metallurgy in London the present authors submitted a paper on Cupellation Experiments, entitled "A simple method for the detection of the Platinum Metals in Cupellation Beads." A considerable amount of interest was taken in this paper, which was illustrated by a number of lantern slides. In view of the work carried out recently by some of the members of your Society on the determination of Platinum Metals in black sands, etc., it is thought that some notes on this subject, amplified in certain directions by results more recently obtained, may be of some value and may give rise to an interesting discussion.

It has long been known that platinum metals in cupellation beads give certain indications of their presence, but very little definite information is obtainable on this subject, and many conflicting statements have been published as to the nature of the indications to be attributed to the various metals.

In the case of platinum for example, a surface crystallisation is visible to the naked eye when a certain amount is present, 2·2 per cent. according to Van Riemsdijk's statement made in 1879, but 1·6 per cent. according to our more recent observations. With more than 1·6 per cent. platinum this crystallisation gives the beads a frosted appearance. Many assayers, experienced in the assay of platiniferous materials, however, claim to be able to detect with the naked eye the presence of smaller quantities than are required to cause distinct crystallisation. According to such observers the beads show a peculiar steely-grey iridescence. This possibility of detection has, however, proved to be very uncertain, for, on submitting a number of beads of known platinum content to experienced assayers, the predictions have proved most disappointing. The indications given by the presence of other metals of the platinum group have proved to be equally disappointing as a means of detection.

In view of the fact that the price of platinum has rapidly and continuously increased since 1894 (from 42s. to 180s. per oz.), and the demand for platinum metals has grown enormously, it is desirable that these metals should be easily detected, especially during the assay for gold and silver. Unless some definite indications of the presence of small quantities of these metals are given to the assayer, he is liable to overlook their presence, especially in the case of platinum and palladium.

The method of detection described in this paper enables the assayer easily and quickly to foretell the presence of these valuable metals, even when present in such quantities as do not give indications to the naked eye.

The method adopted by the authors consists of an examination of the surface of cupellation beads by a low-power microscope fitted with a $\frac{3}{4}$ in. or 1 in. objective. No preliminary polishing, etching or other preparation of the surface is made, the beads being taken direct from the cupels and mounted on microscope slides by means of plasticine. Either oblique or vertical illumination may be used, although the authors prefer vertical illumination.

The plant required is thus seen to be quite simple and inexpensive, and, in fact, is found in most modern assay laboratories.

It is obvious that for microscopic examination, large beads are more suitable than small beads. It has been found possible, however, to detect 1·0 per cent. platinum in silver beads weighing 0.05 grm. and as little as 0.4 per cent. in beads weighing 0.1 grm. In the examination of materials containing an excess of platinum metals over the gold and silver contents, such as some of the "black sands," the assay of which has recently engaged the attention of your members, the authors have found it advisable to add a quantity of silver sufficient to give a good cupellation bead. It is a better practice to ascertain the amount of silver necessary by preliminary experiment and then to add this amount to the subsequent charges during fusion. On the other hand, in the examination of materials containing very small amounts of platinum metals, but more than sufficient gold and silver to enable the bead to be examined, it is often a great advantage to part the bead in sulphuric acid and to cupel the residue with a small amount of silver.

When it is desired to ascertain the presence of platinum metals in material giving minute beads by the ordinary assay methods the best mode of procedure is to collect the parting solutions and residues from a batch of assays and to treat as follows. The silver solution is evaporated carefully in a porcelain basin until quite dry, the residue is then washed into a lead basin and again evaporated to dryness. The lead dish is then scorified and the resulting button cupelled. The bead thus obtained may be examined by the microscope for platinum and palladium. An indication of the presence of palladium will be given by the colour of the parting acid. The gold residues are weighed and cupelled with four times their weight of silver, the resulting bead being afterwards examined.

The authors desire to make it quite clear that they do not suggest that these methods should replace the ordinary assay methods, but rather that they should be used as a means of detecting the presence of platinum metals. When beads prepared by cupelling pure silver are examined under the microscope, a great variation in structure is observed. In some cases crystallization is found to proceed from definite centres as shown in Fig. 1, whilst in other instances distinct crystal grains are to be observed, the result of the growth of crystallites which have grown until interfered with by the encroachment of similar crystallite growths as illustrated in Fig. 2. All the photographs submitted with this paper were taken at a magnification of 100 diameters with the exception of that one relating to the presence of rhodium. In the case of beads prepared by the cupellation of pure gold with lead, distinct crystals are generally to be seen, in some cases even without the aid of the microscope. Fig. 3 shows the type of structure commonly obtained.

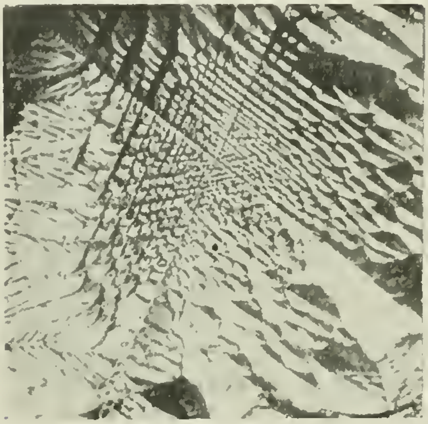


Fig. I.—Pure Silver. 100.

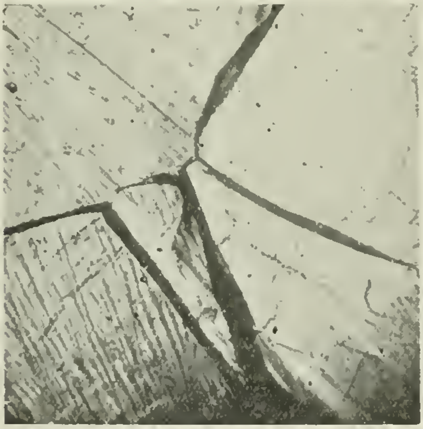


Fig. II.—Pure Silver. 100.

With beads obtained by the cupellation of gold with silver the structure is found to vary according to the relative proportions of the two metals. Beads of high gold content approach in appearance that of pure gold and vice versa.

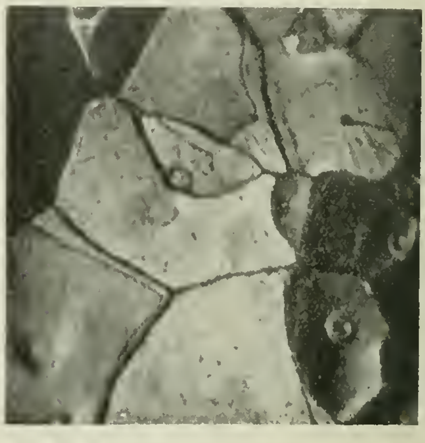


Fig. III.—Pure Gold. × 100.

In describing the effects of platinum on the surface structure of silver beads it is admitted that amounts less than 0·3 per cent. give insufficient indication of their presence. With more than 0.3 per cent., however, there is a tendency for the boundaries of the crystal grains to become distorted and the surface to take on a corrugated appearance. With increasing amounts of platinum these corrugations increase in number until the whole surface of the bead is covered by them. This is well illustrated in Figs. 4 and 5 containing respectively 0·4 and 1·2 per cent. of platinum.

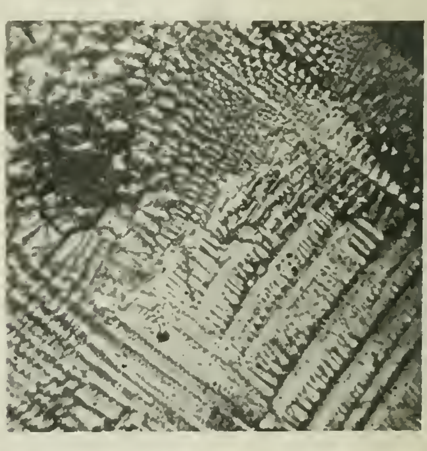


Fig. IV.—Silver containing 0·4% Platinum. × 100.

The effect of platinum on gold beads, although similar, is not so distinct as on silver beads. With the same amount of platinum present corrugations are not so numerous on taking into consideration the whole surface of the bead. It is possible with the lower percentages of platinum, for example 0·5, to find portions of the bead showing no corrugat s whatever. Fig. 6 shows the effect of 1·4 per cent. of platinum on gold. The effect of platinum on beads containing gold and silver is similar to that already described. The structure moreover is more marked than in the case of beads consisting of gold only.

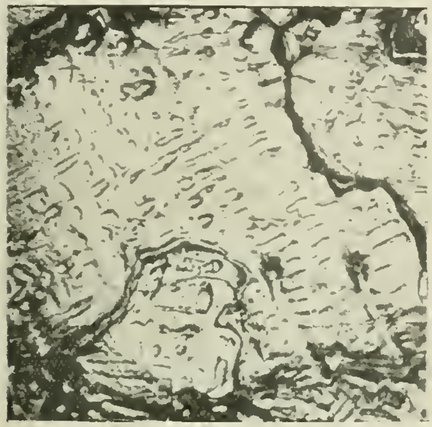


Fig. V.—Silver containing 1·2 Platinum. 100.

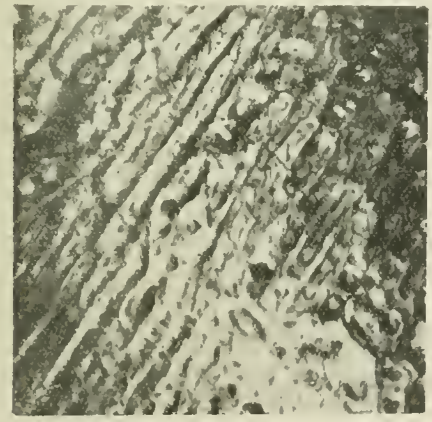


Fig. VI. Gold containing 1·4 Platinum. 100.

With palladium, the surface appearance of the beads is very similar to that produced by platinum as illustrated in Fig. 7, taken from a silver bead containing 2·0 per cent. Unlike platinum, which gives a frosted appearance when more than 1·6 per cent is present, palladium may be present to an extent of 2 per cent. without giving definite indications to the naked eye. Immediately on proceeding to the parting operation, a difference is obtained in the colour of the parting solution. Whether sulphuric acid or nitric acid be used, it will become faintly yellow if palladium is present, deepening in tint as the palladium increases in quantity and as the parting proceeds.

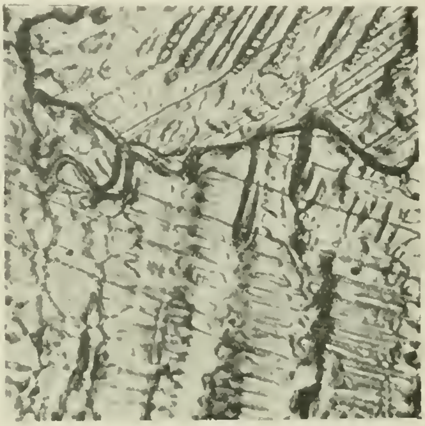


Fig. VII.—Silver containing 2·0 Palladium. 100

Turning now to the effect of iridium on silver beads, it is found that these are always enlarged as compared with silver or silver-platinum beads of the same weight. The shape assumed by these beads is much more spherical both on the upper and lower surfaces giving to them the appearance of having been blown out. Under the microscope the boundaries of the crystals are clearly seen, but the facets are striated in more than one general direction. This appearance suggests that the beads have been subjected to severe internal stresses, and the result may be compared to the faults produced by nature in rocks. In some cases, however, the beads spit, more often from the bottom than from the top, thus tending to relieve the stresses producing the characteristic markings which, however, are not entirely eliminated. On making a cross section through silver-iridium beads which had not spat, a cavity was found in every case, thus supporting the contention that the enlargement was due to some internal force. This force is evidently produced by the liberation of oxygen from the solidifying silver, there being a sufficient thickness of solid crust to prevent its escape with the production of the ordinary spit. The effect of iridium is shown in Fig 8 taken from a silver bead containing 2·0 per cent.

Rhodium is found to have quite a different effect on cupellation beads than either of the other platinum metals.

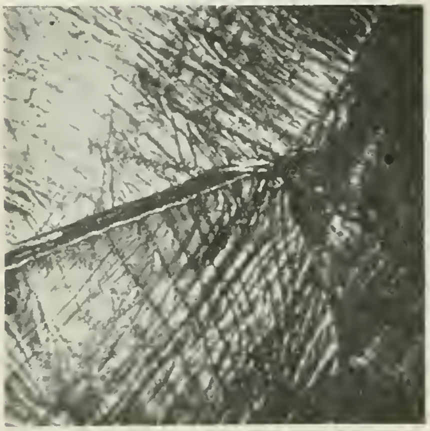


FIG. VIII. Silver containing 2·0 Iridium. 100.

When more than 0·3 per cent. rhodium is present the surface of the beads has a bluish-grey colour which becomes deeper as the amount is increased until the beads become blue-black. With less than the amount of rhodium stated a very beautiful crystallisation visible to the naked eye is produced. This is found to apply equally to beads of gold and gold-silver. Fig 9 illustrates this point, having been taken from a bead containing 0·01 per cent. rhodium and magnified 15 diameters. This extremely coarse crystallisation is probably due to the fact that rhodium prevents the undercooling of the beads, thus reducing the number of centres produced in crystallisation. Another striking phenomenon is the persistent spitting of silver beads when 0.03 per cent. and upwards of rhodium is present. All attempts to prevent this tendency proved futile, and the silver ejected from dark-coloured beads had the characteristic silvery-white colour of pure silver.

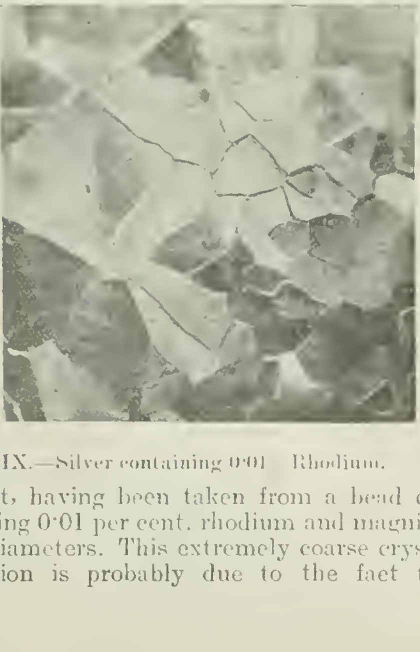


FIG. IX.—Silver containing 0·01 Rhodium. 15.

On repeated cupellation, the rhodium is gradually removed, and it is possible by this means to get rid of so much rhodium from dark coloured beads, that the crystalline surface above noted becomes visible.

In the case of small quantities of ruthenium a black crystalline deposit is always found on the bead. In the majority of cases this deposit is situated at the edge of the bead near the bottom. Unlike the surface film due to rhodium this deposit is distinctly crystalline, and is not so readily eliminated by repeated cupellation.

In conclusion, the authors wish to state that they are aware that complications do arise when two or more of the platinum metals occur together. They consider, however, that valuable indications as to their identity may be obtained by the careful microscopic examination of the surface, coupled with a series of recupellations and partings. In samples of black sands kindly supplied by Mr. G. T. Holloway distinct indications of the presence of the various platinum metals have been obtained from time to time.

ADDENDUM

By Prof. G. H. STANLEY, A.R.S.M., F.I.C.

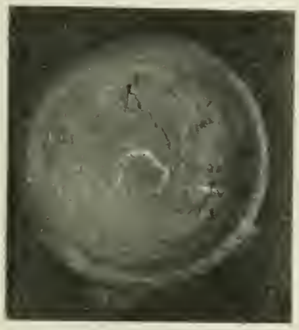
Following on this interesting and valuable communication by my friends, Messrs. Bannister and Patchin, I should like to bring forward a few observations of my own.

I think I first noticed the presence of platinum metals in cupellation beads in 1905 during the assay of some "black sand" from the Robinson Deep G.M.; the effect produced being distinct crystallisation of the surface plainly visible to the unaided eye.

Since that time I have frequently detected such appearances on assaying similar material, though, curiously enough, I have observed that sometimes from the same sample one bead will be free from such indications while another shows signs.

More recently, in the course of experimental work on the concentration of these platinum metals from "black sand," the value of the original materials and various products obtained in "metal insoluble in aqua regia," after the usual cupellation and parting, varied from a few dwt. to about 2,500 dwt. per ton with consequent very disconcerting effects on the assays.

With the smaller amounts the simple crystalline appearance was noted, but as the amounts increased, the beads, instead of remaining coherent, tended to break up, i.e., to remain in disconnected pieces on the cupel, and in extreme cases the metal remained as an irregular network on the cupel surface.



A Typical Case.

It was possible to effect the collection into one button by largely increasing the amount of silver, and when sufficient silver was present to cause this to take place the most notable difference from an ordinary cupellation bead was the presence of a more or less continuous black coating, which was apt to be partially detached during flattening.

This did not resemble in appearance a film, for example, of copper oxide, but its appearance was like that of a thin coating of soot, leading to the belief that it consisted of fine particles of unalloyed metal. No undue tendency to spitting was observed. Unfortunately the microscopic appearance of the surface was not observed, and a cross section with a view to determining the mode of existence of the iridium *i.e.*, whether alloyed or not, gave inconclusive results.

A series of 36 lantern slides, prepared by Messrs. Bannister and Patchin, illustrating the effects produced by the presence of varying quantities of platinum metals in silver, gold, and gold-silver alloys, including the Figures 1 to 9, was then exhibited on the screen.

The President: I am sure we are indebted to Messrs. Bannister and Patchin for a very interesting paper. Is there anyone present who would like to make any remarks on the paper?

Mr. H. A. White *(Vice-President):* I do not think this occasion should be allowed to pass without congratulating ourselves on seeing such a fine collection of slides. The technique of these slides is really equal to the best I have seen from any part of the world.

THE PREVENTION OF DUST IN UNDERGROUND WORKINGS.

(Read at January Meeting, 1914).

By B. C. Gullachsen, M.I.M.E. (Member).

DISCUSSION.

Dr. J. L. Aymard *(Associate):* said he disagreed with the opinion expressed recently by Mr. Kotzé with regard to recent advance in prevention of dust and phthisis. On the contrary, he thought there was no evidence that the efforts made during the past few years have been successful in reducing the steady progress of the disease. He thought that in view of the conflicting opinions still held as to the particular kind of dust causing the disease, experts from England should be called in, and he contended that the employment of steam, as in ganister mines in England, would prove successful.

A METHOD OF ASSAYING CONCENTRATES AND BATTERY CHIPS FOR GOLD AND PLATINUM METALS.

By Andrew F. Crosse, M.Inst.M.M. (Past-President).

(Read at February Meeting, 1914.)

DISCUSSION.

Mr. H. D. Bell *(Member):* Mr. Crosse's paper on "A Method of Assaying Concentrates and Battery Chips for Gold and Platinum Metals" may suggest that the procedure is exceedingly simple, so much so in fact that mine assayers may find themselves called upon to undertake such an assay—in their spare time—and be expected to report the correct result next day. I venture to suggest that assays for platinum metals should be undertaken only by experienced assayers and chemists who have time and the facilities of a well equipped laboratory to do justice to the work. I have found that platinum is dissolved with the silver and cadmium when treating with dilute nitric acid. At first I thought that the silver, etc., and cadmium had not alloyed, and that the nitric acid, acting directly on the silver-platinum metals, took platinum into solution. Repeated tests, giving every opportunity for the proper alloying of cadmium and silver also, dissolving the alloy in hot and cold dilute acids respectively, all showed that platinum goes into solution. When examining for the platinum group of metals it does not appear advisable to fuse with bisulphate of potash in a platinum crucible: the possibilities are obvious. Regarding the precipitation of platinum with magnesium, I have left magnesium ribbon in contact with platinum solution slightly acid with hydrochloric acid over night, dissolved out excess of magnesium, and filtered and found platinum in the filtrate. I have not had the opportunity of making further tests on this point, but no doubt Mr. Crosse will detail the exact procedure necessary for complete precipitation of platinum by magnesium. My investigations prove that the method of assaying concentrates and battery chips for gold and platinum metals suggested by Mr. Crosse is unreliable, and should not be used. When such an important point as the solubility of platinum with the silver and cadmium in nitric acid was not experimented upon, how can it be definitely stated that synthetical tests gave satisfactory results?

NOTES ON HYDRAULIC CLASSIFIERS AND CLASSIFICATION.

(Read at May Meeting, 1914.)

By G. H. STANLEY, A.R.S.M., M.Inst.M.M.
(Vice-President).

DISCUSSION.

Mr. T. Graham Martyn *(Member)*: I have just read Professor Stanley's paper with very great interest, and from considerable personal experience of investigations on the same subject, following a different line, I can appreciate in some measure our great indebtedness to him for publishing the results so far as he has gone. I have no criticism but only gratitude to offer so far as the main part of the paper is concerned, but there is one clause in it which is, I think, misleading, viz., "On the other hand, the underflow product, on account of the hindered settling ratios being much wider than free falling ratios, may contain the heavy mineral of a considerably smaller size than would be obtained under ideal free falling conditions accompanying the gangue minerals: this, if subsequent concentration is to be employed, may be *a disadvantage, as on theoretical grounds separation then becomes more difficult.*" The italics are mine, indicating the particular clause which if not the exact reverse of the truth, comes very near it, for there is a decided *advantage* both practically and theoretically for subsequent concentration in having the heavy mineral considerably finer than the gangue. The finer it is the better; always provided that it is not so fine as to be hurried along in complete suspension over the surface of the concentrating machine instead of settling with the gangue on its surface. If the fine particles settle with the gangue, they find their way to the bottom the more easily the finer they are, and then are propelled by the motion of the surface of the machine in one direction, whilst the coarse particles of gangue are the more easily rolled along by the feed and wash water in another direction *because they are coarse*, and stand relatively high up in the stronger part of the current, just as a big wheel travels more easily over a rough road than a small one. *This is why closely classified products are far more easily concentrated than mixed products, and considerably better even than closely screened products.*

This being the case, it is open to question whether "free falling" products are superior to "hindered settling" for subsequent concentration. In my judgment, at first sight "hindered settling" products would be preferable because the fine heavy mineral should settle with the gangue, and not be in any danger of being hurried along in suspension if the classification had been ideal; but as the ideal cannot be obtained in practice, so that there will always be present some particles finer than there ought to be, it seems to me really preferable to allow a margin of safety by adopting the "free falling" principle, if possible, as advocated by Prof. Stanley.

THE WITWATERSRAND EARTH TREMORS.

(Read at April Meeting, 1914.)

By H. E. Wood, M.Sc., F.R.Met.S. (of the Union Observatory).

DISCUSSION.

Mr. G. Hildick Smith *(Member)*: It may be interesting to some of you to know that the local earth tremors are affecting our underground workings on the Ferreira Deep, causing trouble in the shafts and in the mine generally. When a tremor occurs, the result underground with us is to cause a loud report, often attended with falls of hanging wall, the shock being generally sufficient to extinguish all naked lights in the neighbourhood where it is felt most acutely.

Mr. H. A. White *(Vice-President)*: Are you satisfied that it is not these falls that are causing the tremors, but the tremors that are causing the falls?

Mr. G. Hildick Smith *(Member)*. I think the local earth tremors are due to slight movements occurring along dykes and fault planes. The movement is made possible by reason of the large areas of superincumbent strata, undermined by stoping operations on the central Rand since mining commenced about 1887. Neither sand-filling nor systematic packing will prevent a gentle subsidence of the hanging-wall, portions of which are often under great tension. A general movement, causing an earth tremor, thus gives rise to local cracks in the hanging-wall pillars which often break up and falls of ground occur.

INAUGURATION OF PRESIDENT.

The President: Gentlemen, with your permission I would like to perform a very pleasant ceremony. I regret I shall not be able to be present at the next meeting owing to my immediate departure for England, and before I go I should like to formally introduce your new President to you and welcome him to the chair, and ask him to close this meeting for me.

Professor G. H. Stanley then took the chair.

Mr. A. McArthur Johnston *(Past-President)*: Mr. President, before exercising your new duties and declaring this meeting closed I should like to ask our members here to pass a very hearty vote of congratulation to Mr. Richardson, our retiring President, for the excellent way in which he has conducted the meetings of this Society during the past year. I am sure I need say very little both with regard to the personality of Mr. Richardson and to the way in which he has conducted these meetings. You all know him, or ought to know him, as well now as I do, and I therefore only require to ask you to join with me; but along with this vote of congratulation, I am sorry to say we have to include a farewell note. Mr. Richardson, after a residence of about 25 years on the Witwatersrand, is leaving us and going to try his luck, we might say, somewhere else. I am sure you will all agree with me that we are sorry to lose Mr. Richardson. He has been an excellent member of this Society, and an exceedingly useful man in every part of the Rand particularly, and of the Transvaal, in which he has been situated so long. We all wish him a very pleasant trip Home, and when he reaches Home, or wherever it is that he is determined to make his home, we hope that very good luck and prosperity will attend him in the future.

Mr. Alex. Richardson *(Retiring President)*: I wish to express to you Mr. McArthur Johnston, and to you gentlemen, my deep appreciation of the very kind remarks you have made regarding me, and your hearty endorsement of them. If there is any complaint I have to make it is that by this assurance of your cordial friendship you make it all the harder for me to bring to an end my long association with the Rand and all its ups-and-downs, and my 14 years' close connection with the Society in all its lively and up-to-date work. I am sorry that I shall be unable directly to assist Prof. Stanley and his colleagues in the forthcoming year to any great extent, but I hope that opportunities will arise on the other side of the water to help the Society along, and should they do so, I shall have the greatest pleasure in doing all I can to keep up the high reputation that the Society has achieved on the other side. I do not think there is anything I can add except to thank you all most heartily and to express the hope that, if I cannot see all of you, that I certainly shall see a great number of you before long on the other side, not as leaving the country, but as coming Home for a good holiday.

The President: I do not feel able, at present, to add anything to the thanks I expressed previously for the honour you have done me in electing me to this chair. With regard to the membership, we only want about 250 to equal or surpass the previous record, and I think that one of the ways in which we should signalise the attainment of our majority would be to make a very special effort to increase our membership. There are now nearly 1,000 members, so that if every four of them, as a result of their efforts, can only bring in one more member, we shall just about succeed. Of course, we should like every member to bring in a new member, but, perhaps, that is too much to hope for; still, by concerted effort—and it is not a very great effort either, I think—we should certainly be able to make a fresh record of membership.

Then, with regard to the papers. Mr. Richardson has already appealed for more papers, not necessarily long papers, taking up very much of one's time, but something on the practical and useful points which come to one's notice in the course of one's every day employment. But some of us can do more than that. It is merely a matter of making up one's mind. In the first place, one has to secure a promise to do so, and once a man has promised a paper it is usually forthcoming. I think most of us must know someone who is likely to contribute a paper. It is only a matter of pegging away at him long enough to secure it. It certainly takes time, and I appreciate the fact that I have got two months within which to prepare an address. I must say, in conclusion, that the excellent way in which Mr. Richardson has discharged his duties has made it very hard for me to emulate him; but I will do my best.

The meeting then terminated.

Contributions and Correspondence.

LAUNDER GRADES FOR SAND PULP.*

The accompanying empirical formulæ, table and curve (see next page) are based on observations (within the limits shown) upon the flow of pulp under large scale working conditions, and refer to ordinary sand residue pulp as used for mine filling, which contains about 4% pyrite, and of which up to 10% of the solids are retained on a 60-mesh (0.010 in. aperture) screen. The many factors, both in a reduction works and in underground sandfilling, affecting the launder grade on which pulp will flow, cause the exact grade required to vary in each case.

In general, the conditions tending to necessitate an increased launder grade or higher water ratio are small volume of pulp; frequent sharp curves in launder; wrong shape and rough internal surface of launder; unevenness of launder grade; and large size of sand particles and high percentage of pyrite.

Conversely, a large volume of pulp flowing in a well designed and laid launder without curves, and containing much slime and few coarse or pyritic particles, requires less grade than is usually required for sand pulp in practice.

In the installation of a launder it is usually desirable to increase the grade by, say, one-fifth round curves, as well as for some distance at the head of the launder in order to overcome the initial inertia of the pulp until it attains its normal velocity.

As a ready approximation for the conversion of degrees to percentage grade, each degree may be taken as equivalent to $1\frac{3}{4}$ per cent grade.

W. A. Caldecott.

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* See this *Journal*, Jan., 1909, p. 242; June, 1909, p. 440; Sept., 1913, p. 121; and Dec., 1913, p. 305.

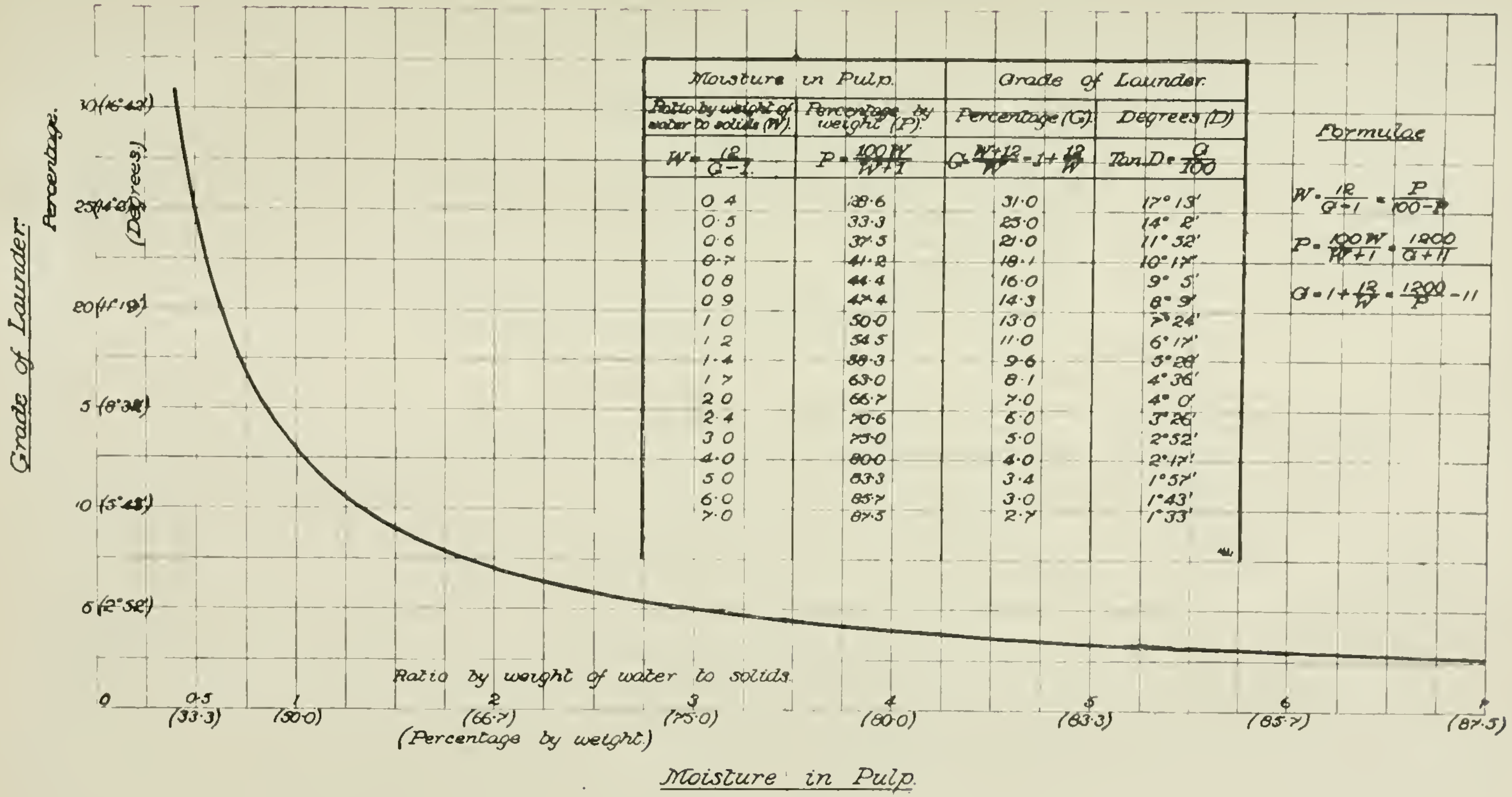
THE THIRD ANNUAL MINING EXHIBITION.

The Third Annual Mining Exhibition, organised by the Chemical, Metallurgical and Mining Society of South Africa, was held at the Volunteer Drill Hall, Twist Street, Johannesburg, during the period 19th to 30th May inclusive.

At the opening ceremony on Tuesday the 19th May, there was a large and representative gathering of leading citizens and mining men. Mr. Alex. Richardson, President of the Society, was in the Chair, and was supported on the platform by the members of the Exhibition Committee—Mr. W. Cullen (Chairman), Prof. G. H. Stanley, Messrs. A. F. Crosse, J. E. Thomas, Mr. E. A. Wallers (President of the Transvaal Chamber of Mines), Jas. Littlejohn, W. R. Dowling, Prof. J. A. Wilkinson (Chairman, South African Red Cross Society), P. Cazalet and F. J. Trump (representing the Judges of the "Safety First" Exhibits), Dr. G. S. Corstorphine (Principal of the S. A. School of Mines and Technology). Amongst other prominent mining men present were Messrs. R. W. Schumacher, S. Evans, J. Munro, H. Stuart Martin, W. T. Graham, R. Shanks (Inspector of White Labour), A. G. Gill, R. Hosken, W. Moses (Inspector of Mines), S. H. Pearce, K. L. Graham, J. Gray, W. Elsdon Dew (President, S. A. Institute of Electrical Engineers), M. Torrente, J. R. Williams and E. Farrar.

LAUNDER GRADES FOR SAND PULP

(W. A. Caldecott.)



Moisture in Pulp.		Grade of Launder.	
Ratio by weight of water to solids (W).	Percentage by weight (P).	Percentage (G).	Degrees (D)
$W = \frac{12}{G-1}$	$P = \frac{100W}{W+1}$	$G = \frac{W+12}{W} = 1 + \frac{12}{W}$	$\tan D = \frac{G}{100}$
0.4	28.6	31.0	17° 13'
0.5	33.3	25.0	14° 2'
0.6	37.5	21.0	11° 32'
0.7	41.2	18.1	10° 17'
0.8	44.4	16.0	9° 5'
0.9	47.4	14.3	8° 9'
1.0	50.0	13.0	7° 24'
1.2	54.5	11.0	6° 17'
1.4	58.3	9.6	5° 28'
1.7	63.0	8.1	4° 36'
2.0	66.7	7.0	4° 0'
2.4	70.6	6.0	3° 26'
3.0	75.0	5.0	2° 52'
4.0	80.0	4.0	2° 17'
5.0	83.3	3.4	1° 57'
6.0	85.7	3.0	1° 43'
7.0	87.5	2.7	1° 33'

Formulae

$W = \frac{12}{G-1} = \frac{P}{100-P}$

$P = \frac{100W}{W+1} = \frac{1200}{G+11}$

$G = 1 + \frac{12}{W} = \frac{1200}{P} - 11$

The Consolidated Gold Fields of South Africa, Ltd.,
Metallurgical Department.
17th June 1914.

Mr. A. Richardson (*President of the Society*), said on behalf of the Chemical, Metallurgical and Mining Society, I beg to extend to you a hearty welcome: and before asking Mr. Wallers to open the Exhibition I should like to make a few explanatory remarks concerning it. The Society has always taken an active interest in matters educational and humanitarian, and these Exhibitions are the outcome of that interest. The educational value of such a Mining Exhibition as this, lies in the opportunity it affords members of the public to come and inspect mining appliances and models, to attend lectures and demonstrations given by experts, who have kindly volunteered their services, and, generally, to bring themselves into touch with mining and its many problems. The humanitarian aspect is one to which we have always been keenly attracted. Many years ago the Society did some notable pioneer work in reference to miners' phthisis and to safety measures in mining, and was successful in arousing a very general interest in matters relating to the amelioration of underground conditions; and that it still keeps its ideals alive is shown by its recently greatly assisting in the formation of the South African Red Cross Society. The dissemination of knowledge and the protection of human life, then, are the two objects chiefly aimed at; and in addition to these it may be mentioned that every encouragement is given to display of the raw material products of the country; and every facility is afforded the small man (small that is, financially speaking), to bring his ideas and inventions to the general notice; and it is very encouraging to be able to record that the objects of the Society have met with the approval of the public, which attended the first Exhibition to the number of 16,000, the second to the number of nearly 20,000, and we hope they will attend this third exhibition in still greater numbers. Of course all this cannot be accomplished without considerable outlay, and here we have to thank the various commercial firms who have rented space, an expenditure which we trust they will find more than justified, and the Chamber of Mines which has given us a monetary grant. With assistance in these ways we are able to meet the expense which must necessarily be incurred. Regarding the general organisation of the Exhibition we have to acknowledge helpful assistance from several quarters. Through the kind offices of Major Jones, the District Staff Officer, we have been enabled to obtain the use of this Drill Hall, and the Exhibition Committee, consisting of Messrs. Crosse, Moir, Stanley and Thomas, with Mr. W. Cullen as its energetic chairman, and Mr. Fred Rowland as its indefatigable secretary, has as I think you will admit, done its work most creditably. We are also much indebted to the Rand Mutual Assurance Society for stimulating interest in safety measures by generously offering prizes for safety devices, and to Mr. R. N. Kotzé, the Government Mining Engineer, and other gentlemen for giving their services as judges in this connection, and to the judges who will make the awards with regard to the Red Cross Society's ambulance exhibits; and finally, we wish to record our appreciation of the sympathetic spirit shown by Mr. E. A. Wallers, the President of the Chamber of Mines, in coming here this afternoon to identify himself with the work that is being done, and to help it on its way; and I now have great pleasure in asking him to open this, the Third Mining Exhibition.

Mr. E. A. Wallers said that it was a great pleasure to him to be invited to open such an Exhibition. To be quite frank, the pleasure was increased by the knowledge that he was not required to do much speech-making—speech-making was one of those occupations classed as dangerous. He took it they were all striving after that efficiency which is absolutely necessary, and the work of that Society, particularly through that Exhibition, was a very valuable contribution to those efforts. He knew that the Chemical, Metallurgical and Mining Society had activities in many directions, but as their Chairman had explained, their two great objects were the encouragement of the small man—the stimulating of his inventive capacity, and the development of the "Safety First" campaign. As regards the first object he imagined that there were few places which offered better facilities than this. The small man was in constant touch with his work on the mine, or was studying at the technical university in the evening. Those who administered this industry, he knew well, had always been anxious to find out the man with those devices and inventions of various descriptions which largely helped to solve the problems with which they were faced. After all, the managers and the heads of departments on a mine were busy people, and it might be that sometimes an invention or device had not received the consideration due to it. This Society stepped into the breach, however, in a most effective manner, as could be seen by the number of exhibits there that day. As to the Safety First Committee it was a movement which appealed to him very much. Employers had always had the greatest concern for the safety and welfare of their employes. In eighteen months the movement had been initiated and put on its present basis, and had gained an enormous impetus. The Committee had collected exhibits from America and other parts of the world. It was possibly to be regretted that the movement was not initiated in its present form earlier. However, as it was to-day, both those who administered the industry and the mine workers themselves had taken it up with great enthusiasm. It was obvious that no movement of that nature could be successful unless it had the sympathy of the men employed on the mines. The Safety First Committee was a very strong one, and it had the advantage of the practical advice and sympathy of the Government Mining Engineer and his officials. Mr. W. Cullen and Mr. Archbold (one of the Mine Inspectors) would deliver lectures dealing with the subject during the period the Exhibition would be open. The men were being interested in the questions that were raised for discussion on the points put before them by Mr. Archbold at other lectures on the mines. The enterprise of the Society in promoting those exhibits had met with the success that it merited. Larger numbers visited the Exhibition of 1913 than in previous years, and he was sure that the present one would be a record. He congratulated the members on their decision to hold the exhibitions annually, and he was certain that the exhibitions would continue to extend, and would continue to the benefit of the mining industry, and therefore of the community generally.

Mr. Wm. Cullen proposed a vote of thanks to Mr. Wallers, who, he remarked, had attended the Exhibition at great personal inconvenience. The presence of Mr. Wallers showed the corporate sympathy of the Chamber of Mines with the objects of the Exhibition. Mr. Cullen then read the report of the judges regarding the "Safety First" exhibits sent in to the Mining Exhibition for competition, showing that as a result of their examination they have come to the following unanimous decision :—

Class A. – The following prizes have been awarded : First prize of £25 and silver medal, No. 12, Mr. E. Nowag, Dynamite Factory, Modderfontein, for a safety clip for the prevention of misfires, etc. Second prize of £15 and bronze medal, No. 10, Mr. A. W. Lindberg, Ferreira Deep, Ltd., Johannesburg, for an overwinding device for electric hoists. Third prize of £10, No. 4, Mr. Otto E. Dick, Geldenhuis Deep, Ltd., Cleveland, for a safety bar with pegs for locking hanging stamps in position. The following are highly commended : No. 13, Mr. J. H. Ruthven, P.O. Box 6253, Johannesburg, for an improved automatic safety catch for mine cages ; No. 9c, Mr. A. Job, Van Ryn Deep, Ltd., Benoni, for a safety hook for attaching winding rope to truck : No. 17, Mr. A. W. R. Schieder, 343, Jules Street, Malvern, for a safety explosives box.

Class B. – The judges make no award under this section.

Class C. The prize of £10 for the best exhibit of an enclosed lamp is awarded to Mr. G. G. Carter, P.O. Box 94, Johannesburg. The lamp exhibited by Messrs. Ryan Bros., was highly commended. The judges regret that no suitable candle lamp has been exhibited.

Mr. R. N. Kotzé (Government Mining Engineer) was the Chairman of the Judging Committee, which consisted of Messrs. P. Cazalet, F. J. Trump, W. Calder and J. E. Thomas.

The ceremony concluded with a vote of thanks to the Chairman, and the large number of visitors proceeded to inspect the exhibits.

For this Exhibition the Safety First Committee of the Rand Mutual Assurance Co., Ltd., offered a number of money prizes and medals for the best exhibits sent in to further "Safety First" on the mines, while the South African Red Cross Society offered further prizes for Ambulance Appliances. It was not found possible to make the awards for the latter exhibits until the closing day of the Exhibition when the judges, Dr. A. H. Watt (of the Simmer and Jack Hospital), Mr. Edgar Pam (Manager of the Geldenhuis Deep, Ltd.), Mr. T. A. Forrest (Ambulance Superintendent of the East Rand Proprietary Mines) and Mr. A. D. Maxwell (Transvaal Miners' Association), awarded the prize of £5 and bronze medal for the best stretcher for underground use to No 3, Mr. H. J. Heasley, Luipaardsvlei Estate and Gold Mining Co., Ltd., P.O. Box 53, Krugersdorp. The Stretcher exhibited by Mr. T. Little (No. 6), was highly commended. No awards were made in the other two classes for which prizes were offered.

The following is a list of the Exhibits.

Note.—The descriptions given were for the most part supplied by the exhibitors.

INDIVIDUAL EXHIBITS.

1. Mr. J. Cawthorne, East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand: Improved Battery Ore Feeder. This is devised to ensure greater regularity of delivery of ore to battery mortar, and increased crushing efficiency

2. Messrs. Craggs & Campbell, Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs :—Patent Overhead Mechanical Haulage Jockey, suitable for wire ropes from ½ in. to 1 in. diameter.

3. Mr. W. Daly, Village Deep, Ltd., 2, West Avenue, Village Deep, Johannesburg :—Model of Tappet and Stem Guide for Stamp Mills. The tappet has a special gib ensuring equal bearing surfaces on front and back of key, thus preventing distortion. The gib, being compressed around stem, secures it to tappet with a non slipping grip. The stem guides provide an easy method of renewin stems, and keep the battery in perfect alignment.

4. Mr S. Evans, 79, Nugget Street, Johannesburg : Three Fly Traps as used at the Crow Mines.

5. Mr. A. R. Foyer, P. O. Ottoshoop : –Portable High-pressure Shower Bath This apparatus is devised for use in mining camps and other places where no water pressure exists. It consists of a water container, holding about two gallons, fitted with pressure pump. A rose is attached to a hand container (fitted with a controlling valve to prevent waste of water) connected with the water container by a light hose

6. Mr. R. Gay, Roodepoort United Main Reef G. M. Co., P. O. Box 115, Roodepoort : Stope Measuring Instrument. A device for economising time and labour and faciliting the work of stope measurement.

7. Mr. J. M. Gourley, Brakpan Mines, Ltd., P. O. Box 3, Brakpan : Method of utilising old wire rope to strengthen mine timber.

8. Mr. F. Hill, P. O. Box 3747, Johannesburg : 1. Two Designs of "Universal Lubricating Rollers," Wire Rope Idlers, Five Rollers, Two Frames, Sliding Cradle with Brake ; with Blue Print. *(a)* These are built up in parts, which can be renewed, and having an internal chamber for grease reserve running on a stationary shaft. The whole surface of roller utilised. *(b)* Revolving shafts run off hollow rollers charged with grease in dust proof boxes. 2 Model of Universal Mining Block and Ties : with Blue Print.—Designed to take load, portable, readily applied, and easily constructed, to carry any weight 3. Universal Reversing Chuck for Binding Small Mining Timber or Poles with Wire : with Blue Print. — Designed for assembling and binding straight wattle tree poles for mining purposes.

9. Mr. A. Job, Van Ryn Deep, Ltd., P. O. Box 225, Benoni : –Arrangement to give alarm if supply of water to cylinder jackets and intercooler of air compressor fails. Drawing.

10. Mr. G. E. Kohlmeyer, P. O. Box 3685, Johannesburg : — One Working Model of "The Paracentric Roller Mill," with four Plans.—The mill consists of a cylindrical shell, in which a number of rollers are caused to turn, crushing the ore in the process. It is suitable for erection in existing buildings, which only require alterations to foundations : and its use will ensure a saving of power, belting, lubricants, and general maintenance.

11. Mr. S. Masters, New Heriot G. M. Co., Ltd., P. O. Box 10, Cleveland : Traffic Indicator for Motor-cars, Road Vehicles, etc. This consists of of two sets of lamps, one set at rear of vehicle for observation by following vehicles, and one set in front for signalling to pedestrians and controllers of traffic.

12. Mr. E. McChesney, Knight Central, Ltd., P. O. Box 91, Germiston :—Machine Designed for making Tamping Sticks, Broom Handles, etc. – This machine should be run at 5,000 revolutions per minute to accomplish its work

13 Mr. D. McLachlan, 16, Melle Street, Johannesburg : *(a)* Models of Detachable Bits. *(b)* Patent Spray for Hammer Drills.– This dust-allayer is designed to carry the water to the end of the hole while it is being drilled.

14. Mr. P. T. B. Morrisby, Witwatersrand G. M. Co., P. O. Box 1, Knights : Improved Slat Gate for Sand Collecting Vats. This device obviates use of

canvas roller blinds; it has water-tight joints, and is easy and efficient in adjustment.

15. Mr. R. Penberthy, Robinson Deep G. M. Co., P. O. Box 1392, Johannesburg:—Specimen of Pitchblende, from Trenwith Mines, St. Ives, Cornwall.

16. Mr. F. G. A. Roberts, Knight Central, Ltd., P. O. Box 91, Germiston: Sample Mixing Table, for use in Assay Offices, designed to prevent the escape and consequent inhalation of dust while the supplies are being mixed and quartered for assay purposes.

17. Messrs. G. A. and D. Robertson, May Consolidated G. M. Co., P. O. Box 50, Germiston:—The Dumb-bell Fine Crushing Mill. This is an improved type of roller mill. The roller is of dumb-bell shape, the ends of which are bevelled and fit into grooves. There are two faces. The machine is fed similarly to tube mills, and the pulp therefrom is classified and over-size automatically returned to mill. It is claimed that it is cheaper and costs less for power and maintenance than tube mills.

18. Mr. A. W. R. Schieder, 343, Jules Street, Malvern: Model of Portable Stope Box, to be used in ordinary method of working; also used as a self-tilting box at the end of shakers in flat stopes.

19. Messrs. A. and P. A. Smorenburg, 1, Old Market Buildings, Johannesburg: Plan and Model of a Self-closing and Self-disinfecting Vault Closet, fly-proof and hygienic. For use either underground or on the surface. It is emptied by means of suction into an hermetically closed tank for removal to depositing site. Odourless.

20. Mr. S. H. Steels, Crown Mines, Ltd., "The Firs," Roodepoort: End Liners for Tube Mills.—The liners are designed to give a maximum life with minimum cost. Wear in 11 months only ½ inch in most cases.

21. Messrs. Thomson & McDougall, P. O. Box 34, Maraisburg: Patent Detachable Head for Rock Drills.

22. Mr. C. Toombs, East Rand Proprietary Mines, Ltd., P. O. Box 159, East Rand: Improved Sample Divider. Device is designed for giving an accurate division of mine samples after passing through crushers. Will give an accurate ⅛th, ¼th, or ½th of original sample as required for pulverising. Can be used safely for giving accurate division of samples from reduction works.

23. Mr. H. Tyzzer, New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni: An Improved Device for Ramming Concrete, Earthworks, and the like.

24. Mr. J. A. Woodburn, Messina, Northern Transvaal: *(a)* Specimens of Mineral and Rock from Messina Mine. *(b)* Model of Portion of Messina Copper Mine.

25. Mr. W. A. Lombard, Rietspruit, P. O. Groot-afdeeling, Marico: Two specimens of "Clay Stone." This is represented to be suitable for table-tops, kitchen floors, and for similar purposes. It is easily worked, and can be obtained in slabs of about 8 ft. × 4 ft., and smaller sizes.

"SAFETY FIRST" EXHIBITS.

Prizes were offered for the best Exhibits of a Safety Device, Appliance, or Method of Procedure, the *bona-fide* idea of the Exhibitor, and to be of practical value for the purpose of preventing accidents to workers, or minimising the effects of such accidents, in or on mines, and be such as have not hitherto been in general use.

Class A.—First Prize, £25 and Silver Medal; 2nd Prize, £15 and Bronze Medal; Third Prize, £10.

Class B. (open to Commercial Firms and Agents).—First Prize, Silver Medal; Second Prize, Bronze Medal. (No award made).

1. Mr. Robert Brown, Glencairn Main Reef G. M. Co., P. O. 191, Germiston:—A Safety Device for Use on Skips in Incline or Vertical Shafts (model of skip with device attached.—Safety catches, which, on the rope breaking, are actuated by a spiral spring in a boss attached to the distance piece of the bridle of the skip, to catch on to the timber framing in the shaft.

2. Mr. J. P. Cunniff, 3, Rand Buildings, Avenue Road, Fordsburg:—*(a)* Safety Device for the Prevention of Accidents in Shafts, Lifts, and the like (drawing). — The device is worked by means of clutches connected with rope tackle, which, when hoisting rope breaks, causes the weight of cage to draw clutches in position against face of guides, thus gradually stopping falling cage. *(b)* Safety Device for the Prevention of Accidents in Shafts, Lifts, and the like. When disconnection occurs two cams are pulled down by a spring, which forces safety catches to grasp sides of guides, thus gradually stopping descent of cage.

3. Mr. William Daly, Village Deep, Ltd., P. O. Box 1411, Johannesburg: Model of Cage with Safety-Catch Device. Upon the fracture of rope an instantaneous grip is secured between the runners and the device fitted on top of cage, which is brought to rest without shock by a frictional grip set up between the guides and runners.

4. Mr. Otto E. Dick, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland:—A Safety Bar, with Pegs for Locking Hanging Stamps in Position. This device is intended to prevent hanging stamps from falling or being forced from stamp supports, and injuring men whilst making renewals or repairs; also a safeguard against the stamps which are under repair being dropped. (Third prize).

5. Mr. R. M. Dixon, City and Suburban G. M. Co., Ltd., 6a, Nourse Street, Troyeville:—Air Filter, to be attached to rock drills to purify air after it leaves machine, thus preventing Miners' Phthisis. This device is intended to purify the air leaving the exhaust of the machine, to act as a silencer, and to prevent diffusion of dust.

6. Mr. P. H. Hemming, P. O. Box 32, Maraisburg: Device to draw the attention of Hoisting Engine Drivers to their duties. An indicator dial to be operated by driver to remind him of duty in hand.

7. Mr. J. Higgins, New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston:—Explosives Box (model) for the safe storing of explosives underground, and their safety in transport.

8. Mr. F. C. W. Ingle, Ferreira Deep, Ltd., P. O. Box 1021, Johannesburg:—Procedure for Prevention of Accidents to Trammers, with drawing.

9. Mr. A. Job, Van Ryn Deep, Ltd., P. O. Box 225, Benoni:—*(a)* Safety Arrangement attached to Overwinding Prevention Device for use when lowering or raising persons. Drawing only. *(b)* Arrangement attached to quadrant of control lever of Electric Hoist to prevent driver putting reverse current into motor. Drawing only. *(c)* Safety Hook for attaching Winding Rope to Truck. Drawing only. (Highly commended). *(d)* Safety Brake attached to Man Cage. Drawing only. *(e)* Overwinding Prevention Device for use when winding persons, and to act also above tips. Drawing only. *(f)* Safety Arrangement attached to Overwinding Prevention Device for use when lowering or raising persons. Drawing only.

10. Mr. A. W. Lindberg, Ferreira Deep, Ltd., P. O. Box 1021, Johannesburg:—Overwinding Device for Electric Hoists. (Second prize).

11. Mr. J. Malcolm, Ayliff Street, Newcastle, Natal:—Tamping Rod.

12. Mr. E. Nowag, Dynamite Factory, Modderfontein: A "Safety Clip" for prevention of misfires due to the pulling out of fuse and detonator from the primer. The clip holds detonator and fuse firmly in primer cartridge, preventing misfires which are frequently caused by the pulling away of fuse and detonator from primer cartridge while charging and tamping hole. Obviates unsatisfactory method of tying with string. (First prize).

13. Mr. J. H. Ruthven, P.O. Box 6253, Johannesburg:—Improved Automatic Safety Clutch for Mine Cages.—A working exhibit with half cage and 30ft. vertical rails. Designed to stop cage gradually and to avoid shock. The device can be adapted to incline shafts. (Highly commended).

14. Messrs. Ryan Bros., New Primrose G.M. Co., Ltd., P.O. Box 193, Germiston:—"Safety First" Dynamite Box.—This box offers no attraction as a resting place, and will not hold a lamp.

15. Messrs. H. E. Rygor & Wiesholtz, P.O. Box 290, Benoni:—Safety Device for Skips running in inclined shafts.

16. Mr. F. T. Sanders, Glencairn G. M. Co., Ltd., P. O. Box 191, Germiston:—Safety Clamp to prevent stamps from slipping off fingers when men are working in mortar box; for battery use only.

17. Mr. A. W. R. Schieder, 343, Jules Street, Malvern:—A Safety Explosive Box for Storing Explosives Underground.—This has two lids, the inside one opening right to back of box, necessitating removal of lights and material on top of box before it can be opened. (Highly commended).

18. Mr. R. J. Smythe, Ferreira Deep, Ltd., P.O. Box 1021 Johannesburg:—One Water Tap for high or low pressure, with safety device to prevent persons placing mouth against nozzle (common occurence underground), thus preventing tuberculosis infection.

19. Mr. A. W. Stockett, P.O. Box 5393, Johannesburg:—Safety Door for Multiple Deck Man Cage.—By the use of this device all decks of cage are under control of man in charge of lower deck, and until doors of upper decks are properly closed it is impossible to close bottom deck door. The closing of bottom door automatically locks upper doors, which cannot be opened until bottom door has been opened by man in charge.

20. Mr. J. H. Veasey, New Kleinfontein Co., Ltd., P O. Box 2, Benoni:—Device (and method of using same) for Securing Greater Safety for Miners when blasting in Isolated Vertical or Steeply Inclined Winzes.—This device enables miners to disconnect all fuses from their primers, should the necessity arise, and if a "running fuse" occurs the first explosion will disconnect all other fuses from their primers.

21. Mr. R. M. P. Williams, Ginsberg G.M. Co., Ltd., P.O. Box 16, Knights:—Safety Automatically Emptying Car.—The contents of car are emptied by bottom discharge, and do not require hand-tipping, the discharge automatically opening and closing and locking and unlocking. It ensures saving in time and labour, safety in operation, and in cost of manufacture.

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22. Mr. E. H. Eeles, 1 Fleming Street, Capetown:—"The Scout" Device.—An electrical instrument for indicating and signalling movements of caving or shifting ground. Pressure of ground in stopes, drives, etc., causes compression of a spring, which by an electrical contact rings a bell and shows a red light, thus warning those in vicinity of danger.

Class C: Prize of £10 for the best Exhibit of an Enclosed Lamp, suitable for use in Metalliferous Mines by Miners whilst handling explosives:—

1. A. E. G. Electrical Co., of S.A. Ltd., P.O. Box 1129, Johannesburg: *(a)* Miner's Safety Lamp. *(b)* Miner's Safety Lamp.

2. Messrs. Blane & Co., P.O. Box 435, Johannesburg:—The Float Electric Hand Lamp. "Miner Type," "Police Type," and "Household Type."

3. Mr. G. G. Carter, P.O. Box 94, Johannesburg:—An attachment for enclosing the flame of an ordinary Acetylene Hand Lamp, for use in mines by miners whilst handling explosives. (First prize).

4. Messrs. A. H. Johnson & Co., P.O. Box 5391, Johannesburg:—Edison Electric Safety Lamp.

5. Mr. H. B. Johnson, 144, Banket Street, Hillbrow:—Safety Candle-Saving Lamp.—A simple and cheap device which effects a saving of 75 per cent. in candles, and by the light of which explosives can be handled with perfect safety.

6. Mr. W. Muir, Crown Mines, Ltd., P.O. Box 101, Crown Mines: A Safety Lamp for use when transporting explosives underground.

7. Messrs. Ryan Bros., New Primrose G.M. Co., Ltd., P.O. Box 193, Germiston:—*(a)* Enclosed Lamp. A safety protector to be attached to any lamp or open flame. (Highly commended). *(b)* Safety Protector for Acetylene Lamps, Candles, or Naked Light, for use whilst handling explosives. A device for attachment to any lamp or open flame, so constructed that no flame can reach explosives.

AMBULANCE APPLIANCES.

Prizes offered by the S.A. Red Cross Society.—Entries for Best Underground Stretcher. Prize of £5 and Bronze Medal.

1. Dr. J. L. Aymard, Castle Mansions, Johannesburg: A Stretcher for general use underground, providing for suspension and fixation of patient, and transit on runners or rails.

2. Mr. J. M. Gourley, Brakpan Mines, Ltd., P. O. 3, Brakpan:—Brakpan Stretcher.

3. Mr. H. J. Heasley, Luipaardsvlei Estate and G. M. Co., P. O. Box 53, Krugersdorp:—Underground Stretcher. A plank stretcher specially designed for the speedy and safe removal of a patient from underground. (First prize).

4. Knight Central, Ltd., P. O. Box 91, Germiston:—Stretcher for Underground Use.

5. Lennon, Ltd., P. O. Box 928, Johannesburg:—Kirker Ambulance Sleigh.

6. Mr. T. Little, Crown Mines, Ltd., P. O. Box 106, Crown Mines:—Improved Red Cross Stretcher, as used by winning team in Crown Mines Underground Competition. Weight 32½ lbs., suitable for transport in vertical or steep stopes, with fittings to meet all underground requirements, and, being narrower than ordinary stretcher, will easily pass through narrow openings. (Highly commended).

7. Wm. Hosken & Co., P. O. Box 667, Johannesburg: Patent Collapsible Knapsack "Veleda" Stretcher. Made of stamped non-corrosive steel; weight 23 lbs., will fold into a knapsack 12 in. × 12 in. × 6 in. Suitable for underground use of any kind.

The following was exhibited not for competition:

Dr. J. L. Aymard, Castle Mansions, Johannesburg: Surface Stretcher.—A modified Army stretcher for general surface use, with movable legs and detachable portions.

S.A. Red Cross Society.—Entries for Ambulance Appliances other than Stretchers. Prize of £5 and Bronze Medal. (No award was made in this section).

1. Mr. J. M. Gourley, Brakpan Mines, Ltd., P. O. Box 3, Brakpan:—Miner's Pocket Ambulance Case.

2. Mr. T. Little, Crown Mines, Ltd., P. O. Box 106, Crown Mines:—Underground Ambulance Trolley.

This is used on the Crown Mines, Ltd., for speedy and comfortable transport of injured underground, and is specially suitable for low and narrow drives and winzes. Has a wheel-base of 23 inches, and will run on 18 in. gauge rails. Fitted with spiral springs and rubber studs to prevent jarring on bad tracks. Weight, 166 lbs. Fittings allow for quick attachment of stretcher. Can be used by one man.

3. Wm. Hosken & Co., P. O. Box 667, Johannesburg. (*a*) First Aid Haversack, in strong canvas, leather bound, fully equipped with all essentials for use in mines. (*b*) "Pretoria" Reviving Apparatus for administering Oxygen to "gassed" miners.

The following was kindly loaned by the Vereeniging Estates, Ltd. The "Brat" Reviving Apparatus.

S.A. Red Cross Society.—Entries for Best First Aid Box for use on Mines. Prize of Bronze Medal. (No award was made in this section).

1. Mr. J. M. Gourley, Brakpan Mines, Ltd., P. O. Box 3, Brakpan:—First-Aid Box.

2. Lennon, Ltd., P. O. Box 928. Johannesburg:—First-Aid Case.

3. Wm. Hosken & Co., P. O. Box 697, Johannesburg:—"Simonis" Ideal Ambulance Box, "A."

COMMERCIAL EXHIBITS.

Geo. Angus & Co., Ltd., corner Main and Loveday Streets; P. O. Box 348, Telephone 1031, Tel. Ad. "Angustate," Johannesburg. A branch of the well-known Newcastle-on-Tyne rubber firm, which supplies Leather Belting and Rubber Hose, and which has introduced several novelties adopted by the Rand and other gold mines. Enamelled ware is a speciality of this firm, and the latest thing in printed enamelled plates which has been put upon the market is the Mining Regulations. It is required by law that the regulations, or abstracts of them, must be posted about the mines. Up to the present this has been done by notices printed on linen, which speedily become torn and illegible. Enamelled plates can now be obtained with every notice which may be required, at a price cheaper than linen sheets, besides being practically ever-lasting and always readable. The firm's enamelled lettered buckets are too well-known to need further description, while their Portable Amalgam Safe, Champion Block and Tackle, Balata, Leather and Link Belts, Fire Hose and appliances, etc., etc., may be inspected on this stand.

Bartle & Co., Ltd., Loveday House, Loveday Street, Johannesburg; P. O. Box 2466, Telephones 3553 and 3554, Store Telephone 1457, Telegraphic Ad. "Faggot," Johannesburg. A large and varied stand, comprising:—Gillespie's Patent Truck Bearing, Thomas' Patent Truck Bearing, Simon Patent Jockey and Feed Nut, Drilling Machine for assay office, Daly's Patent Safety Skip Catch, Phillios' Patent Safety Warning Device to prevent over-winding, Lindberg Electrical Safety Device, Thomas' Patent Skip Locking Device, Stockett & Wilson's Patent Device for Locking Man-hauling Cage, "Boss" Hose Connections, Patterson's Patent Sprays, Cleveland Hammer Drills and Valve Grinders, Bickford's Radial Drills, Cincinnati Milling Machines, Osters and Woodhead's Pipe and Bolt Screwing Machines, Smith's Stem Lifter, Newall's Patent Gear for Windmills, Night Watchman's Time Detectors, Hy. Pels & Co.'s Up-to-date Punching, Shearing and Cropping Machines, hand and power; Dickinson's Shaping Machines; Corner Pneumatic Ratchet Drill, Simson's Rope Pulley's with renewable Centres, Adams' Patent Valves for Slimes, Wells' Patent Battery Lock Nut, Telescopic Bars for Drills, Lathe Emery Grinders, Gravity Vices, Independent and Universal Chucks, Handy Vulcan Pipe Vices, Pioneer Combined Pipe and Bench Vices, Jacks, Safety Lamps, Revolution Counters, Recording Gauges and Thermometers, Anemometers, Special Mulcott Camel Hair Belting, Link Leather and Laminated Belting, Martin's Patent Catch and Safety Hooks for Skips, Bartle's Air Saving Spuds, Russell's Patent Device for connecting pulley centre to cam shaft. Every kind of labour-saving tool, Pooley's Weighbridges and Scales; in fact everything that is required for the equipment and upkeep of a mine.

H. F. Benger, 70, Loveday Street, P. O. Box 772, Telephone 3295, Tel. Ad. "Efandef," Johannesburg. A useful exhibit of Spray Pumps for whitewashing, disinfecting and for allaying dust.

British South African Explosives Co., Ltd., Dynamite Factory, Modderfontein, Tel. Add.: "Dynamite," 'Phone 683, Johannesburg:—The exhibit comprised samples of the various Acids used for mining purposes, including Hydrochloric, Sulphuric, Bisulphate and Nitric Acids. Dummy samples of Explosive, various types of Exploders, Cables, Reels and Instruments used in Electric Blasting. Instantaneous Electric Detonators and Electric Delay Action Fuses.

Clarington Brook Co., "The Shovel House," 93, Cullinan Building, Main Street; P.O. Box 5409, Telephone 2775, Tel. Add.: "Ceebeecee," Johannesburg:—This firm carries the most comprehensive stock of shovels in South Africa, and are showing a large assortment of round and square mouth, also loco type and coal pan shovels of the famous "Bulldog" and "Springbok" brands, manufactured by Messrs. Wm. Park & Co., Wigan. Interesting features include their patent "Vee-Dee" handle, which has demonstrated its superiority over all other types for durability and efficiency: also a "Randic" Shovel, which although continuously used in a stope for many months still retains its shape and a perfect handle. Other exhibits include a splendid selection of "Bulldog" Mining Hammers and Picks of all descriptions. A speciality showing the advance of modern tools will be found in a "Footprint" Portable Tube Vice, which, although only weighing 14lbs. is suitable for tubes from ⅛in. to 3in. diameter.

Consolidated Rand Brick, Pottery and Lime Co., Ltd., 39-46, Cullinan Buildings, Main Street; P. O. Box 609, Telephone 1232, Tel. Ad. "Industry," Johannesburg. This well-known Company showed the following goods, all manufactured at its works at Olifantsfontein:—Machine-made Building Bricks (three varieties); Machine-made Firebricks; Hand-made Firebricks (two varieties); Assorted Arch and Special Firebricks; Fireclay Slabs (assorted); Baffle Bricks (various); Fireblocks for Taverner Furnace; Flue Covers; Muffle and Furnace Doors; Crucibles (assorted sizes); Liners, clay (assorted sizes); Muffles (assorted); Fireclay; Salt Glazed and White Glazed Bricks; Stoneware Pipes and Fittings; Garden Tiles.

Geo. Cradock & Co., Ltd., 603-606, Sixth Floor, Consolidated Buildings, Fox Street; P. O. Box 316, Telephone 539, Tel. Ad. "Ropes," Johannesburg, exhibited:- 1. Wire Rope show case, containing wire ropes manufactured by Geo. Cradock & Co., Ltd.: 2. Show case containing samples of Packing and Belting, etc., manufactured by Messrs Small & Parkes, Ltd., of Manchester: 3. Model of Humble's Patent Safety Detaching Hook for the prevention of accidents in case of over-winding: 4. Samples of Bradford Inverted Plug Cocks; 5. Sample of Cotton Driving Rope: 6. Sample of Kinghorn Metallic Valves: 7. Sample Miner's Safety Lamps: 8. Samples Cradock's Patent Haulage Clips: 9. Sample of Cradock's Patent Pipe Lining for Acid Water, Sandfilling, etc.

The Electric Utility Co. of South Africa, Pollock Buildings, Eloff Street, P. O. Box 5673, Telephone 2168, Tel. Ad. "Galvanic," Johannesburg. Manufacturers of "Universal" Dry Batteries for all purposes, and specialists in Portable Electric Lights. Exhibit comprises complete specimens and the component parts of "Universal" Dry Batteries, together with Electric Torches and Hand and Pocket Lamps, in which the batteries are the source of light. The Company also show a range of Electric Motor-car and Motor-cycle Lamps and Accessories, and a few examples of Reading Lamps and Fancy Shades.

Engineering Supplies, Ltd., 33-36, Bettelheim Buildings, Simmonds Street; P. O. Box 4068, Telephones: Office 309 and 4844, Store 294: Tel. Ad. "Brazilians," Johannesburg. The exhibits on this stand were: "Pulleystone." This is a patent paste, which is applied to the face of a pulley. The adhesion of a pulley so coated is 38 per cent., as against 18 per cent. of ordinary metal pulley. Amongst the advantages obtained by the use of "Pulleystone" are: Increased efficiency of drives; increased life of belts; obviates too tight belts, thus preventing belts and bearings running hot: obviates use of belt dressings, etc. Webb's Patent Jockeys. Webb's Patent Jockey is fitted with reversible and renewable grip blocks, and is forged solid from the best quality mild steel in specially made dies. The two vertical prongs are square, and each prong is fitted with a round renewable grip block, which is held in position by a split pin. When one of the four wearing faces of a grip block becomes worn it is only necessary to take out the split pin and turn the grip block 90 degrees to secure a new wearing face. Two types of this jockey are made, "hinged" and "fixed." The hinged type is specially designed for use where low head room has to be contended with. Webb's Patent Tappet.—The chief advantage of which is its holding properties. Patent "Ubel" Instantaneous Hose Couplings for Rock Drill Hose. Does away with the use of a spanner upon a union nut. The "Ubel" Air Saver.—This small valve is made to screw into the pipe line, and readily comes into action immediately the throttle of drill is opened.

Globe Foundry, corner Fox and Delvers Streets; Tel. No. 1780, Johannesburg:—This foundry showed the following:—5in. Suction Valve Box; 5in. Delivery Valve Box; 5in. Pole Case; Anton Patent Truck Bearing; Candy's Stem Guide Bearing; Cable Junction Box; Cast Iron Pinion; Wood Sill, with two of Smith's Patent Cable Clamps attached; Cast Iron Standard; Cast Iron Pipe, with nozzle attached.

Jacob Hansen, Kiel, Germany:—Johannesburg Agents, Bull & Oehmen, Ltd., P.O. Box 4040, Johannesburg, and Harry Levy, Simmonds Street, P.O. Box 2617, 'Phone 3679, Johannesburg: A handsome show case of tools, including spanners, wrenches, planes, augers, chisels, drills, hammers, pliers, knives, &c.

Henley's (S.A.) Telegraph Works Co., Ltd., 19-21, Standard Bank Chambers, Commissioner Street; P.O. Box 5015, Telephone 695, Tel. Add.: "Megohm," Johannesburg:—Two show cases were exhibited by this firm which contain an interesting variety of sections of Electric Light and Power Telegraph and Telephone Cables. The sections show how these cables are made up, from the copper conductor to the outer serving. One of the cases shows the sections of cables which were supplied to the Robinson Mine in connection with the Electrification Scheme a few years ago. Visitors and Mine Electricians will find this little exhibition of very great interest. Messrs. Henley's also advertised on this stand their well-known "Why Not" Golf Balls, which are, like all their manufactures, of the very highest class.

Richard Heymann & Co., 73-75, Exploration Buildings, Commissioner Street; P. O. Box 2425, Telephone 351, Tel. Ad. "Extra," Johannesburg. An exhibit was made on this stand of "Brunsviga" Calculating Machine, for addition, subtraction, multiplication, division, etc., the latest machine being Brunsviga "H." By this machine most complicated calculations, such as dividing the result of two multiplications by the sum of two other multiplications can be effected in two operations by applying a shortened subtraction device specially installed. Arithmotpye.—Calculations of any description are possible with the aid of this machine, including the process of continued multiplication. The speciality of this machine consists in the apparatus attached to this type for recording calculations on paper. Miniature "MB."—A machine weighing 8 lbs., fitted into a leather bag for travelling purposes, its capacity being the same as that of the standard type. Other machines shown are the Standard "A," Miniature "A," the "Trinks Triplex" (by which two calculations can be done at the same time), and the Brunsviga "N" (for continued multiplication), all well-known on the Rand for their enormous usefulness and reliability. Special attention is drawn to the practically noiseless Miniature types. They also supply "3 Star" Crucibles, Mabor Cupels, and Mabor Material, which are unbeaten in quality, uniform, and absolutely reliable. White Lime, slaked and unslaked (high grade).

Holmes & Son, 141, Main Street, P. O. Box 4114, Johannesburg. This firm showed Gould's Patent Belt Fasteners, which are specially adapted for Electric Driving and all High Speed Machinery Belts.

W. E. Hortor & Co., Ltd., City House, Harrison Street, P. O. Box 6151, Telephone 1126, Tel. Ad. "Emboss," Johannesburg. This well-known firm of Manufacturing Stationers, Printers and Engravers showed the following:—"The Comptometer," the only Calculating Machine that adds, multiples, divides and substracts and works both in decimals and £ s. d. It is noiseless in action and easy to operate. The "Underwood" Typewriter, the machine that won the Elliott Cresson Gold Medal for ingenuity, skill and perfection of workmanship. The Globe Wernicke System of Filing Cabinets and Card Index Drawers, in steel and oak—horizontal and vertical. Globe Wernicke Elastic Bookcases, for house or office, on the Unit system with dustproof doors. Globe Cabinet Safes, fire resisting, on roller bearings; shelves can be automatically adjusted to suit requirements; stocked in five sizes. The Protectograph Machine, for use on cheques—absolute safeguard against forgery. Balaban Copy Holder, for use with Typewriter. Gamble's Telephone Recorder, for recording messages and giving a copy at one operation. Loose Leaf Ledgers and Specimens of Artistic Printing.

WM. HOSKEN & CO., Corner of Fox and Rissik Streets; P. O. Box 667, Telephones: General 4113, Stores 4115, Sales 4118, Tel. Ad. "Hosken," Johannesburg. At this stand could be seen Climax Hammer and Water Drills, with spray attachments for allaying dust; the Proto Breathing Apparatus; Salvator Breathing Apparatus; Pretoria Oxygen Reviving Apparatus; Simonis Smoke Helmet; Simonis Ambulance Cycle; Knapsack Valeda Stretchers; Ideal First-aid Ambulance Box; and the L.N.R. Railway Ambulance Fittings for the transport of wounded in railway wagons or ordinary trollies.

A. H. JOHNSON & CO., LTD., corner Von Brandis and Main Streets, P O. Box 5291, Telephone 1175, Tel. Ad. "Sprinkler," Johannesburg This firm showed specimens of the Edison Storage Battery, which has the following advantageous characteristics: (1) Long life; full rated capacity after four years' service. (2) Light weight; output 18 watt. hours per pound. (3) Ruggedness; no buckling or injury by vibration, over-charging, high rate discharging, high temperature or standing idle. (4) Ease of operation; no sulphation, sediment, jar-breaking or corrosive fumes to contend with. Uses for Edison Storage Batteries: Mining Locomotives, Lighting for Cars, Mining Signals, Commercial Trucks, Telephones, and any purpose which requires direct current

THE LEYLAND & BIRMINGHAM RUBBER CO., LTD., Barsdorf's Buildings, Marshall Square, 'Phones: Office 938, Store 471; P.O. Box 4856, Telegrams "Suction," Johannesburg. J. & R. Niven, Managers, South African Branch. This firm displayed its various specialities, which have, after exhausted trials, proved that the Leyland and Birmingham wares have reached the high-water mark. A special feature is being made of Water and Air Hoses, armoured and unarmoured, and where life and efficiency are required this firm's goods will be found to be of the very best. Grading Machine: Messrs. Thos. Locker & Co., the screening manufacturers of Warrington, England, have on view a perfect grading machine, which may be seen working. The battery screening manufactured by this firm is of the very best workmanship and highest quality, every mesh from 9 to 1 600 being made with an absolutely square aperture, and every wire so woven that slipping is entirely prevented. Hoses: The No. 1 quality for sand-filling and washing-out cannot be equalled. The Leyland Company are the patentees of the Trunk or Tapered Hoses for Slimes tanks, and results to date have proved that for efficiency and life this make cannot be improved upon.

GEORGE B NEILSON, 8b, Fourth Street, Eleventh Avenue, Boksburg North. Mr. Neilson showed two patent holders containing stereoscopic views, illustrating Glimpses of Rand Gold Mining. These views can be made up in sets of 24, 48 and 84 views, and also include a stereoscope as part of the outfit. Each view in the sets illustrates in practice some operation or condition in connection with mining, the larger sets naturally being more complete in detail. With the object of making the sets representative, the views are selected from different mines along the Reef, the larger sets including views from no less than thirty separate mines. The views have been taken over a period of eight years, and furnish a good object lesson, showing the various improvements made to date in ventilating conditions, types of drilling machines in use, and the unskilled labour units, both Chinese and native.

J. & R. NIVEN, Barsdorf's Buildings, Marshall Square; P. O. Box 4022, Telephone 813, Tel. Add. "Nivonia," Johannesburg. This firm exhibited the following:—Belting: The well-known brand of "Lincona" Balata, for power transmission and belt conveyors; "Dawson's" best oak tanned Leather Belting and Cup Leathers. Rope Rollers: The "Woodstone" Patent Self-oiling Shaft Roller, 9½in. long by 6½in. diameter; two patent "Simplex" Haulage Rollers, 6in. diameter by 9in. long and 8in. diameter by 12in. long. Belt Conveyor Idlers: The "Woodstone" Patent Self-oiling Troughing Idler for 24in. belt conveyor; also Return Idler for same; Jeffrey Standard Troughing Idler, for 24in. belt conveyor; Jeffrey Standard Return Idler, fitted with self-aligning bearings for 24in. belt conveyor. Haulage Drum: Ashton's Patent Internally Driven Haulage Drum.

ORENSTEIN-ARTHUR KOPPEL, LTD., Central House, Simmonds Street; P. O. Box 3256, Telephone 2185, Tel. Ad. "Koppelrail," Johannesburg. At this stand could be seen diagrams, photographs, etc., of Pokorny & Wittekind's Reciprocating and Rotary Compressors, Steam Turbines, Vacuum Pumps, Pneumatic Tools, Exhaust Steam-driven Power Plants; Sulzer Bros.' Centrifugal Pumps, Uniflow Steam Engines, Garbe Upright Water-tube Boilers, and Diesel Engines; also specimens of Mannesmann Solid-drawn Steel Piping, No. 1, for water and air mains (from 1½ in. to 10 in.); No. 2 for water and compressed air, with flanged joints: No. 3 having double border joints for high-pressure mains; Nos. 4, 5, 6, 7 and 8 with various joints of Mannesmann solid-drawn boring and lining pipes for deep bores and wells.

PREMIER GATE, FENCE AND WIRE CO., LTD., Town Office and Showroom, corner Main and Von Brandis Streets, P. O. Box 4854, Telephone 1175, Johannesburg. Factory, Denver. An exhibit of a Premier Steel Change Locker for mines, workshops, etc., the framing made of heavy galvanised pressed steel, with galvanised steel wire panels, fitted with triple latches.

RINTOUL & DAVIS, 23-4, 35-6, Cullinan Building, Main Street; P. O. Box 1954, Telephones: Office 422, Store 2868, Tel. Ad. "Gondetta," Johannesburg. On this stand might be inspected:—A 4 h.p. St. Mary's Oil Engine; Gould's Fig. 1454 x 6, 8 or 10in. stroke adjustable Power Working Head; Gould's Fig. 1524 No. 6 Columbia Double Cylinder Force Pump; Gould's Fig. 924 4 x 6 Single Acting Triplex Plunger Pump; Higham's Patent Automatic Pulp Sampler; Patent Dividing Pan for use with Higham's Patent Automatic Pulp Sampler; Gould's Fig 904 3¾in. x 24in. stroke Brass Artesian Cylinder; Model American Steam Pump; Gray Wilson Patent Blow-down Valve; Keg W. R. DuMond's Solventine. Electrical Section: Faraday Bell Model A. 6in. Gong; Faraday Bell Model 367 10 in Gong, single stroke; Faraday Bell Skeleton 8in. Gong, trembling; Mine Hooter, in watertight case, 12 volts; Mine Hooter, in brass case, 6 volts; No. 2868 Locked Bell Pull; No. 2868 Ordinary Bell Pull; No. 2869 Lever Bell Pull; Reel "Salamander" Fireproof Magnet Wire; "Hi-po"

Dry Cells: Type M.P. Westinghouse Arrester: Type C. Westinghouse Arrester; Magneto Exploder, Dupont, capacity 50 shots: Pape Miner's Hand Lamp, Ferranti House Service Meter: Double Grip Tape.

SELCRAIGS (COLONIAL), LTD., Mackay's Buildings, Rissik Street, P. O. Box 1187, Telephone 3720, Tel. Ad. "Graphite," Johannesburg. This firm exhibited one complete set of sample bottles of "Suspensol" Brand Graphite Lubricating Oils and Greases. One sample Heinz "Original and Only Genuine" Steam Trap No. 3. One sample of "Phenix" Oil Separators No. 2, 5 in. diameter. One set of samples of Welsh's Patent Electric Tramways Trolley Wheel and Mechanical Ears. One set samples of "Scandinavia" M.C.M. All-woven Cotton Belting, with various belt fasteners.

THE SKEFKO (SOUTH AFRICA) BALL BEARING CO., LTD., 23, Pritchard Street, P. O. Box 2651, Telephones 5658 and 5659, Telegrams "Skefko," Johannesburg. This exhibit illustrated a very striking and novel ball bearing, and its special features render it of singular interest to all concerned with ball bearing applications. The essential feature of the bearing is its ability of taking up any deflection or out-of-alignment of the shaft or housing in which it is mounted. Amongst the exhibits was noticed a unique model showing the application of the Skefko SKF Ball Bearings to a shaft running out of alignment, also a small countershaft fitted with a bent shaft illustrating the applicability of the bearing to line and countershaft work, where the essential of self-alignment is well known by all. Drawings may be seen of the application of the bearing to all classes of machinery, *i.e.*, Tube Mills, Stamp Batteries, Surface Trucks, Belt Conveyor Pulleys, Jockey Pulleys, etc. The innovation of this bearing will be especially welcome in the mining engineering world.

BICKFORD SMITH & CO., LTD., P. O. Box 5237, Telephone 109, Tel. Ad. "Fuse," Johannesburg. Mr. J. H. Houghton, Agent. This firm showed a handsome show case, containing the following:—Bickford's Patent White Countered Guttapercha No. 27 Fuse and No. 27a Fuse; Bickford's Patent Instantaneous Fuse, for subaqueous work: Bickford's Patent Fuse, "Camborne Brand," for use in wet ground: Bickford's Patent Double Guttapercha Fuse, No. 15, for use in 300 ft. of water: Bickford's Detonating Fuse (speed 5,500 yards per second): Bickford's Delayed Action, combining Electric and Safety Fuse (12); Low Tension Electric Powder Fuse (6); Low Tension Electric Detonator Fuse (6), Red Shot Firing Cable, taped and braided: Black Shot Firing Cable, taped, flat and braided: Shot Firing Cable, V. and R., flat yellow; Bickford's Patent Electric Exploders, 80 shots, 15 shots, 2 shots, Cap Nippers, and Renfort's four Blocks of Lead experimented on with Renforts.

THE SOUTH AFRICAN ALKALI, LTD., c/o Messrs. Windram & Taylor, Norwich Union Buildings, Fox Street; P. O. Box 3547, Telephone 141, Tel. Ad. "Accuracy," Johannesburg. This Company exhibited products refined from the raw material which is obtained from the "Zoutpan," in the Pretoria district. Amongst the products shown are: A Soda Ash containing about 70% Na_2CO_3: Powdered and Granulated Refined Alkali; Soda Crystals—an excellent washing soda: and a Boiler Fluid.

SOUTH AFRICAN DISSOLVED ACETYLENE, LTD., Business Managers: Baerecke & Klendgen, 312, Consolidated Building, P. O. Box 3102, Telephone 4757, Tel. Ad. "Sadal." On this stand could be seen a demonstration of Oxy-Acetylene Welding and Cutting with Dissolved Acetylene, which is Acetylene compressed into steel cylinders filled with a porous substance (high-pressure system). Steel, Cast-iron and Aluminium will be welded. Acetylene can only be compressed in a state of absolute purity, and through the purity of the gas produced by the D.A. system increased strength of weld is assured, and is guaranteed to be within 5 to 7½ per cent. of the original strength of the material welded. Dissolved Acetylene Miners' Lamps will be shown. These Lamps are filled at the Company's works, and their use obviates the necessity of taking carbide underground, a practice which has not only caused many accidents, but contributed to the vitiation of the air of the mine. The purity of the compressed gas ensures a steady, brilliant, white light.

SOUTH AFRICAN GENERAL ELECTRIC CO., LTD., Commercial Exchange Buildings: P. O. Box 1905, Telephone 4321, Tel. Ad. "Electrical," Johannesburg. This firm showed an improved Flow Meter for Steam, Air or Water. The instrument is of the Indicating, Recording and Integrating Type, and is connected to the main pipe by means of a Nozzle Plug which makes it unnecessary to dismantle any parts of the main. Special care has been taken in the design of these meters to adapt them to severe operating conditions. There are no delicate working parts or adjustments to get out of order, and the meters will, therefore, retain their initial calibration indefinitely. They are not delicate laboratory meters, requiring ideal surroundings and expert attention, but are designed for use where an accurate, simple and practical meter is required. As a water-flow meter they will be found very accurate and satisfactory for measuring the feed water supplied to boilers, as there is nothing to deteriorate or get out of order. In measuring the flow of water through large-size pipes they are desirable on account of the ease of installation and the comparatively low cost of the meter. They are admirably adapted for measuring the flow of air to mines for ventilating purposes, operations of drills and other compressed air machinery or for determining the total output of compressors. The "Victoria" Turbo Pump exhibited is remarkable for its solid construction, and the Pipe Sections are worthy of close inspection.

TRANSVAAL GRAPHITE, MINING AND MANUFACTURING CO., LTD., 242, Anderson Street; 'Phone 2560, Johannesburg:—An exhibit of Graphite in various forms, mined and prepared for use in the Transvaal.

TRANSVAAL RUBBER CO., LTD., Castle Block, Eloff Street; P.O. Box 1969, Telephones 2533 and 2534, Tel. Ad. "Waterproof," Johannesburg. An exhibit of "Electric" Rubber Hose for rock drill, steam and delivery purposes. The construction of this hose is entirely different from that of ordinary grade rubber, over which is braided one or more fabrics or jackets of tightly twisted and cabled cotton yarn. Between each fabric and on the outside is a layer of rubber. The hose is then vulcanised in a mould under heavy pressure, which drives the rubber through the cotton from tube to cover, making a homogeneous body. This seamless, circular braiding makes the unwrapping of the plies (as so frequently occurs in other makes of hose) an utter impossibility. "Electric" hose is stronger than any regular made hose of double the number of plies, and will stand enormous pressure, ¾in. x 2 ply, for instance, will withstand, without bursting, 500 lbs. hydraulic pressure. "Electric" hose is made in lengths of 500 feet.

THE UNION STEEL CORPORATION (OF S.A.), LTD., Vereeniging, Transvaal, P.O. Box 3174, Tel. Add.: "Firefly," 'Phone 3718, 34-5, Permanent Buildings, Johannesburg.—Manufacturers of Iron and Steel Bars in very large variety: Steel Tram Rails, Tube-mill Bars and Grizzly Bars in the sections in common use along the Reef. The works, which are situated on the Vaal River at Vereeniging, consist of the most modern and useful plant obtainable in the way of melting and re-heating furnaces, rolling mills, 600 ton press, and all the necessary equipment of a thoroughly up-to-date iron and steel works.

LANCELOT USSHER PATENTS, 33, Exploration Buildings, Commissioner Street; P. O. Box 345, Telephone 1608, Johannesburg. Amongst the exhibits shown on this stand were:—Ussher Clamp, for mending burst water or air pipes. No nuts or bolts are used, but a unique and original form of wedge is provided. Amongst its advantages are that it can be fixed by an unskilled person in less than two minutes, even without tools, and it can be easily fixed in most difficult and close places. Ussher Saddle Clamp.—This clamp enables a branch to be taken off any pipe at any point in a few minutes. Ussher Hose Clamp.—For drill and water hoses, especially for use underground, the advantages of a clamp that can be put on or taken off in a few seconds without the use of any tools is obvious. Ussher Hose Mending Clamp. One of these takes the place of two ordinary hose clamps, and its use enables the proper internal diameter of the hose to be maintained without reduction. Ussher Safety Clamps for Rock Drills.—This clamp can be put on in about five seconds and taken off in less than one second. No tools other than a hammer are needed, and no nuts or bolts are used. Ussher Motor-car Pump Holder. This simple device simply cuts the labour of pumping in half, and avoids profanity and perspiration.

LECTURES AT THE EXHIBITION.

Three lectures were given in the small hall during the period the Exhibition was open, which were all largely attended and greatly appreciated by those present. Short reports are printed for the benefit of those members who were unable to be present, and did not have the opportunity of perusing them in the press.

MINE ACCIDENTS ON THE WITWATERSRAND.

Mr. A. M. Archbold *(Deputy Inspector of Mines, Krugersdorp)* lectured on the subject of "Mine Accidents on the Witwatersrand," on the evening of the 19th May.

Mr. R. N. Kotze *(Government Mining Engineer)* presided, and the large attendance followed the lecture with the keenest interest. Mr. Kotze, in introducing the lecturer, took the opportunity of congratulating the organisers of the Mining Exhibition on their work, and commended the attention of all to the Safety First Committee and their objects.

The lecturer, **Mr. Archbold**, explained that he was delivering a course of lectures to miners, and he proposed that night to deal with the managerial side. He was in the position of the man on the fence who saw more of the game than those who were playing. In 1913 there were 2,345 accidents on the Witwatersrand mines, 808 fatalities, and 2,005 cases of persons being injured. The death rate was 3·66 per thousand. The ignorant native was not alone to blame for this, and while accidents could not altogether be avoided, it was the duty of everyone to reduce to a minimum the possibilities of accidents. An excess of economy meant more accidents, while an excess of safety meant a loss in economy. He attributed the number of accidents on the mines to speeding up, insecurity of tenure of officials and miners, the contracting system, the nomadic working population, carelessness and inexperience, and lack of co-operation between the officials and the men to secure safety. He instanced how extra tube mills were put on to a mine with the result that there had to be extra shifts, the skips were run faster, and in the general rush there was a risk of increased shaft and other accidents. There was no time to cut the necessary safety pillars, and the miner had no time to render his working face safe or keep up the timbering. The directors could not be blamed for endeavouring to do the best for their shareholders, but if they could be got to co-operate with the engineers it would lead to increased safety.

The greatest faults the lecturer found on the Rand were the inadequacy of the number of safety pillars and the filling in of worked out areas.

There was a tendency upon these fields to lock the stable when the horse had bolted, and it was only quite recently that filling had been done to any extent. He deprecated the use of ashes for filling on the ground of possible spontaneous combustion, and went on to advocate over-hand stoping in mines which dipped at an angle of 50 degrees

Dealing with explosives, Mr. Archbold said they should endeavour to protect the workmen against their own carelessness. One suggestion he made was that the gelatine boxes should be given a sharp roof like a dog kennel, so that the men could not sit on them and smoke or stick candles on them. He attributed accidents from partially fired holes and misfired holes largely to the condition of the detonators, which deteriorated through being exposed too long to the damp air of the mines. This had been proved by experiments in America, and Mr. Archbold suggested that to avoid this the detonators should be packed in smaller numbers for issue to the miners. He had suggested as a certain measure of safety that the charge should be reversed, but this had been demurred to because it required a more expensive type of fuse which would not burn the gelatine in passing. He thought, however, that this should not be an obstacle where human life was concerned. In the case of shaft ladder ways where the dip was between 35 and 40 degrees, the lecturer thought that partial bratticing would be useful in preventing accidents from falling stones.

Strong character, determination and tact were the leading characteristics required by mining officials of all grades. There was very little fault to find with the managers, but he was not prepared to say the same of the overseers and shift bosses. Discipline among the men was lax sometimes except when the overseers had notice that the Inspector of Mines was coming along. The shift bosses were very important officials in preventing accidents, and if all defects in mines were faithfully reported by the shift bosses accidents due to negligence would be very considerably diminished. The shift bosses should be a little better educated than the ordinary workmen, and they should be trained up on the mines and interest taken in them by those who might one day require their services in an administrative capacity. When the staffs of mines were constant, the low accident rate stood out in sharp contrast. On the West Rand Central—a small mine certainly—there had not been a fatal accident for nine years because the staff stuck to their jobs. The manner in which managers changed mines was remarkable.

The contract system, proceeded Mr. Archbold, was a sweating system, different from the ordinary understanding of the word only in the sense that the men were encouraged to sweat for large cheques which their admittedly expensive habits required. Work was scamped, and the best miners were placed in the easiest stopes, while the indifferent miners were put in the difficult places. The system killed ambition, because the good miner would take a more responsible position for less money. Were the contract system to be abolished, he would suggest a rising scale of pay, and no man should be engaged or re-engaged except on the lowest grade of pay.

In regard to carelessness, inexperience and negligence, he pointed to the prosecutions in the Courts and the disciplinary steps taken by managers. The Safety First Committee had been started, and it was doing excellent work. Shift bosses and overseers did not carry out the regulations as strictly as they should, and in his opinion the responsibility of the guiltily negligent persons should be smartly brought home to them. Real miners were getting scarcer and scarcer every day, and they were raising a class of man who had learned everything from sitting in the stope and watching the natives work. If a man failed as a grocer, a barber or painter, he went underground, hating it all the time, because he was pulled by an empty stomach. He seriously commended the mine training school at Wolhuter to those who intended to make their sons miners. Unless the class of miners—indeed all classes of workmen—strove to make themselves efficient, he would find that the native would be a keen competitor. There was an utter lack of co-operation among white employes to prevent accident.

Discussion followed, mostly of an interesting nature. Questions regarding the colour bar were ruled out as political, and in the end Mr. Archbold was very heartily thanked for his lecture.—(*Rand Daily Mail* report.)

AMBULANCE AND RESCUE APPLIANCES.

On Saturday night, the 23rd May, **Dr. Louis G. Irvine**, of the Crown Mines, author of "First Aid in Mining," lectured upon "Ambulance and Rescue Appliances."

Prof. J. A. Wilkinson (Chairman of the S.A. Red Cross Society) presided.

Dr. Irvine said:—The Exhibition Committee has made safety-first and rescue appliances a feature of this year's Exhibition, and I hope that they will continue to do so, so that opportunity may be given to the demonstration of the advances which take place every year in both of these directions. Safety-first and first-aid are both necessary, and are complementary, not antagonistic. It is with the view of encouraging improvement in first-aid equipment, that the S.A. Red Cross Society has offered medals and prizes for the best first-aid box suitable for underground use, for the best underground stretcher, and for the best additional device other than these and suitable for use by a first-aid team underground. The selection of rescue appliances shown in the Exhibition is very fairly representative, and is sufficient to enable one to demonstrate the lines of development on which they are proceeding. Organised first-aid work is now assuming a recognised place in the life of every industrial community, and I am convinced that so long as it is diligently taught, seriously studied, and adequately organised, public confidence in its value will steadily increase. This is the object of the South African Red Cross Society. Few mines at present have properly organised their first-aid and ambulance departments. But, in view of the number of accidents which occur yearly, it is essential that every large industrial undertaking, and especially every mine, should possess an adequately equipped and organised department to deal with them. Take a few facts and figures: On the gold mines of the Rand alone in 1912, 845 persons were killed by or died from the effects of accident, and, of these fatalities, 763 occurred underground. In addition, there were over 1,800 serious accidents, with their inevitable train of consequences of suffering, and disablement—temporary or permanent. This means a big sum in suffering and disability and loss of working time.

Somebody has to pay for all this in the long run, and although the cash payment does not represent the total loss to the individual and the community, the bill in cash is a large one. For the year ending March, 1913, the Rand Mutual Assurance Co. paid out, in respect to accidental injuries to white mine employees on the Rand, a sum of £92,000; if the whole of the gold-mining industry were included that total would be increased by about 50 per cent.

In order to secure efficient first-aid work, three things are needed:

(1) Every society or association which issues first aid certificates must guard the reputation of its certificates highly. They should not be issued to men who are not really competent. If there is to be competition between these bodies it should be competition for a high standard—not for a low one. Certificate-snatching should be altogether discouraged.

(2) Every man who takes up first-aid should take it up seriously. It is knowledge worth having, and any intelligent man can acquire it if he gives his mind to it. But that he must do, and he must improve his efficiency by continued periodic practice. The half-trained man is only a nuisance, and is only likely, when called upon, to bring discredit on himself and on the movement. The well-trained man may frequently prove of the utmost use.

(3) Every large industrial undertaking, and especially every mining company should organise its ambulance department as a routine part of its business organisation. Accidents frequently occur in mines at points remote from any possibility of medical assistance. The organisation should not be left to voluntary enthusiasm, although this should always be made use of. The mining companies during the past few years have given the movement generous financial assistance. But that is not enough, nor does the requirement of the Mining Regulations that certain mine officials should possess ambulance certificates, take us very far. During the past year a good beginning has been made on the Crown Mines. Six ambulance officers have been appointed on the staff of the company. They are not whole time men, but are part-paid for this work. The shafts of the mines have been divided up between them, and, whenever a serious accident occurs underground, we know that there is someone at hand to take charge of that accident on the spot. Our aim is that every serious accident occurring underground should be treated underground by first aid methods. We are fortunate in having, in addition, some 50 voluntary helpers organised as a Red Cross corps, and these have rendered great service. Their keenness is stimulated by the offer of handsome prizes by the company, to be competed

for annually by the employees. That is a sort of system which I think every company should adopt. First-aid will help safety first, for the more men are interested in seeing and dealing with accidents, the more will the causes of those accidents be forced on their attention, and the more they will think of means to prevent them.

Now I shall proceed to demonstrate some typical ambulance and rescue appliances. Take first ordinary first-aid equipment for mine use :

(1) The first part of the equipment should be a properly-fitted first-aid box. I show one fitted up by Messrs. Lennon to the requirements of the Red Cross Society. It is handy, portable, and contains everything in the way of splints, bandages, tourniquets and dressings that the first-aid man is likely to want. Note especially the long thigh splint in three sections, easily fitted together. Note also that the dressings for wounds are in small compressed packets, each containing enough for one wound. This is a great preventive of waste.

(2) As to stretchers : Transport of the injured underground and up the shafts of a mine is an all-important matter. To be suitable for use underground a stretcher must fulfil the requirements of portability in narrow places and protection to the patient from uneven surfaces, and also of providing means for securing the injured man, so that he may be carried vertically without his position on the stretcher being altered. The ordinary canvas stretcher does not fulfil these requirements. In my opinion, the stretcher which meets them best is the Red Cross plank stretcher, which has been evolved by the Crown Mines, and in its improved form is shown by Mr. Little in this exhibition. It is handy, portable, durable, will stand rough usage, and is readily cleaned. In underground work its rigidity is an actual advantage, and it adequately meets the requirement of securing the patient for transport in a vertical position. Members of the Crown Mines Ambulance Corps demonstrated the use of the first-aid box and the method of using the improved Red Cross stretcher to the audience

(3) Another device, also designed by Mr. Little, is the ambulance trolley, which is used in combination with the plank stretcher for conveying injured men along long and low drives to the shaft station. This device is also shown in the exhibition. It is, further, of great use in extricating an injured or unconscious man from a winze—*e.g.*, in gassing accidents and secures an enormous saving of time in this operation, where time is of the utmost value not only to the man injured, but to the rescuers. The use of the ambulance trolley was also demonstrated. These three things—the first-aid box, the plank stretcher, and the ambulance trolley—with rope and blanket, form, in my opinion, a useful equipment for an underground first-aid team

Coming now to the question of accidents due to poisonous gases. There are several devices which may be used or are recommended :

(4) The air mask or smoke mask, fed by a hose-pipe, deriving its supply from a pump or bellows, or from the compressed-air service pipe, although required by the Mining Regulations, is not a practical device in the circumstances in which it is supposed to be used—namely, in gassing accidents. . . . The ordinary winze can be blown out in 15 minutes or less if proper air-pipes are provided. The danger zone in cases of gassing in metalliferous mines is always limited. If deep winzes are sunk, say, to 500 ft. or so, the remedy lies either in providing more adequate ventilation appliances or in training men to the use of an oxygen breathing apparatus such as I am about to describe. And of these, the former alternative is unquestionably, in my opinion, the right one. The regulation requiring the provision of the smoke mask should be withdrawn from the Mining Regulations.

In coal mines the problem is quite different. Here, as in consequence of fire-damp and coal dust explosions or underground fires, the whole mine may become flooded with dangerous gases, and it may be necessary to seal up the mine altogether in order to starve out a fire. The whole mine may become a danger zone. The demand in this case is for an apparatus which is carried wholly on the body of the wearer, which will permit of free movement, and which will enable the wearer to do active work in a poisonous atmosphere for not less than two hours.

The lecturer then explained and demonstrated the uses of various oxygen breathing apparatus, such as the Proto, Meco, W.E.G., Draeger and Shamrock. He stated however that a caution was necessary. None but trained and experienced men should use them. It is of no use to have a couple of these things dotted about on mines or collieries with no one trained to use them. Even in experienced hands lives have been lost through their use.

Dr. Irvine also described and demonstrated the Pneumatogen apparatus : and referred to the Aerophore, in which liquified air is used to supply the oxygen required, and finally dealt with the various forms of oxygen reviving apparatus.

In conclusion Dr. Irvine expressed his thanks to Messrs Lennon, Reunert & Lenz, and W. Hosken & Co., for their kind permission to demonstrate the appliances mentioned

A hearty vote of thanks was extended to the lecturer for his lucid and excellent address.—(*Transvaal Leader* report abridged).

SAFETY MEASURES IN REGARD TO THE USE OF EXPLOSIVES.

Mr. Wm. Cullen, *Manager of the Modderfontein Dynamite Factory*, gave a lecture on the 26th May on the above subject.

Mr. Neuhaus, Chairman of the Safety First Committee, presided, and in the course of his introductory remarks, expressed his horror at the number of accidents which took place on these fields, and at the carelessness which was responsible for the great majority of them.

Mr. Cullen said that if some of the theories which he advanced were thought worthy of further consideration then he should be prepared to transfer them to the more debatable atmosphere of the scientific societies, but meantime a great many problems in connection with blasting required investigation, and he had no doubt whatever that it would be much more for the benefit of the mines than for the explosives companies if these problems were tackled in a thoroughly sound scientific manner. After all the consumption of explosives might be reduced temporarily, but inasmuch as this would broaden the basis of mining, the explosives companies would benefit in the long run.

Proceeding, he stated that at a time like this when the public conscience and public interest had been aroused by the Safety First campaign, recrimination and blame should be put aside. It was only by what the Americans called team work that anything great was accomplished, and this was a time for team work if ever there was. He would like to see a Safety First team on every mine—not made up of officials but of working men. He was not a great believer in formal lectures. Rather did he favour small informal "pow wows," but the "pow wower" must be a man who knew his job. He must have abounding patience, and above all he must be a good fellow. He would strongly advise the mines to get one or two such men and to pay them well. After recounting his own early experiences when lecturing on the mines, Mr. Cullen passed on to his subject proper. The first point he considered was "fuse lighters." He burned in a glass jar a small quantity of blasting gelatine, which produced a very large amount of nitrous fumes. It was because of these fumes that the old "cheesa" or firing stick stood condemned. He next proceeded to demonstrate how essential it was to have a "cheesa" stick which would remain alight under the most trying conditions—as a man's life frequently depended on the light of his "cheesa" stick. Incidentally he brought out the fact that a really satisfactory "cheesa" stick was very difficult to produce. He next passed on to the question of exudation, which, he stated, had in the past received practically no consideration from users. He explained how exudation was caused, and how dangerous it was under certain conditions. He was firmly convinced that many unexplained accidents were due solely to exudation, and although miners in many parts of the world liked "wet gelatine" it should on no account ever be used. Proceeding, he directed attention to the attempts not infrequently made to jam cartridges into a hole too small for them. This in itself was, of course, a danger even with a wooden tamping rod, but it was increased tenfold with exudation.

He next passed on to a somewhat lengthy consideration of fuse. Broadly speaking, he thought that the quality supplied on these fields was good, but he would like to see a better quality used, and better certainly with regard to regularity of burning speed. Such a fuse as he had in his mind might cost 1d. per coil more, but what was that compared to the occasional hanging up of a round or bench? He explained very clearly how certain conditions might produce a so-called running fuse—but he was very definite on the point that there was no such thing as a running fuse. He next passed on to the question of premature explosion, and although he stated that his ideas were for the greatest part theories, they had a very large measure of probability, and certainly what he had to say in this connection gave room for thought. Regulation No. 106 (22), which he read and which has to do with the interposition of tamping between cartridges, he supported strongly and showed clearly by experiment and example that the practice is highly dangerous and ought to be condemned. Although experiments which he had himself carried out did not support this view, still all explosives were not equally sensitive, and it was surprising what small things could lead to unexploded cartridges being left in the hole. Probably more accidents were the result of drilling into unexploded cartridges than any other cause, but after examining the various proposals which had been advanced to obviate this, he had reluctantly come to the conclusion that none of them were infallible, and this being so, they would only increase present dangers. Mr. Cullen said that a most important point and one to which insufficient attention had been given, namely, was the proper storage and handling of detonators. He contended that a very small quantity of sawdust could prevent the spit of the fuse reaching the detonator composition, and this in turn could lead to all sorts of dangers. He stated that "damp storage" was far more inimical to detonators than actual immersion in water, even up to an hour, and this anyone could try for himself. He also mentioned that if through carelessness the end of the piece of fuse got wetted, most probably the detonator would not go off. This could also be easily demonstrated. He next met some of the objections which had been advanced against the use of the "Nowag" clip. In practice in a great many of the mines the cartridge was not properly affixed to the fuse and detonator as was provided by the regulations, and the clip was an easy way out of the difficulty. A large number of views, some of them extraordinarily good, were next thrown on the screen, and as they passed along Mr. Cullen made each the subject of a few interesting remarks.—(*Rand Daily Mail* report.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Assay of Tin Ores.—"Experiments in which tin ores and tailings were heated under various conditions with zinc oxide, lime, etc., showed that by heating with lime for 15 to 20 min. over a large Teclu burner, it is possible to convert the whole of the tin into calcium stannate, which can then be dissolved in hydrochloric acid, and the tin estimated volumetrically in the usual way. The new process is a modification of the 'Beringer' method."—H. W. Hutchin, *Inst. Min. and Met.*, Feb. 19, 1914; *Jour. Soc. of Chemical Industry*, March 16, 1914, p. 262. (J. G.)

The Assay of Tin Ores and Concentrates.—"A series of tests in which various minerals were added to tin ores and concentrates of known tin content, and the mixture then assayed for tin by the Pearce-Low method (Low, 'Techn. Methods of Ore Analysis,' Third Ed., 1908, p. 208) proved that the method is very accurate except when compounds of titanium, tungsten, or bismuth are present. The modification proposed when titanium is present, is to fuse the ore with potassium bisulphate, and treat the melt with dilute sulphuric acid, when all the titanium goes into solution, and the residue, containing all the tin, is fused with sodium peroxide in the usual way. Tungsten, if present with titanium, can be completely removed by treating the residue from the titanium solution with ammonium carbonate. Ores containing tungsten may also be digested, after finely powdering, with sodium hydroxide solution, when all the tungsten goes into solution. When tin ores contain bismuth, the bismuth is precipitated by the nickel. As the precipitate consumes iodine, it must be removed by filtration, and the solution again reduced before titration. In general, in carrying out the Pearce-Low assay, the authors recommend grinding the sample to 100 mesh at least: fusion in nickel crucibles, and reduction with nickel; the temperature of the solution at titration to be below 70° F. (21° C.); avoidance of large amounts of cuprous chloride and ferrous chloride; the bulk of the solution before reduction to be 400 cc., containing 125 cc. of hydrochloric acid"—E. A. Wraight and P. L. Teed, *Inst. Mining and Met.*, Feb. 19, 1914; *Jour. Soc. of Chemical Industry*, March 16, 1914, p. 262, (J. G.)

THE ASSAY OF TIN ORES AND CONCENTRATES.—THE PEARCE-LOW METHOD.—"*Summary of Authors' Views on the Pearce Low Assay.* As the various experiments and deductions made therefrom are to some extent spread over the whole paper, a summary will now be given:

1. The degree of fineness of the ore must be at least 100 mesh, otherwise a representative sample cannot be obtained.

2. Nickel crucibles are superior to iron ones; the time of reduction is not so long if nickel crucibles are used for fusion. For tailings, fusion in an iron crucible should be avoided; it is desirable to keep the amount of ferrous chloride in the solution as low as possible, particularly when the tin content is small, otherwise high results are obtained.

3. The amount of hydrochloric acid should be about 125 cc.

4. The bulk of the solution before reduction should be about 400 cc.

5. The temperature of the tin solution at titration should not be more than 70° F.

6. The strength of the standard solution should not generally be more than 11 gm. of iodine and 20 gm. of potassium iodide per litre, or less than one-third the above strength.

7. Before titration the calcite should have entirely dissolved.

8. Titanium, tungsten and bismuth must be removed by the means already described before titration.

9. Copper and iron should, under special circumstances be removed.

10. Nickel should always be used for reduction.

If these precautions are taken and care is exercised in the fusion and other manipulation, the authors are of opinion that the error in the results obtained should certainly not exceed 2 lb. of black tin per ton with rich ores, and 1½ lb. per ton with low ores."—E. A. WRAIGHT and P. LITHERLAND TEED, *Bulletin, I.M.M.*, No. 113, Feb. 12, 1914. (C. T.)

THE ACTION OF SULPHURIC ACID ON COPPER.—"The present investigation shows that cuprous sulphate, rather than cupric sulphate, is one of the primary products, as may easily be tested by pouring off the sulphuric acid at any stage of the reaction through a Gooch filter into water, when a precipitate of finely divided copper is produced."—(The late) JAMES TUDOR CUNDALL, *Proc. Chem. Soc.*, Vol. 29, No. 421, p. 344. (C. T.)

THE ESTIMATION OF MERCURY.—"Penfield's well-known method for determining water, especially in minerals, has been adapted for the determination of mercury in ores and minerals. In a piece of glass tubing 20 cm. long and 5 mm. wide, two bulbs are blown, one at the closed end and another near the open end. A small quantity of the powdered sample mixed with lime is filled into the clean, dry tube through a drawn-out glass funnel. The tube is then filled up to a depth of 8 cm. with iron filings and quicklime. An asbestos plug is pushed in midway between the bulbs, and the open end of the tube drawn out to a fine capillary. The end bulb, with its charge, is heated at first inside an iron pipe, finally in the flame itself, the mercury being driven out and condensed in the second bulb, which is kept cooled. This bulb, with the mercury, is then drawn off, the moisture present is removed by a current of dry air, and the bulb and its contents weighed. The mercury is then run out and completely removed by drawing a current of air through the hot tube. Finally the empty tube is weighed. If sulphides are present lead chromate should be added to the mixture before heating."—CUMMING and MACLEOD, *Journal Chem. Soc*, 103, 513; *The Mining and Engineering Review*, March 5, 1914, p. 171. (J. E. T.)

THE ELECTRO-DEPOSITION OF ZINC AT HIGH CURRENT DENSITIES.—"At certain very high-current densities the electro deposition of zinc can be effected in presence of a high concentration of free acid. Under these conditions the ratio of zinc to hydrogen liberated actually increases with the acid concentration up to a certain value, and also increases with the current density. In this way, with a concentration of sulphuric acid of about 15 gms. per 100 cc. and a current density of between 20 and 50 amps. per sq. dcm the metal can be deposited with an efficiency of about 95°.

With lead anodes this electrolysis is achieved with a potential difference about 5 volts and with zinc anodes 3 volts.

The presence of small quantities of colloidal matter exerts a marked effect on this reaction, and enables the production of bright, adherent deposits. The presence of the colloid also enables the application of a higher current density, and in this way raises the current efficiency.

In these solutions a very strong retardation was observed in the deposition of iron present in the electrolyte. On account of this, considerable quantities of this metal in the electrolyte caused very little contamination of the zinc.

The results obtained could not be entirely ascribed to effects of over-voltage, or surface tension, or viscosity of the electrolyte, and are probably mainly determined by influences which control the rate of the reactions involved in the change from the ionised to the free element.

The results obtained in this work indicate the most favourable conditions under which zinc can be obtained from commercial solutions, either with a view to the recovery of the metal or to its application for the purpose of electro-plating."—J. NORMAN PRING and URLYN CLIFTON TAINTON.—*Proc. Chem. Soc.*, Vol. 30, No. 421, pp. 27-8. (C. T.).

DETERMINATION OF ANTIMONY AND ARSENIC IN LEAD-ANTIMONY ALLOYS AND ANTI-FRICTION METALS.—"From 1 to 10 gm. of the metal are heated with 20—60 cc. of sulphuric acid until any precipitated sulphur has redissolved and sulphur dioxide has been completely expelled, the solution is diluted with its own volume of water, and arsenic and antimony determined together by titrating the boiling solution with permanganate (the iron factor of the permanganate multiplied by 1·0714 gives the equivalent amount of antimony and arsenic in terms of antimony). Another portion of the metal is heated with ferrous sulphate and hydrochloric acid, and arsenic determined in the distillate with iodine solution as usual. If much iron be present, it must be determined separately and deducted from the antimony-arsenic result. Tin, bismuth, and zinc do not interfere with the determination, nor does copper unless present in such quantity that its colour affects the end point in the permanganate titration."—L. BERTIAUX, *Ann. Chim. analyt.*, 1914, *19*, 49-51; *Jour. Soc. of Chemical Industry*, March 16, 1914, p. 262. (J. G.)

SOLUBILITY OF SILICA IN HYDROCHLORIC, NITRIC AND SULPHURIC ACIDS.—"When hydrated silica is evaporated to dryness with hydrochloric acid, the residue heated to 110° C., and the silica separated in the usual way, about 0·75% remains in solution, but may be recovered by evaporating the filtrate to dryness and collecting the insoluble residue thus obtained. Similar results are found when nitric acid is used, but, owing to the difficulty of removing all the nitric acid by washing, there is a tendency for the results to be too high. About 3·5 % of the weight of the silica remains in solution when sulphuric acid is used (in this case, the mixture cannot be evaporated to complete dryness), but the loss is recovered by a second evaporation. Hydrated silica was found to dissolve in dilute hydrochloric acid of various strengths (1 : 2 ; 1 : 1 : 2 : 1) to the extent of from 0·03 to 0·33 gm. per litre, the largest quantity being dissolved at 100° C. by the 2 : 1 acid. Nitric and sulphuric acids dissolved smaller quantities, namely, from 0·02 to 0·1 gm. per litre."—W. **Wunder** and A. **Suleimann**. *Ann. Chim. anal. appl.* 1914, *19*, 45-49 ; *Journ. Soc. of Chemical Industry*, March 16, 1914, p. 279. (J. G.)

METALLURGY.

Grinding Ores for Cyaniding.—"Referring to the letter by George E. Collins on 'Grinding Ores for Cyaniding,' where he compares the work of the tube mill with that of the Hardinge mill, I assume his tube mill is the ordinary type having the central overflow discharge. From both of these machines the pulp is discharged by displacement, and as Mr. Collins says, crowding the feed beyond the capacity of the mill for complete grinding will result in a coarser product, or more oversize.

Careful consideration will, I am sure, make it clear to any tube mill operator that there can be no zone in pebble mill where all of the particles of ore are of a desired size. This is true regardless of type of discharge. The varied sizes in the feed and the hindered progress of the particles of ore through the mass of pebbles, make it quite impossible for such a condition to be. It is because of this fact that the practice has become quite general of operating the tube mill in a closed circuit with a classifier, even with a central overflow discharge. Since, therefore, we find particles of the finished size throughout the entire cross-section of the charge, we are bound to get the best results from a mill which offers the easiest passage to such particles, to the outlet.

It is evident that the conical end even though filled with pebbles, offers an easier passage than the square end. The crowding action of the feed can act from the centre to the periphery in moving particles up the incline, but it is evidently much more difficult to force out the particles in the corner.

Many years ago, a device was developed at the El Oro Mines, El Oro, Mexico, to overcome the dead corner. This is like that in Fig. I. It consists of the introduction of a false head made of perforated plates secured to the discharge end head by means of bolts and distance pieces, forming a space free from pebbles into which particles from the entire cross section of the charge can freely pass and freely rise to the outlet.

This device improved the discharge in just the same manner that the conical end does, but to my mind in a greater degree, for I agree with Mr. Collins in being unable to see how there can be any really effective stage crushing in this cone.

In the mills of the Cobalt Lake and McKinley-Darragh Mines, at Cobalt, Ontario, I had an opportunity to compare the efficiency of an Allis Chalmers 5 ft. × 20 ft. mill with the ordinary central overflow discharge ; an 8 ft. Hardinge and a Chalmers and Williams 5 ft. × 16 ft. mill fitted as in Fig. I., with perforated end plates and reverse-screw pebble feed, with the result that the efficiences were in the reverse order, the Chalmers and Williams having the highest, the Allis-Chalmers having the lowest.

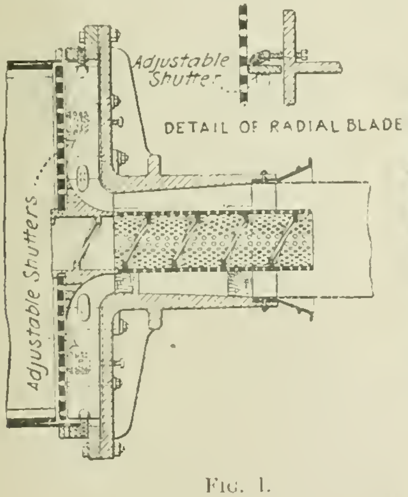

Fig. I.

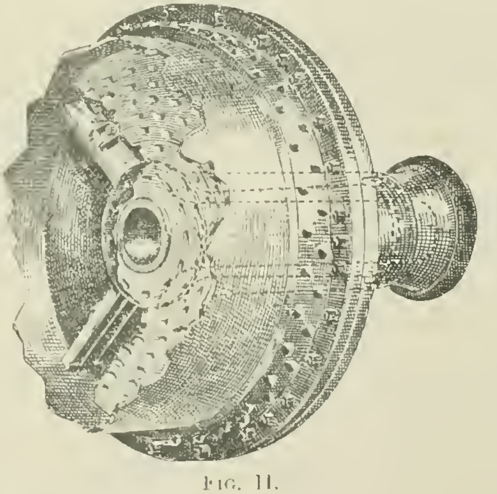

Fig. II.

The Rapid Discharge System for Tube Mills.

It is doubtless the effect of the dead corner that is responsible for tendency on the part of tube mill users to lean toward shorter lengths, when using mills of this type as there is evidently a critical length of mill for each individual case beyond which the displacing effect of the feed is lost, due to the hindering action of the pebble load. The pebble load beyond this point is merely consuming power and using up pebbles uselessly.

The Chalmers and Williams adjustable quick discharge was developed to facilitate the passage of the pulp through the tube mill to an even greater extent than the inclined end or the device shown in Fig. I.

It consists of lifters so placed in the space between the perforated plates and the discharge-end head as to lift up and send out, through the trunnion, the pulp as soon as it accumulates. This action tends to create a void in the space behind the perforated places and can be varied from zero to full effect.

In general, the most efficient mill should be the one that provides to the greatest possible extent for the removal of the particles of ore as soon as they have reached the finished size, for even though a large percentage of the discharge is returned to the tube mill as oversize, the useful work of the mill is increased by the exact amount of finished material removed to make room for so much more new material. I would like to emphasise this idea, for I do not believe that tube-mill users are keenly enough alive to the appreciable amount of both power and pebbles that are wasted on the finished material which is prevented from getting away.

By the use of the quick-discharge device, the entire length of a long tube mill can be made effective for fine grinding, thus increasing the capacity over the ordinary central overflow-discharge type of tube mill, or a greater capacity can be obtained in short mills for coarser grinding, for I believe that where an all-slime product is desired, with this device the long mill will be the most efficient. But as I have said above, there is undoubtedly a critical length for tube mills beyond which they become less efficient on account of the hindering action of the charge of pebbles to the passage of particles of ore through the mill, which no type of discharge can entirely overcome."—W. B. EASTON.—*Engineering and Mining Journal*, April 4, 1914, p. 723. (H. A. W.)

MELTING-POINTS OF COMMERCIAL BRASSES AND BRONZES.—"As the result of tests made under the direction of the United States Bureau of Mines, the following tables of the melting-points of commercial brasses and bronzes has been summarised :—

	Approximate Composition.				Melting-point.	
	Copper.	Zinc.	Tin.	Lead.	C°.	F°.
Gun metal ...	88	2	10		995	1825
Leaded gun metal	85½	2	9½	3	980	1795
Red brass ...	85	5	5	5	970	1780
Low-grade red brass	82	10	3	5	980	1795
Leaded bronze ...	80	—	10	10	945	1735
Bronze with zinc	85	5	10	—	980	1795
Half-yellow, half-red	75	20	2	3	920	1690
Cast yellow brass	67	31	—	2	895	1645
Naval brass ...	61½	37	1½	-	855	1570
Manganese bronze	—	-	—	—	870	1600

The melting-point given is the 'liquidus,' or point where the alloy is completely molten. The tests are described in detail in Technical Paper No. 60. —ANON., *Amer. Mach.*, xxxix., No. 24, 984.—*Journal of the Franklin Institute*, April, 1914, p. 463. (J. G.)

MINING.

THE TECHNICAL COMMITTEE.—"A new departure appears to have been made in the case of the Burma Corporation. Instead of employing a consulting engineer to guide the mining and metallurgical operations, the board of directors have co-opted a committee of three technical experts, giving them seats on the board, so that they become a sub-committee of the central administration. Something of the kind was foreshadowed by the reference to three engineers, instead of the usual solitary advisor, in the case of the Russo-Asiatic Corporation. Apparently men of diverse experience and attainments are to be retained, so that the varied mining and metallurgical problems are to be submitted to individuals having special aptitude for solving them in terms of eventual dividends, while, at the same time, this committee of experts includes at least one man of such calibre as to be able to envisage the whole scheme of operations. This departure from established custom is bound to prove highly interesting to the profession and to those dependent upon the profession for guidance in the complexities of the mining industry. Frankly, we deem it advantageous to all concerned. The multitudinous phases of modern mining and metallurgy cannot all be within the compass of any one engineer's equipment, whether young or old, and special knowlege concerning one of them is usually obtained at the expense of the others. Even the youngest of us is not infallible. Those who know most are best aware of their deficiences. Co-operation is the mark of a higher civilisation. Generalities apart, those engaged in mining are made often unpleasantly aware of the importance of the personal equation ; hence they will appreciate the usefulness of a new factor likely to correct the aberrancies of the individual. Every engineer, like ordinary mortals, has his times of maximum and minimum efficiency, as modified by temperament, physical health, or mental vigour. It happens occasionally that a critical decision, a choice of methods or a line of policy, has to be made at a moment when a sluggish liver or a domestic disturbance renders it difficult to intent the whole force of a normally capable mind upon a given subject. To decide is imperative. At such a juncture it is not only comforting to discuss the matter with two capable colleagues : it is an obvious safeguard against error. This is true of most human affairs. For example, the general administration of a company is commonly in the hands of a chairman or some other dominant member of the directorate, presumably checked by consultation with his associates on the board. The consulting engineer, to whom matters of chief importance are submitted, stands alone ; for technical problems of any intricacy are generally beyond the capabilities of a board, which, moreover, usually desires to escape all responsibility in such matters. Apart from the personal idiosyncrasies that we try to ignore in business, the fact remains that the special branches of technology are now so numerous and the application of them in actual mining operations arises so suddenly that it is an enormous advantage to fall back on the resources of more than one man however gifted. In any series of technical operations there are single points that are apt to be overlooked, to the subsequent chagrin of directors and shareholders alike. It is better to be sure than sorry. The breaking of ore and the extraction of metal constitute a small part of the processes involved. These must include provision for supplies of water and labour, the peaceful control of workmen, the establishment of pleasant relations with the local government, the vagaries of metal markets, the financial considerations with which mining begins and ends, with their ultimate bearing upon working capital equipment, and dividends. Of course, these are matters vaguely recognised at the inception of any mining enterprise, but one or other of them usually escapes proper attention owing to the fact that they are not of equal interest to a new board of directors, while the consulting engineer cannot be in two places at once, on the mine and in the London office. Moreover, the solitary engineer faces a board of five, seven, or nine directors, and must be a man of unusual force not to feel the odds against him in matters concerning which there is disagreement, especially when one of the directors, as is frequently the case, controls the position, supported by assenting dummies. In such a case the technical committee carries greater weight, with less insistence, and is consulted by the board in a spirit of larger deference. In short, the technical committee becomes a powerful sub-committee of the administration. Under these conditions, we suggest, the Great Cobar fiasco, for example, would have been much less likely, if not impossible. The financial blunders, the technical errors, and the break-down in management, were due in large part to asking engineers to estimate the capacity of a mine without regard to the working capital to be provided, expecting a copper metallurgist to act as the chief executive of a big business, and throwing a multiplicity of problems at the head of a single engineer. After all, the cost of technical advice represents a cheap form of insurance. It is the one item of expense that is treated with most uneconomic niggardliness. In any important undertaking the difference between the retainers paid to three, as against one, is negligible, if the desired result be achieved. Moreover, each one of the three can afford to accept appointment for a smaller fee than he would ask if employed alone; the responsibility is shared; the risk of error is diminished; the anxiety is lessened to no considerable degree. The engineer in demand for such honourable engagements can serve no more than one of these committees and enjoy greater freedom than if he were the sole advisor of one big enterprise. Speaking broadly, we believe that this departure tends to enhance the usefulness and the dignity of the profession. It will make a desirable inroad upon large and unwieldy boards of directors, by eliminating the mere dummies and stool-pigeons, giving their places at the council-table to the engineers, while at the same time affording the forceful and hard-working director the sort of colleagues he needs most. We assume that not only will the members of the sub-committee be directors, but that their share-holding will be limited to the legal qualification, to be furnished for them in compliance with the law, but without any idea of participation in speculation. We hope, also, that each member of the technical committee will be prepared, and will be asked, to give his word not to deal in the shares of the company with which he is professionally connected. For those who look upon mines as a means of financial legerdemain and for those who regard them as counters in a gamble controlled from the board-room, these ideas will prove most objectionable, but to the real captain of industry, as distinguished from the *chevalier d'industrie* the innovation will have obvious merit."—*Mining Magazine*, June 1914, p. 404. (H. M.)

METHANE INDICATORS.—"A new form of methane indicator for the detection of fire-damp in mines is described. It depends on the principle that differences in chemical composition of a gas are indicated by differences in the sound resulting when a pipe or whistle is blown by means of the gas. The apparatus is called the 'fire-damp whistle,' and consists of a closed brass cylinder, 25 × 6 cm., containing 2 stopped pipes tuned to the same pitch when filled with the same gas. One pipe communicates with the outer atmosphere and the other with the air of the mine, freed from dust, CO_2, etc., by suitable apparatus within the cylinder. The apparatus is operated by pulling down the lower part of the cylinder, thus drawing in the air, the partial vacuum thus produced in one portion of the cylinder returning the piston and forcing the air through a pressure regulator to the mouthpieces of the pipes. With 1% CH_4 about two beats per second are produced, the number of beats increasing rapidly with the CH_4 content until near the explosion limit (about 5·5%) a characteristic thrill results. The sound is perfectly audible in a mine at distances of over 100 metres. This indicator has given practical results and has advantages of safety not obtained by indicators depending on the presence of sparks, incandescent wires, etc. A sketch and detailed description are given."—F. HABER, *Chem. Zeit.*, *37*, 1329-30. *Chemical Abstracts*, Feb. 10, 1914, p. 575. (J. G.)

CATCHING COMPRESSED AIR LEAKS. An Illinois manufacturer, recorded in *Factory*, at fixed regular intervals, puts essence of peppermint into the air tank and then has the entire system inspected for leaks. If the slightest amount of air escapes it is detected by the odour.—*Engineering and Mining Journal*, No 6, Feb. 7th, 1914, p. 324. (G. F. A.)

New Books.

Clowes, Frank and Coleman, J. B. Elementary Practical Chemistry. Part II. Analytical Chemistry. Qualitative and Quantitative. 7th Ed. Cr. 8vo., pp. 258. *Churchill.* Net 3s. 6d.

Forster, George. Safety Lamps and the Detection of Fire-Damp in Mines. 8vo., pp. 92. *Routledge.* Net 1s. 6d.

Freeman, Nat H. Baumé and Specific Gravity Tables for Liquids Lighter than Water. 18mo. *Spon.* Net 2s. 6d.

Harrison, B. A. Practical Well-Sinking. Cr. 8vo., limp. *Technical Publishing Co.* Net 1s. 6d.

Havelock, J. H. Electric Wireman's Work. A Practical Handbook. Cr. 8vo., pp. 340. *C. Lockwood.* Net 5s.

Hobbs, W. H. Simple Directions for the Determination of the Common Minerals and Rocks. 8vo., swd. *MacMillan.* Net 1s.

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Abstracts of Patent Applications.

(C.) 285/13. Robert Robson (1). Salters (2). Regulating feed for sand cones. 20.6.13.

This patent application has reference to means for controlling the underflow of pulp separating apparatus such as sand cones, etc.

The apparatus consists of a nozzle plate fitted with two nozzles so placed that either of them may be brought at will under the orifice of the classifier. The nozzles are interchangeable and fit into recesses in the plate, where they are retained by a flange or taper. Pressure is exerted on the nozzle plate so as to keep it close against the cone by means of a set of springs placed between supporting brackets secured to the pressure plate.

(C.) 298/13. Alfred Whitby (1), Josiah Whitby (2). Improved pole for mine packs. 26.6.13

This invention relates to mine pack holes prepared by bolting bearing blocks to each end, so that the usual pigsty construction may be built up with these. One claim is for having the heads and nuts of the bolts which secure the blocks to the timber so placed that when built into a pack the ends of the bolts are locked against outward displacement.

(C). 374/13. John Oliver. Improvements in tube liners. 14.6.13.

The above refers to a tube mill liner consisting of a number of bars placed in the shell of the mill longitudinally to its axis, and held in position by means of segmental rings suitably placed.

The rings are provided with recesses where the bars are placed and keyed by means of wooden, metal or any suitable wedges driven between their outer edges or faces and the shell of the mill. The idea is to keep the bars at a suitable distance from the shell of the mill so as to allow to wear away and utilise practically the whole depth of the bars for retaining a portion of the charge.

Selected Transvaal Patent Applications.

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INDEX.

INDEX TO VOL. XIV., 1913-1914.

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Those portions in Italics refer to Original Notes and Papers read before the Society.

A

PAGE

Abstracts, Chemistry 9. 73, 152. 200, 270, 309, 345, 382, 414, 434, 466, 499
——, Metallurgical 11, 75, 155, 204, 274, 314, 347, 384, 415, 436, 467, 501
——, Mining 31, 98, 161, 219, 279, 330, 358, 386, 417, 440, 502
——, Miscellaneous 39, 107, 165, 223, 282, 333, 361, 388, 441, 468
—— of Patent Applications 47, 110, 230, 364, 391, 419, 442, 504
Accident and Death-rate, Relation of, to Rainfall 395
—— with a Benzole Locomotive at the Rossleben Potassium Salt Mines. Ziebarth 360
Accounts, Mine and Mill, A System of Keeping. M. W. Maclachlan .. 138
Accuracy in Reading the Balance, Limit of 70
Acetanilide, Preservation of Hydrogen Peroxide, by means of A. M. Clover 414
Acetylene E. E. Smith 441
Acid Proof Lining for Tanks 388
—— Solutions in Cyanide Work 78
Acidity and Alkalinity of Waters, The Determination of Dr. J. Moir 114
Discussion 118, 294
Do. Reply to 404
—— and Regeneration 357
Acids, Solubility of Silica in Hydrochloric, Nitric and Sulphuric W. Wunder and A. Suleimann 500
Action of Oxidisers in Cyaniding M. Green Reply to Discussion .. 65
Address, Changes of 48, 112, 168, 232, 288, 336, 364, 392, 420, 444, 468, 504
——, *Inaugural A. Richardson (President)* 51
Advisory Technical Directors 502
Africa and Madagascar, Graphite Deposits of, some Observations on the, E. R. Bawden Reply to Discussion 1
Aghan, W. H.—Improved Process for Separating Metals from their ores. (Patent abstract) 442
Agitating in Barren Solution 77
Agitation, Cost of, ... 439
—— for Cyaniding. Slime H. A. Megraw 436
——, Mechanical, Varied form of 437
——, Objects of 439
—— Tests in Cyanide Solution 12
Agitators, Air-Lift 437
——, Types of 18

PAGE

Agricola, Georgius, De Re Metallica (Review) 45
Agriculture, Cyanide in E. J. Pranke 153
——, Dynamite in C. Dusserre .. 228
——, The Use of Explosives in 284
Air, Amount of Dust in 98
—— Blasts 331
—— Blasts in Indian Mines 34
——, Compressed, *versus* Hydraulic Power for Rock Drills G. A. Denny 163
—— Compressors 279
—— ——, Explosions in 284
——, Hydrocyanic Acid in 123
—— Leaks, Catching Compressed 503
—— Lift Agitators 437
—— in Mines, Formula for the Friction of 197
——, Nitrogen from, Extraction of E. M. Wiley 272
Alaska Treadwell Mill. Tube Mill Circuit at the 320
Alcohol as Fuel in South Africa 203
Alkali, Protective 357
—— *Trade, Some Notes on the, The Natural Soda Deposits of Africa; with John Watson* 235
Discussion 241, 377
Do. Reply to 463
Alkalinity or Acidity of Waters, The Determination of Dr. J. Moir 114
Discussion 118, 294
Do. Reply to 404
—— of Potable Waters 295
—— and regeneration 357
Allen, A. W.—Simplification of Gold Ore Treatment 348
—— ——, Solution Control in Cyaniding 354
Alumina, Anhydrous, as a Desiccating Substance 272
Aluminium, Analysis of 274
——, Chemical Analysis of, J. Czochralski 312
——, Coating Iron with 276
—— Corroded by Mercury 346
—— Dust, Emulsifying Apparatus for .. 316
——, Electrometallurgy of Fedotieff and Iljinsky 11
—— Nitride, Reactions of F. Fitcher and A. Spengel 155
—— Precipitation 325
—— ——, Cost of 317
—— —— at Deloro, Ontario S. F. Kirkpatrick 316
—— —— at Nipissing E. M. Hamilton 157
——, Solder for 276
Amalgamation Process, Improved A. Argles (Patent Abstract) 391

PAGE
Amalgamation, Simple 348
—— Tests of Heating Mill Solutions 12
Ambulance Appliances at the Third Annual Mining Exhibition 491
—— Competitions 113, 169 289
—— and Rescue Appliances (Lecture) Dr. L. G. Irvine 497
Ammonia-Soda Process 239
Ammonium-Molybdate Assay of Lead, The Effect of Tungsten on the. H. Lavers 466
—— Sulphate Industry 74
Amylacetate as Absorbent of Nitrous Gases 272
Analyses, Grading, by Elutriation H. Stadler 85
Analysis of Aluminium, Chemical J. Czochralski 312
—— Coal, Accuracy and Limitations of A. C. Fieldner 313
——, Screen, Standard Screens for R. H. Richards 12
Ancient Copper Workings 194
—— Workings at Messina 53, 55
Anderson ——, Explosive Efficiency 360
—— W. T., Johnson, E. H., and Nethersole, R. R. Improvements in Treating Mine Dumps, etc. (Patent abstract) 442
Andrew, I. I. The Dissolution of Gold and Silver in Cyanide Solutions 383
Andrejew, J. J. Retarding Effects on the Dissolving of Gold and Silver in Aqueous Cyanide Solutions 312
Annealing of Gold, The T. K. Rose .. 276
Annual Dinner, The 169, 266, 290
Anstey, N. (Mayor) at Annual Dinner .. 266
Anti-friction Metals, Determination of Antimony and Arsenic in Lead, Antimony Alloys and L. Bertiaux 500
Antimony and Arsenic, Determination of, in Lead Antimony Alloys and Anti-friction Metals L. Bertiaux .. 500
—— —— in Ores 329
Antiseptics and Disinfectants D. Sommerville 389
Archbold, A. M. Mine Accidents on the Witwatersrand (Lecture) 496
Argles, A. Improved Amalgamation Process. (Patent abstract.) 391
—— ——, Improvements in Apparatus for Treatment of Ores, etc. (Patent abstract.) 391
Argon and its Place in Nature. Sir W. Ramsay 75
Arnold, M. Improvements in Ore Pulverising Machines. (Patent abstract) 110
Arsenic and Antimony, Determination of, in Lead-Antimony Alloys and Anti-friction Metals L. Bertiaux .. 500
—— —— in Ores 329
——, Detection of, by Bettendorf's Reagent L. W. Winkler 9
——, Hostile Influence of, upon Precipitation 158
Artificial Coal from a Wood Steamer E. Heuser 334
Ash, Coal, Rare Metals in A. Jorissen 316

PAGE
Ashcroft, E. A. Improvements in the Electrolytic Manufacture of Light Metals, &c., &c. (Patent abstract) 391
Ashford, J. Improvements in Tube Wells (Patent abstract) 110
Assay of Gold and Silver by Iron Nail Method E. J. Hall and C. W. Drury 415
—— *of Mine Samples, Notes on the, Robert Dures Discussion* 68, 173
Reply to Discussion 263
Assaying Concentrates and Battery Chips for Gold and Platinum Metals, A Method of A. F. Crosse 373
Discussion 374, 422 483
—— by Iron-nail Method W. J. Sharwood 345, 415
—— and Sampling, Coarse Gold in F. White 80
Assays, Referee 367
Associates New 49, 113, 233, 289, 365 421, 445, 474
Atack, F. W. The detection and estimation of Nickel by means of α—Benzildioxime 345
Auditors, Election of 474
Aymard, Dr. J. L., on Prevention of Dust in Underground Workings 483
Azides for Detonators Neitzel 282
——, Sensitiveness to Light of Wöhler and Krupko 270

B

Bag Precipitate Filter 84
Baker, H. B. The Spitting of Silver .. 467
——, H. H. Indian Gold Mining Methods 279
Ball or Pebble Mills, Improvements in Linings for H. A. Hansen and J. M. Schmueser (Patent abstract) 392
Balliet, L. Efficiency in Machine Drilling 220
——, Mistakes made in Mine Management 387
Banket 361
Banks, C. A. Dewatering Tank 215
——, J. H. G., on Cyaniding of Gold and Silver Ore I
Bannister, C. O., on the Effect of Charcoal in Gold-Bearing Cyanide Solutions 342
—— and Patchin, G. Detecting Platinum Metals in Beads from Cupellation 435
—— ——, *Detection of the Platinum Metals in Cupellation Beads by means of the Microscope* 478
Discussion 483
Barium Peroxide as Part Cargo, Danger of P. V. Dupré 345
Barker, A. E. Improvements in Bearings for Conveyor Belt Idlers. (Patent abstract) 110
Battery Chips and Concentrates, A Method of Assaying, for Gold and Platinum Metals A. F. Crosse .. 373
Discussion 374, 422, 483
Bawden, E. R. Some Observations on the Graphite Deposits of Madagascar and Africa. Reply to Discussion I

PAGE
Beads, Foreign Matter on 68
——, Frosted 71
Beaker for Analytical Work, Modified .. 274
Beams and Props in Mines. Reinforced Concrete Prof. S. M. Dixon 31
——, Tests of Reinforced 32
Beck R. Conglomerate of the Witwatersrand 105
Bell, H. D., on Assaying Concentrates and Battery Chips for Gold and Platinum Metals 483
——. J. W. Grading Analyses and their Application to Cyaniding 211
Bennett, V. C. Lead Salts in Cyaniding 213
Benzildioxime, α, Detection and Estimation of Nickel by means of F. W. Atack 345
Benzole Locomotive at the Rossleben Potassium Salt Mines. Accident with a Ziebarth 360
Bernewitz, M. W. von The Berrill Cam 214
——, Cyanide Practice (Review) 390
——, Dry *versus* Wet Crushing at Kalgoorlie 30
Berrill Cam, The M. W. von Bernewitz 214
Berry, S. L. The Pinder-Berry Stamp Mill 89
——, Stamp Mill Curves and Speeds 155
Bertiaux, L. Determination of Antimony and Arsenic in Lead Antimony Alloys and Antifriction Metals .. 500
Bettendorf's Reagent, Detection of Arsenic by L. W. Winkler 9
Bhattacharyya, Haripada. A Rapid Method for the Determination of Phosphorus in Steel 347
Black Sand, The Recovery of J. M. Neill. Discussion 72
Reply to Discussion 133
Blane, R., and Silcock, J. D. Improvements in Rollers for Haulage Ropes. (Patent abstract) 442
Blast Furnace Dust by Electrical Precipitation. Separation of W. A. Forbes 415
Blasting, Electric W. Cullen, T. Donaldson and W. Waters 174
Discussion 189, 248, 308, 375
Do. Reply to 431
—— —— and Fuse, Comparative Safety of 103
——, Important Improvement in W. Macnab 441
—— in the Lake Superior Iron District, Method of C. S. Hurter .. 103
—— Time in Sinking Operations 101
Blenkinsop, G. H. Transvaal Tin Deposits 33
——, *Ventilation of the Mines of the Rand: the Problem of Obtaining Healthier Conditions* 148
Discussion 151, 195 244, 306
Do. Reply to 402
Blood Pressure Variations F. E. Clough 228
Bogert, M. T. The Chemical Industries and the Universities 311
Böhm, C. R., Mantles for Incandescent Gas Lighting 44

PAGE
Boiler Cleaning System, Pyro 285
Books, New, and Reviews 45, 109, 151, 166, 230, 286, 335, 362, 381, 389, 418, 441, 503
Borax, Deleterious Effect of, on Crucibles.. 72
Bowen, D. C. and Dodds, W. B. Improvements in Linings for the Ends of Tube Mills. (Patent abstract) 364
Boydell, H. C., on Ventilation of the Mines of the Rand 245
Brasses and Bronzes. Melting Points of Commercial 502
Breakell, T. Rotary Vacuum Filter .. 214
Breakers, Dustless 286
Brokaw, A. D. Precipitation of Gold by Manganous Salts 201
Bronze Medal of Society, Presentation of.. 475
Bronzes and Brasses. Melting Points of Commercial 502
Brown, F. C. Tube Mill Practice 467
——, J. W. South African Coal for Gas Manufacture 274
——, Dr. Johnstone (Obituary) 344
Browne, C. A. Cyanide from Sugar Mill Residues 202
Brühl, Paul T., on the Effect of Charcoal in Gold-bearing Cyanide Solutions.. 343
Bucher, J. E. Method of Producing Metals. (Patent abstract) 391
Bullion. Determination of Platinum in Trenker 310
Burden, A. G. N. Improvements in Ore Feeders for Stamp Mills. (Patent abstract) 110
Burettes, Improved Taps for Prof. J. A. Wilkinson 114
——, Use of 357
Burgess, G. K. Some Exact Melting Points of Metals 209
——, W. T. Solubility of Calcium Carbonate and its Bearing on Water Purification 435
Burgstaller, A., and Rothmund, V. Rate of Decomposition of Ozone in Aqueous Solution 75
Burma Corporation. The Technical Committee of the 502
Bushell, B. D. Notes on Sinking Operations at the Springs Mines 101
Butters, C. Treatment of Refractory Gold and Silver Ores 317

C

Cadman, J. Notes on the Effect of Temperature in Mines in Great Britain 389
Caetani, G. Nomenclature in Ore Dressing 89
Cain and Tucker. Determination of Phosphorus in Steels containing Vanadium 10
Calcium Carbonate and its Bearing on Water Purification. Solubility of W. T. Burgess 435
—— Carbonate, Ill Effects of 97
—— and Magnesium in the same Solution. Titration of P. J. Fox 415

PAGE

Calcium, The Volumetric Determination of. T. Döring 415
Calculation of Extraction in Cyaniding H. A. Megraw 276
Caldecott, Dr. W. A., on Fine Screening .. 394
—— on Launder Grades for Sand Pulp.. 486
—— and others; Rand Metallurgical Practice (Review) 166
—— *and Powell, O. P. The Sand Filling of Mines* 119
Discussion 131, 264, 303, 368
Do. Reply to 409
Cam Curves, Method of Laying Out 156
—— Involute. Diagram of Inscribing Circle for 157
—— Lift and Drop, Velocity Curve for .. 156
Campbell and Dixon, Effect of Incombustible Dusts on the Explosion of Gases 281
Canada. Working Costs in 360
Cape Colony. New Fertiliser Factory in 362
Carat, The Metric 44
Carbide, Production of 153
Carbon Oxide, A New H. Meyer and K. Steiner 10
Carbonates, Alkaline, Removal of Fusions in, from the Crucible R. Howden 202
——, Temperatures of Decomposition of K. Friedrich 9
Carnotite 108
Carvalho, A. de, and Lepierre, C. Analysis of Minerals containing Phosphates and small Amounts of Uranium 347
Catalysis and Catalysers 154
Carcira Mine, Portugal, Cyaniding of Gold and Silver Ore at J. Hutton
Discussion 1
Cellier, Prof. J., on Electric Blasting 190
—— on Relation of Falls of Roof in Collieries to Weather Changes 400
Cement. Gripping Power of Portland H. Luftschitz 388
—— Injection for Consolidating Rocks M. de Marillac 333
Chalkley, G. A. Improvements in Supporting Rock Drills. (Patent abstract) 47
Chambers, A. L. On Mining Copper Ores at Messina 192
Chauvenet, Regis, Hydrometallurgy .. 350
Changes of Address: 48, 112, 168, 232, 288, 336, 364, 392, 420, 444, 468, 504
Charcoal in Gold-Bearing Cyanide Solutions with Reference to the Precipitation of Gold, The Effect of Morris Green 257
Discussion 341, 371
Do. Reply to 452
Chemical Abstracts 9, 73, 152, 200, 270, 309, 345, 382, 414, 434, 466, 499
—— Analysis of Aluminium J. Czochralski 312
—— ——, Methods of F. A. Gooch (Review) 109
—— Engineering, Elements of J. Grossmann (Review) 335
—— Fertilisers, Production and Consumption of 155
Chemical German F. C. Phillips (Review) 389
—— Industries and the Universities. The. M. T. Bogert 311
—— Reactions. The Role of Pressure in. J. Johnston 313
Chemistry of Gold, Contribution to the V. Lenher 152
——, Industrial, edited by A. Rogers (Review) 389
——, A New Era in H. C. Jones (Review) 390
Chenall, J. W. Chemical Extraction of Tin from the Ore 19
——, W. Patent Tin Extraction Process 278
Chilean *versus* Hardinge Mills R. Franke 216
—— Mills, Slow Speed E. J. Schrader 274
Chilton, J., on Electric Blasting 375
—— on Explosives in Hand and Machine Labour Stopes 243
—— on the Sand Filling of Mines 133
Chloride of Lime in Sanitation A. H. Hooker 285
Chlorides in Water. The Estimation of Traces of, M. Lombard 383
Chlorine in Natural Waters. The Determination of I. Tillmans and O. Heublein 414
Chlorplatinate Method for the Estimation of Potassium, Rapid Modified W. B. Hicks 271
Christian M. Disinfection and Disinfectants (Review) 230
Chromium and other Metals, Alloys of Cobalt with E. Haynes 15
—— in Presence of Iron, Volumetric Determination of A. Kurtenacker 203
——, Separation of Iron from A. Deshayes 203
Ciusa, R., and Terni, A. A Reaction of Iodine-ions 203
Clark, A. J., and Sharwood, W. J. Grinding Pans *versus* Tube Mills 95
Classification, Better, in Ore Dressing .. 314
——, *Hydraulic Classifiers and G. H. Stanley* 445
Discussion 452, 484
Classifier. Three and Twelve Spigot 314
Classifiers, Improvements in Means for Controlling the Underflow from O. Harrison and J. R. Pratt (Patent abstract) 47
Clay Purification by Electromosis 74
Clennell, J. E. Molybdenum in Cyanide Solutions 436
Clevenger, G. H. Rapid Silver Estimation in Mill Solutions 200
Clough, F. E. Blood Pressure Variations 228
Clover, A. M. Preservation of Hydrogen Peroxide by means of Acetanilide 414
Coade, M. E., and Werner, E. A. The Estimation of Nitrites by means of Thiocarbamide and the Interaction of Nitrous Acid and Thiocarbamide in the Presence of Acids of Different Strength 346
Coal Analysis. Accuracy and Limitations of A. C. Fieldner 313
—— Ash. Rare Metals in A. Jorissen 316

PAGE

Coal, The Composition of D. T. Jones and R. V. Wheeler 435
—— Cutters. Improvements in Mountings for Percussive W. Mauss. (Patent abstract) 110
——. Dangers of Dennstedt and Schafer 285
—— Deliveries, Sampling S. Pope .. 271
—— Distillation, &c. A Simplified Method for J. Schramm 346
—— for Gas Manufacture, South African J. W. Brown 274
——, Self Ignition of 313
——. Storage of Dennstedt and Schafer 468
——. Water in, Determination of P. L. Teed 152
—— from a Wood Steamer. Artificial. E. Heuser 334
Coals, A Test for Coking M. A. Pishel 208
Cobalt with Chromium and Other Metals. Alloys of E. Haynes 15
——. Mining Methods at 281
——. A New Use for 213
—— and Nickel Ores 329
—— and Uranium in Steel. Determination of H. Köning 414
Cobbe, H. N. G., on the Effect of Charcoal in Gold-bearing Cyanide Solutions 342
Coe, H. I. Influence of Sulphur on Cast Iron 95
Cohen, E., and Wootton, W. O. Mineralogy of the Rarer Metals (Review) 230
Coking Coals, A Test for M. A. Pishel 208
Cole. A. A. Mining Methods at Porcupine 164
Coleman, Dr. A. P. The Nickel Industry (Review) 335
Collieries. Relation of Falls of Roof in, to Weather Changes C. J. Gray 395
Discussion 400, 433
Colloid 83
Colloidal Solutions of Metals. Electrical Methods of Preparing H. Morris-Airey and S. H. Long 382
Colour Changes in Colloidal Gold S. H. Long 382
—— Reaction for the Detection of Thiosulphates E. Pozzi-Escot 73
Commercial Exhibits at the Third Annual Mining Exhibition 492
Compressed Air Leaks, Catching 503
Compressors, Air. Explosions in 234
Concentrate. Treatment of J. W. Hutchinson (Review) 110
Concentrates and Battery Chips, A Method of Assaying, for Gold and Platinum Metals A. F. Crosse 373
Discussion 374, 422, 483
——. Tin Ores and, Assay of E. A. Wraight and P. L. Teed 499
Concentrator. J. F. Isbell (Patent abstract) 442
——, Launder Data of the Washoe 320
Concrete Props and Beams in Mines, Reinforced Prof. S. M. Dixon .. 31
——, Strength of, Influence of Amount of of Water in 32
—— Structures, Waterproofing of.. 282
Conglomerates of the Witwatersrand R. Beck 105

PAGE

Congress, International Engineering 49
——. Sixth International, of Mining Metallurgy, Engineering and Practical Geology 393
Conklin. H. R. Handling Cyanide Precipitate at Lluvia de Oro 84
Contributions and Correspondence 200 486
Control of Noxious Fumes. The A. H. Martin 319
Conveyor Belt Idlers, Improvements in Bearings for A. E. Barker (Patent abstract) 110
—— —— Side Tip for W. J. Nicol .. 96
Copper. The Action of Sulphuric Acid on J. T. Cundall 500
—— Mining at Messina. Costs of 563
—— *Ores at Messina, Mining J. Allan Woodburn* 53
Discussion 192, 244, 306
Do. Reply to 401
——. Rare Metals in 313
—— Solutions, Electrolysis of J. H. Stansbie 270
Correspondence and Contributions 200 486
Corrosion of Metals, Electrolytic Methods for Preventing the W. W. Haldane Gee 94
Costs in Canada. Working 360
Council. Annual Report of 469
Craib. R., and Wilson. J. H. Improvements in Sand Pumps (Patent abstract) 391
Croasdale, S. Gold Recovery by Volatilisation 315
Crocker, W. J. Economic Efficiency in Mining 34
Crosse, Andrew F., on the Acidity or Alkalinity of Waters 118
—— —— on the Effect of Charcoal in Gold-Bearing Cyanide Solutions .. 371
—— —— on Fine Screening 394
—— —— *a method of Assaying Concentrates and Battery Chips for Gold and Platinum Metals* 373
Discussion 374, 422, 483
—— —— on Mining Exhibition .. 51, 421
—— —— on Referee Assays 367
—— —— on Seymour Memorial Library 290
—— —— on the Solubility of Iodine in Sodium Iodide Solutions 292
—— —— on Union Patents Bill 302
—— —— Witwatersrand Earth Tremors 426
Crown Mines, Ltd., Visit to 412 422
Crucibles, Deleterious Effect of Borax on .. 72
——, Magnesite, D. M. Liddell 278
Crushing Circuit, Flow of Ore in 322
——. Coarse, Milling by, in Cyanide Solution E. H. Simonds 27
——. Dry *versus* Wet, at Kalgoorlie. M. W. von Bernewitz 30
—— Efficiency, Mechanical 217
—— in Solution 17
—— -Surface Diagram A. O. Gates .. 91
—— Tests 28
—— in Water 17
Cullen, William, on Assaying Concentrates and Battery Chips 375
—— —— on Gold Production in Relation to Humanity 64

PAGE
Cullen, William on Mining Exhibition 367, 488
—— —— *Primer Clip for Explosives* 366
—— —— on Safety Measures in regard to the Use of Explosives (Lecture) 498
—— —— on Union Patents Bill 408
—— —— *Donaldson, T., and Waters, W. Electric Blasting* 174
Discussion 189, 248, 308, 375
Do. Reply to 431
Cumming, A. C., and Macleod, J. Estimation of Mercury as Metal by the Dry Method 203, 500
—— ——, and Finlay, T. M. Estimation of Zinc as Zinc Ammonium Phosphate and as Zinc Pyrophosphate 346
Cundall, J. T. The Action of Sulphuric Acid on Copper 500
Cupellation. Detecting Platinum Metals in Beads from C. O. Bannister and G. Patchin 435
Cuprous Chloride, Preparation of 9
Cyanamide in Agriculture E. J. Pranke 153
——, Manufacture of 154
Cyanide, Estimation of Free 354
—— Extraction at Champion Reef Mine 67
—— Lead N. M. Gupta 434
—— Poison, First Aid for W. H. Kritzer 43
—— Practice. M. W. von Bernewitz (Review) 390
—— —— *in India. H. M. Leslie. Discussion* 66
Reply to Discussion 66
—— Precipitate, Handling, at Lluvia de Oro. H. R. Conklin 84
—— Present in Solutions, Determination of the Total 123
—— Process of Gold Extraction J. Park (Review) 286
——, Regeneration of 161
—— ——, Assay for Silver by 316
—— Retreatment of Precipitate 79
—— *Solution, Analysis of a, from the Meyer and Charlton Mine H. E. Williams* 475
—— ——, Milling by Coarse Crushing in, E. H. Simonds 27
—— ——, Copper in 328
—— ——, Rate of Solution of Gold in, J. Gross 79
—— Solutions, The Dissolution of Gold and Silver in I. I. Andreev .. 383
—— ——, *The effect of Charcoal in Gold-Bearing, with Reference to the Precipitation of Gold Morris Green* 257
Discussion 341, 371
Do. Reply to 452
—— ——, Electrolysis of E. F. Kern 384
—— ——, Molybdenum in J. E. Clennell 436
—— ——, Precipitation of C. W. Merrill 97
—— ——, Retarding Effects on the Dissolving of Gold and Silver in Aqueous J. J. Andrejew 312
—— from Sugar Mill Residues C. A. Browne 202
PAGE
Cyanide Work, Acid Solutions in 78
—— —— The Use of Lime in A. M. Merton 321
Cyanides. Electrolysis of Aqueous Solutions of the Simple Alkaline, Clevenger V. Hall 416
Cyaniding, The Action of Oxidisers in M. Green. Reply to Discussion 65
——, Application of Desulphurising Process to Gold 326
——, Calculation of Extraction in H. A. Megraw 276
—— *of Gold and Silver Ore at Careira Mine J. Hutton. Discussion* 1
——, Grading Analyses and their Application to J. W. Bell 211
——, Grinding Ores for W. B. Easton 501
——, Intermittent. L. P. Hills 13
——, Iron in 327
——, Lead Salts in V. C. Bennett .. 213
——, Ores Amenable to H. A. Megraw 326
——, Silver, at Tonopah H. A. Megraw 16
——, Slime Agitation for H. A. Megraw 436
—— Slimy Ore by Continuous Decantation H. C. Parmelee 25
——, Solution Control in A. W. Allen 354
Czochralski, J. Chemical Analysis of Aluminium 312

D

Daly, W. Improvements in Stamp Guides (Patent abstract) 47
—— ——, Improvements in Stamp Stem Guides (Patent abstract) 47
Dams, Underground 339
Danger of Barium Peroxide as Part Cargo. P. V. Dupré 345
Davenport, C. Winding and Haulage Drums (Patent abstract) 419
Deacon, M., and Gore, W. Improvements in Apparatus for Filtering Water (Patent abstract) 442
De Beers. Visit to 366, 474
Decantation. Continuous, Cyaniding Slimy Ore by H. C. Parmelee 25
—— System, Intermittent, Flow Sheet .. 14
Decomposition of Carbonates, Temperatures of K. Friedrich 9
—— of Dolomite. Temperature of O. Kallauner 383
Dekker, N. H. M. Improvements in Electrolytes for Use in Electro-Metallurgy (Patent abstract) .. 110
Deloro, Ontario. Aluminium Precipitation at S. F. Kirkpatrick 316
Delport, T. D. Explosives in Hand and Machine Labour Stopes 241
Discussion 307
Dennstedt and Schafer. Dangers of Coal 285
—— —— Storage of Coal 468
Denne, A. B. Annual Report of the Chief Inspector of Explosives for the Union of South Africa for 1912 417
Denny, G. A. Compressed Air *versus* Hydraulic Power for Rock Drills 163
——, J. J. Desulphurising Silver Ores at Cobalt 323

PAGE

De Re Metallica, Georgius Agricola (Review) .. 45
Deshayes, A. Separation of Iron from Chromium .. 203
Desiccating Substance, Anhydrous Alumina as a .. 272
Desulphurizing Ore, Process of W. A. Hall (Patent abstract) .. 231
—— Process, The Hall Ore W. A. Hall 275
—— Silver Ores at Cobalt J. J. Denny 323
Determination of Gold in the Presence of Iridium and Allied Metals in Materials such as Black Sand J. Gray .. 2
Do. C. Toombs .. 4
Discussion .. 119
Do. Reply to .. 292
Detonators, Azides for. Neitzel .. 282
—— should Point, How .. 282
——, Use of Multiple .. 417
Dewar, T. N., on Relation of Falls of Roof in Collieries to Weather Changes 433
Dewatering Tank C. A. Banks .. 215
—— Wheel for Pulp R. B. Seeber .. 276
Diamonds, Weighing of, The Metric Carat and the .. 44
Dilatometer, Combination Specific Gravity Bottle and .. 272
Dimethylglyoxime, Applicability of M. Wunder and V. Thüringer .. 414
Dinner, The Annual .. 169, 266 290
Dips, South African Sheep .. 109
Disinfectants, Antiseptics and D. Sommerville .. 389
Disinfection and Disinfectants M. Christian (Review) .. 230
Dissolution of Gold and Silver in Cyanide Solutions, The I. I. Andreev 383
Dixon and Campbell. Effect of Incombustible Dusts on the Explosion of Gases .. 281
——, Professor S. M. Reinforced Concrete Props and Beams in Mines 31
Dodds, W. B., and Bowen, D. C. Improvements in Linings for the Ends of Tube Mills (Patent abstract) .. 364
Dolomite. Temperature of Decomposition of O. Kallauner .. 383
Donaldson, Sir F. Theory and Practice 39
——, T. On Union Patents Bill .. 302
—— ——, *Waters, W., and Cullen, W. Electric Blasting* .. 174
Discussion .. 189, 248, 308, 375
Do. Reply to .. 431
Doring, T. The Volumetric Determination of Calcium .. 415
Dorr Agitator .. 438
Drilling Averages in Different Rocks .. 164
——, Core, Steel Points for M. L. Martel .. 220
——, Machine, Efficiency in L. Balliet 220
——, Sprays in, Disadvantage of .. 281
Drills, Efficiency of Air .. 387
——, Sharpening of .. 387
——, Testing Rock .. 332
Drives, Drilling in .. 338
Drums, Winding and Haulage C. Davenport (Patent abstract) .. 419

PAGE

Drury, C. W., and Hall, E. J. Assay of Gold and Silver by Iron Nail Method .. 415
Dry *versus* Wet Crushing at Kalgoorlie. M. W. von Bernewitz .. 30
Dumps, Mine, Improvements in Treating. W. T. Anderson, E. H. Johnson, and P. R. Nethersole (Patent abstract) .. 442
——, Treating Tailings .. 415
Dunderdale, R. J. Paraffin as a Motor-Car Fuel .. 166
Dupré, F. H. and P. V. Constant Temperature Apparatus .. 272
——, P. V. Danger of Barium Peroxide as part Cargo .. 345
Dures, Robert Notes on the Assay of Mine Samples. Discussion 68, 173
Reply to Discussion .. 263
—— ——. *Notes on Waste Rock Values* .. 233
Dusserre, C. Dynamite in Agriculture 228
Dust in Air, Measurement of .. 98
——, Analyses of .. 99
—— by Electrical Precipitation. Separation of Blast Furnace W. A. Forbes .. 415
—— Losses in Dry Crushing .. 30
——, Measurement of .. 340
—— in Mine Air, Character of .. 99
—— Preventer, A G. J. L. Wannenburg (Patent abstract) .. 391
—— in Rises .. 338
—— *in Underground Workings, The Prevention of B. C. Gullachsen* 337
Discussion .. 433, 459, 483
Dustless Breakers .. 286
Dusts, Effect of Incombustible, on the Explosion of Gases, Dixon and Campbell .. 281
——, Inflammable Dr. R. V. Wheeler 39
Dynamite in Agriculture C. Dusserre 228

E

Earth Tremors, Witwatersrand H. E. Wood .. 423
Discussion .. 426, 466, 485
Easton, W. B. Grinding Ores for Cyaniding .. 501
Economic Efficiency in Mining W. J. Crocker .. 34
Eddying Resistance, Law of .. 87
Elasticity, Young's Modulus of .. 205
Electric Air Drill .. 163
—— *Blasting Wm. Cullen, T. Donaldson, and W. Waters* .. 174
Discussion .. 189, 248, 308, 375
Do. Reply to .. 431
—— Detonator Fuses .. 179
—— Discharge, an Active Modification of Nitrogen produced by the R. J. Strutt .. 152
—— and Fuse Blasting, Comparative Safety of .. 103
—— Mine Hoist, The Largest .. 279
—— *Parting Apparatus, Description of an F. Wartenweiler* .. 367

PAGE

Electric Conductivity of Minerals A. *Whitby* 170
Discussion 171
Do. Reply to 235
— Method of Preparing Aqueous Colloidal Solutions of Metals H. Morris-Airey and S. H. Long .. 382
— Precipitation. Separation of Blast Furnace Dust by W. A. Forbes 415
Electrodeposition of Zinc at High Current Densities. The J. N. Pring and U. C. Tainton 500
Electrolysis of Aqueous Solutions of the Simple Alkaline Cyanides Clevenger V. Hall 416
— of Copper Solutions J. H. Stansbie 270
— of Cyanide Solutions E. F. Kern 384
Electrolytes for Use in Electro-Metallurgy, Improvements in N. H. M. Dekker (Patent abstract) 110
Electrolytic Methods for Preventing the Corrosion of Metals W. W. Haldane Gee 94
Electrometallurgy of Aluminium Fedotieff and Ijinsky 11
Electrosmosis, Clay Purification by 74
Electro-osmotic Extraction of Water. Dr. G. B. Schwerin (Patent abstract) 231
Elsden, A. V., and Stansfield, J. F. Determination of Small Quantities of Lead 152
Elutriation Apparatus E. Schoene's .. 85
——, Grading Analyses by H. Stadler 85
Energy, Curves of 354
Engineering Congress, International 49
—— Schools 165
Engineers, Registration and Training of. G. H. Julius 42
Escot, E. Pozzi- Colour Reaction for the Detection of Thiosulphates .. 73
Evening Class Examination Results 289
Ewan, D., and Kaye, G. W. C. Sublimation of Metals at low Pressure 209
Exhibition, The Third Annual Mining 51, 169, 367, 394, 421, 471, 475, 486
Explosions in Air Compressors 284
Explosive Efficiency. Anderson 360
Explosives in Agriculture, The Use of .. 284
——, Economy in 35
—— *in Hand and Machine Labour Stopes. T. D. Delprat* 241
Discussion 307
——, International Commission for Unification of the Testing Methods of the Stability of 417
——, *Primer Clip for Wm. Cullen* .. 366
——, Safety Measures in regard to the use of (Lecture) W. Cullen .. 498
——, Sensitiveness of E. A. Mann .. 440
——, Suggestions to Lower Cost of 243
—— for the Union of South Africa for 1912. Annual Report of the Chief Inspector of A. B. Denne .. 417
Extraction, Calculation of 19
—— in Cyaniding, Calculation of H. A. Megraw 276

F

PAGE

"Falling Stream" Principle in Sampling 21
Falls of Ground 331
—— *of Roof in Collieries, Relation of, on the Middelbury Coalfield to Weather Changes C. J. Gray* .. 395
Discussion 400, 433
Fedotieff and Ijinsky. Electrometallurgy of Aluminium 11
Fennell, W. Improvements in Tube Mills (Patent abstract) 231
Ferric Salts, Reduction of, and their Titration with Pot. Permanganate A. Leclere 153
Ferro-Titanium, Analysis of A. R. Scott 274
Fertiliser Factory in Cape Colony, New 362
——, Nitrogenous 153
Fertilisers, Chemical, Production and Consumption of 155
Fichter, F., and Muller, E. Volumetric Determination of Tin with Potassium Bromate by the Method of H. Zschokke 10
—— ——, and Spengel, A. Reactions of Aluminium Nitride 155
Fieldner, A. C. Accuracy and Limitations of Coal Analysis 313
Figgis, W. E. Past and Present Metal Markets (Review) 151
Filling, Sand A. H. Martin 358
Filter Bags, Patterns for Making 84
—— Flask in Two Parts F. Mach .. 74
—— Frame Leaves 171
——, a Rotary Vacuum 214
Filters, Vacuum, at Kalgoorlie 31
Filtration, Vacuum, at the Victoria Mill, Waihi W. Macdonald
Discussion 67, 171
Do. Reply to 430
Fink, C. G. Applications of Ductile Tungsten 205
Finlay, J. R. Principles of Mine Valuation 36
——, T. M., and Cumming, A. C. Estimation of Zinc as Zinc Ammonium Phosphate and as Zinc Pyrophosphate 346
Fire Protection 280
Fires, Extinguishing, with Sawdust 284
Firing (Cheesa) Sticks, Nitrous Fumes in, Note on the Quantitative Determination of Dr. L. Heymann.
Reply to Discussion 118
Correspondence 200
First Aid for Cyanide Poison W. H. Kritzer 43
Fixation of Nitrogen Progress in Scandinavia 346
Flow Sheet, Ogle Mountain Mill 76
Fletcher, A. L. Determination of Radium in Radio-active Substances 383
Fluids, Metering and Recording the Flow of J. W. Ledoux 223
Fluxes, Effect of Change in 71
Foersterling, H. Solvent for Precious Metals 383
Forbes, W. A. Separation of Blast Furnace Dust by Electrical Precipitation 415

PAGE

Formula for Extraction Percentages .. 277
Fox, P. J. Titration of Calcium and Magnesium in the same Solution .. 415
Franke, R. Hardinge *versus* Chilean Mills 216
Frauds, Fakirs and Fables 350
Free Milling Ore, Treatment of 349
Freiberg Mines to Close 219
Friedrich, K.—Temperatures of Decomposition of Carbonates 9
Friend, J. N. Preservation of Iron .. 204
Fuel, Liquid V. B. Lewes 284
Fuller, J. P. (Obituary) 73
Fumes, The Control of Noxious A. H. Martin 319
Furnaces for Decomposing Pyrites, Improvements in W. A. Hall (Patent abstract) 231
Fuse and Electric Blasting, Comparative Safety of 103
Fuses, Electric Detonator and Delay Action 179
Fusions, Removal of, in Alkaline Carbonates from the Crucible R. Howden 202

G

Gas Manufacture, South African Coal for, J. W. Brown 274
Gascoyne, R. Stamps *versus* Tube Mills 416
Gases, Explosion of, Effect of Incombustible Dusts on the Dixon and Campbell 281
—— from Springs, Helium in C. Moureu and A. Lepape 10
Gassing from Benzole Fumes 360
Gastaldi, E., and Pertusi, C. Detection of Hydrocyanic Acid 152, 203, 347
Gates, A. O. The Crushing Surface Diagram 91
—— —— Efficiency of Hardinge Mill .. 352
Gee, W. W. Haldane Electrolytic Methods for Preventing the Corrosion of Metals 94
Geology, Text Book of Mining J. Park (Review) 46
Geothermic Gradients 388
Gibson, J.—On Ventilation of the Mines of the Rand 244
Gill, H. W. The Solubility of Iodine in Sodium Iodide Solution 290
Discussion 292, 343, 378
Do. Reply to 476
Gitsham Process, The 78
Gneiss, The Fundamental 192
Gold, The Annealing of T. K. Rose .. 276
—— *Bearing Cyanide Solutions, The Effect of Charcoal in, with Reference to the Precipitation of Gold Morris Green* 257
Discussion 341, 371
Do. Reply to 452
—— —— Scheelite in New Zealand, Treatment of C. W. Gudgeon .. 436
—— Bullion, Silver in 69
——, Chemistry of V. Lenher 152
——, Coarse, in Sampling and Assaying F. White 80
Gold, Colour Changes in Colloidal S. H. Long 382
—— in Cyanide Solutions, Rate of Solution of J. Gross 79
——, Delicate Reactions for 9
—— Mine, The Ocean a 351
—— Mining Methods, Indian H. H. Baker 279
—— Ore Treatment, Simplification of A. W. Allen 348
—— —— Platinum Metals Assay, Bibliography 4
—— ——, Rhodium and Iridium, Separation of Palladium from the Noble Metals M. Wunder and V. Thüringer 414
—— *and Platinum Metals, a Method of Assaying Concentrates and Battery Chips for A. F. Crosse* .. 373
Discussion 374, 422, 483
——, *Precipitation of, The Effect of Charcoal in Gold-Bearing Cyanide Solutions with Reference to the Morris Green* 257
Discussion 341, 371
Do. Reply to 452
——, Precipitation of, by Manganous Salts A. D. Brokaw 201
—— *in the Presence of Iridium and Allied Metals in Materials such as Black Sand, Determination of J. Gray* 2
C. Toombs 4
Discussion 119
Do. Reply to 292
—— *Production in Relation to Humanity E. M. Weston. Discussion* 64, 136
Reply to Discussion 252
—— Recovery by Volatilisation S. Croasdale 315
—— Reserves, State 166
—— and Silver in Aqueous Cyanide Solutions, Retarding Effects on the Dissolving of J. J. Andrejew 312
—— ——, Assay of, by Iron Nail Method E. J. Hall and C. W. Drury 415
—— —— in Cyanide Solutions, The Dissolution of I. I. Andreev .. 38[illegible]
—— —— *Ore, Cyaniding of, at Caveira Mine J. Hutton Discussion* 1
—— —— Ores, The Treatment of Refractory C. Butters 317
—— —— Zinc Dust Precipitation of A. M. Merton 317
——, Solubility of, in Impure Solutions 78
—— Telluride Ores of, Concentration of H. E. Wood 205
—— by Volatilisation, Extracting, B. Howe 14
—— Volatilisation Process 208
—— The Volumetric Determination of V. Lenher 309
Gooch, F. A. Methods of Chemical Analysis (Review) 109
Grading Analyses and their Application to Cyaniding J. W. Bell 211
—— —— by Elutriation H. Stadler 85
Graphite Deposits of Madagascar and Africa, Some Observations on the E. R. Barden. Reply to Discussion 1

PAGE

Graphite Industry of Madagascar 166
—— in Madagascar 286
Gray, C. J. Relation of Falls of Roof in Collieries on the Middelburg Coalfield to Weather Changes 395
Discussion 400, 436
——, Jas., on Assay of Mine Samples .. 174
—— —— *Determination of Gold in the Presence of Iridium and Allied Metals in Materials such as Black Sand* 2
Discussion 119
Do. Reply to 292
Green, Morris. The Action of Oxidisers in Cyaniding. Reply to Discussion .. 65
—— ——, *The Effect of Charcoal in Gold-Bearing Cyanide Solutions with Reference to the Precipitation of Gold* 257
Discussion 341, 371
Do. Reply to 452
Greenwell, A., and Elsden, J. V. Practical Stone Quarrying (Review) 166
Grinding, Fine, Necessity of 324
—— Ores for Cyaniding W. B. Easton 501
—— Pans *versus* Tube Mills A. J. Clark and W. J. Sharwood 95
Gripping Power of Portland Cement H. Luftschitz 388
Gross, J. Precipitation Temperatures 78
—— Rate of Solution of Gold in Cyanide Solution 79
Grossmann, J. Elements of Chemical Engineering (Review) 335
Gudgeon, C. W. Treatment of Gold-Bearing Scheelite in New Zealand 436
Gullachsen, B. C. The Prevention of Dust in Underground Workings .. 337
Discussion 433, 459, 483
—— —— on a Primer Clip for Explosives 366
Gulliver, G. H. Metallic Alloys (Review) 335
Gupta, N. M. Lead Cyanide 434
Guy, A. H. An Improvement in Bearings for Shaft Rollers (Patent abstract) 230
—— —— Shaft Rollers, An Improvement in (Patent abstract) 110

H

Haber, F. Methane Indicators 503
Hall, Clevenger V., Electrolysis of Aqueous Solutions of the Simple Alkaline Cyanides 416
——, E. J. and Drury, C. W. Assay of Gold and Silver by Iron Nail Method 115
—— W. A. Furnaces for Decomposing Pyrites, Improvements in, (Patent abstract) 231
—— ——, Process of Desulphurizing Ore, (Patent abstract) .. 231, 275
Hamilton, E. M. Aluminium Precipitation at Nipissing 157
Hansen, H. A., and Schmueser, J. M., Improvements in Linings for Ball or Pebble Mills (Patent abstract) 392

PAGE

Hardach, E. E., on Sand-Filling of Mines 368
Hardinge *versus* Chilean Mills R. Franke 216
—— Mill, Efficiency of A. O. Gates .. 352
Harrison, O, and Pratt, J. R. Improvements in Means for Controlling the Underflow from Classifiers (Patent abstract) 47
Haynes, E. Alloys of Cobalt with Chromium and other Metals 15
Health Conditions on Rand 330
Heating Mill Solutions 11
—— Solutions 18
Helium in Gases from Springs C. Moureu and A. Lepape 10
Hentschel, G. Hydraulic Lime 10
Heublein, O., and Tillmans, J. The Determination of Chlorine in Natural Waters 414
Heuser, E. Artificial Coal from a Wood Steamer 334
Heymann, Dr. L. Note on the Quantitative Determination of Nitrous Fumes in Firing (Cheesa) Sticks
Reply to Discussion 118
Correspondence 200
Hicks, W. B. Rapid Modified Chlorplatinate Method for the Estimation of Potassium 271
Hill, G. T. and Stone, C. G. Improvements in Manufacture of Portland Cement (Patent abstract) 364
Hills, L. P. Intermittent Cyaniding 13
Hoist, Electric Mine, The Largest 279
Holes, The Safe Handling of Missed .. 386
Holman, J. M., and J. L. Improvements in Ore Concentrating Tables (Patent abstract) 419
Hooker, A. H. Chloride of Lime in Sanitation 285
Horwood, C. B. Iridosmine from the New Rietfontein Mines 360
Howden, R. Removal of Fusions in Alkaline Carbonates from the Crucible 202
Hewe, B. Extracting Gold by Volatilisation 14
—— —— The Howe Volatilisation Process 385
Hurter, C. S. Method of Blasting in the Lake Superior Iron District 103
Hutchin, H. W. Assay of Tin Ores .. 499
Hutchinson, J. W. Treatment of Concentrate at the Goldfield Consolidated Mill (Review) 110
Hutton, J. Cyaniding of Gold and Silver Ore at Carcira Mine. Discussion 1
Hydraulic Classification, Factors in 87
—— *Classifiers and Classification, Notes on G. H. Stanley* 445
Discussion 452, 484
—— Concentration, Improvements in L. D. Mills (Patent abstract) 231
—— Lime G. Hentschel 10
—— Power *versus* Compressed Air for Rock Drills G. A. Denny .. 163
Hydrazine Hydrochloride as a Reduction Reagent for Gold 311

PAGE

Hydrocyanic Acid 357
—— —— in Air 123
—— ——, Detection of C. Pertusi and E. Gastaldi 152, 203, 347
Hydrogen Peroxide and Ozone, Determination of 75
—— ——, Preservation of, by means of Acetanilide A. M. Clover 414
—— Sulphide in Natural Water. Determination of small Amounts of. W. Winkler 414
Hydrometallurgy Regis Chauvenet 350

Ignition of Coal, Self 313
—— of Wood 389
Iljinsky and Fedotieff Electrometallurgy of Aluminium 11
Illumination Underground 387
Improvement in Blasting. Important. W. Macnab 441
Inaugural Address *A. Richardson (President)* 51
India, Cyanide Practice in *H. M. Leslie. Discussion* 66
Reply to Discussion 66
Indian Gold Mining Methods H. H. Baker 279
Indicators, Methane F. Haber 503
——, *A Study in. The Determination of the Acidity or Alkalinity of Waters* *Dr. J. Moir* 114
Discussion 118, 294
Do. *Reply to* 404
Industries and the Universities. The Chemical M. T. Bogert 311
Inflammable Dusts Dr. R. V. Wheeler 39
International Commission for Unification of the Testing Methods of the Stability of Explosives 417
—— Congress of Mining 334
Invention, Definition of 297, 380
Iodine, The Solubility of, in Sodium Iodide Solution *H. W. Gill* 290
Discussion 292, 343, 378
Do. *Reply to* 476
—— -ions, a Reaction of R. Cusa and A. Terni 203
Ipomœa-Root, a Rhodesian P. H. Marsden 75
Iridium and Allied Metals, Determination of Gold in the Presence of, in Materials such as Black Sand
J. Gray 2
C. Toombs 4
Discussion 119
Do. *Reply to* 292
—— Prize Papers 50
——, Separation of Palladium from the Noble Metals Gold, Platinum, Rhodium, and M. Wunder and V. Thüringer 414
Iridosmine from the New Rietfontein Mines C. B. Horwood 360
Iron, Analyses of 274
——, Cast, Influence of Sulphur on H. I. Coe 95
—— from Chromium, Separation of A. Deshayes 203
Iron, Coating, with Aluminium 276
—— in Cyaniding 327
—— Nail Method, Assaying by W. J. Sharwood 345
—— —— —— Assay of Gold and Silver by E. J. Hall and C. W. Drury 415
——, Preservation of J. N. Friend 204
——, Rusting of, a New Chemical Cause of Vaubel 271
—— and Steel, Corrosion of W. H. Walker 204
——, Volumetric Determination of Chromium in Presence of A. Kurtenacker 203
Irvin, D. F., Zinc Dust Precipitation Equipment 206
Irvine, Dr. L. G. Ambulance and Rescue Appliances (Lecture) 497
Irwin, T. K. Improved Apparatus for the Aeration and gasification of Liquids (Patent abstract) 419
Isbell, J. F. Concentrator (Patent abstract) 442

J

James, A. Improvements in Apparatus for Agitating or Collecting Ore Pulp (Patent abstract) 47
John W. E., and Spoor, A. L. *The Union Patents Bill* 296
Discussion 302, 379, 408, 428, 465
Johnson, E. H., Anderson, W. T., and Nethersole, P. R. Improvements in Treating Mine Dumps, &c. (Patent abstract) 442
——, and Winterton, A. E. Ore Separating Means (Patent abstract) 230
Johnston, A. McA. on the Acidity or Alkalinity of Waters 474
—— —— on Election of Auditors 474
—— —— *Fine Screening, and Discussion* 394
—— —— on Inauguration of President 485
—— —— on the Sand Filling of Mines 303
—— J. The Role of Pressure in Chemical Reactions 313
Johnstone, S. J. Titanium and its Uses 347
Jones, D. T., and Wheeler, R. V. The Composition of Coal 435
—— H. C. A new Era in Chemistry (Review) 390
Jorissen, A. Rare Metals in Coal Ash 316
Journals and Papers 1913-1914 470
Julius, G. A. Registration and Training of Engineers 42

K

Kallauner, O. Temperature of Decomposition of Dolomite 383
Kaye, G. W. C., and Ewan, D. Sublimation of Metals at Low Pressure 209
Kedesdy, E. Detection of Molybdenum 414
Kern, E. F. Electrolysis of Cyanide Solutions 384
Kick's Law 91
Kirkpatrick, S. F. Aluminium Precipitation at Deloro, Ont. 316

PAGE
Köning, H. Determination of Cobalt and Uranium in Steel .. 414
Kotze, R. N. Witwatersrand Mining Conditions .. 330
—— At Mining Exhibition Lecture .. 496
Kritzer, W. H. First Aid for Cyanide Poison .. 43
Krupko and Wöhler Sensitiveness to Light of Azides .. 270
Kurtenacker, A. Volumetric Determination of Chromium in Presence of Iron .. 203

L

Laboratory Work. Care in G. Surr 382
Lang, H. Mill-Sites .. 212
Launder Data of the Washoe Concentrator .. 320, 321
—— Grades for Sand Pulp Dr. W. A. Caldecott .. 486
Launders and Barriers in Sand Filling .. 128
Launay, M. de The formation of Metalliferous Deposits .. 161
Lavers, H. Effect of Tungsten on the Ammonium-Molybdate Assay of Lead .. 466
Lead, Ammonium-Molybdate Assay of, The Effect of Tungsten on the H. Lavers .. 466
—— Antimony Alloys and Antifriction Metals. Determination of Antimony and Arsenic in L. Bertiaux .. 500
—— Cyanide N. M. Gupta .. 434
——, Determination of .. 210
—— —— of small Quantities of A. V. Elsden and J. F. Stansfield .. 152
—— Salts in Cyaniding V. C. Bennett .. 213
—— and Zinc in Ores .. 328
Leblanc Process .. 239
Leclére, A. Detection of Nitrous Acid .. 346
—— —— New Method for Reduction of Ferric Salts, and their Titration with Potassium Permanganate .. 153
Lectures at the Third Annual Mining Exhibition:—
Accidents on the Witwatersrand A. M. Archbold .. 406
Ambulance and Rescue Appliances Dr. L. G. Irvine .. 497
Safety Measures in Regard to the Use of Explosives W. Cullen 498
Ledoux, J. W. Metering and Recording the Flow of Fluids .. 223
Lehfeldt, Dr. R. A. On Gold Production in Relation to Humanity .. 136
Lenher, V. Contribution to the Chemistry of Gold .. 152
—— The Volumetric Determination of Gold .. 309
Lepape, A., and Moureu, C. Helium in Gases from Springs .. 10
Lepierre, C., and Carvalho, A. de Analysis of Minerals containing Phosphates and small amounts of Uranium .. 347

PAGE
Leslie, H. M. Cyanide Practice in India. Discussion .. 66
Do. Reply to .. 66
Lewes, V. B. Liquid Fuel .. 284
Lewis, J. V. Determinative Mineralogy (Review) .. 166
Liberty Bell G.M. Co. Heating Mill Solutions at .. 11
Library, Seymour Memorial .. 290
Liddell, D. M. Magnesite Crucibles .. 278
"Life" Factor in Mine Valuations .. 36
Lime, Characteristics of .. 323
——, Chloride of, in Sanitation A. H. Hooker .. 285
—— in Cyanide Work, The Use of A. M. Merton .. 321
——, Determination of Available .. 323
——, Hydraulic G. Hentschel .. 10
——, Percentage of, in Milk of Lime .. 323
——, Solubility of, in Water .. 323
——, Storage of .. 323
—— in Sugar, Solubility of .. 323
Lindgren, W. Mineral Deposits (Review) .. 389
Lining for Tanks Acid Proof .. 388
Liquid Fuel V. B. Lewes .. 284
Liquids, Improved Apparatus for the Aeration and Gasification of T. K. Irwin (Patent abstract) .. 419
Litharge, Silver in .. 69
Lombard, M. The Estimation of Traces of Chlorides in Water .. 383
Long, S. H. Colour Changes in Colloidal Gold .. 382
—— —— and Morris-Airey, H. Electrical Method of Preparing Aqueous Colloidal Solutions of Metals .. 382
Luftschitz, H. Gripping Power of Portland Cement .. 388

M

MacArthur, J. S. A New Method of Precipitation by Zinc. Discussion 1
Reply to Discussion .. 265
Macdonald, W. Vacuum Filtration at the Victoria Mill, Waihi. Discussion .. 67, 171
Do. Reply to .. 430
Mach, F. Filter Flask in Two Parts 74
Machine Drilling, Efficiency in L. Balliet .. 220
—— Drills .. 332
Mackay, H. S. Process for Extracting Metals from Low Percentage Ores (Patent abstract) .. 419
Maclachlan, M. W. A System of Keeping Mine and Mill Accounts, Costs, and Metallurgical Records .. 138
Macleod, J., and Cumming, A. C. Estimation of Mercury as Metal by the Dry Method .. 203, 500
MacNab, W. Important Improvement in Blasting .. 441
Madagascar and Africa, Graphite Deposits of, Some Observations on the E. R. Bawden. Reply to Discussion 1
——, Graphite in .. 286

PAGE

Magadi Soda Deposits 236

Magnesite Crucibles D. M. Liddell .. 278

Magnesium as Ammonium Magnesium Phosphate and Ammonium Magnesium Arsenate. Determination of M. Wunder and C. Schuller 347

—— in the same Solution. Titration of Calcium and P. J. Fox.. 415

Management. Mistakes made in Mine. L. Balliet 387

Mann. E. A. Sensitiveness of Explosives 440

Manganese. Analyses of 274

—— Ores 329

Manganous Salts, Precipitation of Gold by A. D. Brokaw 201

Mantles for Incandescent Gas Lighting C. R. Böhn 44

Marillac. M. de Cement Injection for Consolidating Rocks 333

Marsden. P. H. A Rhodesian Ipomœa-Root 75

Martel. M. L. Steel Points for Core Drilling.. 220

Martin, A. H. The Control of Noxious Fumes 319

—— —— Sand Filling 358

Martyn. T. G. On the Assay of Mine Samples 70

—— —— on Hydraulic Classifiers and Classification 484

Material on Ordinary Scales. Weighing Light 334

Mauss. W. Improvement in Mountings for Percussive Coal Cutters (Patent abstract) 110

McCrae, J. On the Solubility of Iodine in Sodium Iodide Solution 343

McNeill. Bedford On the Effect of Charcoal in Gold-Bearing Cyanide Solutions 343

Mechanical Stirring of Pulp 437

Megraw. H. A. Calculation of Extraction in Cyaniding 276

—— —— Ores Amenable to Cyaniding .. 326

—— —— Silver Cyaniding at Tonopah.. 16

—— —— Slime Agitation for Cyaniding 436

Meissner. W. Comparison of Redwood's with Engler's Viscosimeter 201

Melting Points of Commercial Brasses and Bronzes 502

—— —— of Metals. Some Exact G. K. Burgess 209

Members. New 49. 113, 169, 233. 289. 365. 394. 421. 445. 474

Membership and Papers 411

Mennell F. P. A Manual of Petrology (Review) 109

Mercury. Aluminium Corroded by 346

—— Destructive of Zinc 328

—— as Metal. Estimation of, by the Dry Method A. C. Cumming and J. Macleod 203, 500

Merrill. C. W. Alloy of Zinc as Precipitant 205

—— —— Precipitation of Cyanide Solutions 97

Merrill. C. W. Process for Precipitating and Recovering Metal Bearing Materials from Solutions, &c. (Patent abstract) 391

Merton, A. M. Selection and Operation of Tube Mills 23

—— —— Specifications and Tests for Zinc Dust 209

—— —— The Use of Lime in Cyanide Work 321

—— —— Zinc Gold Precipitation of Gold and Silver 317

Messina. Mining Copper Ores at J. Allan Woodburn 53

Discussion 192, 244, 306

Do. Reply to 401

Metal Bearing Materials, Process for Precipitating and Recovering from Solutions, &c. C. W. Merrill (Patent abstract) 391

—— Markets. Past and Present W. E. Figgis (Review) 151

—— from Solutions, Improvements in Processes for Precipitating R. L. L. Towne and C. Robinson (Patent abstract) 419

Metallic Alloys G. H. Gulliver (Review) 335

Metalliferous Deposits. The Formation of. M. de Launay 161

Metallurgical Abstracts 11. 75. 155. 204. 274. 314. 347, 384, 415. 436. 467. 501

—— *Records, a System of Keeping Mine and Mill Accounts, Costs and Metallurgical Records M. W. Maclachlan* 138

Metallurgy. American. a German Criticism of 205

—— Progress in 439

Metals in Coal Ash, Rare A. Jorissen 316

—— Corrosion of, Electrolytic Methods for Preventing the W. W. Haldane Gee 94

——. Electrical Method of Preparing Aqueous Colloidal Solutions of. H. Morris Airey and S. H. Long 382

——. Light. Electrolytic Manufacture of. &c. &c. E. A. Ashcroft (Patent abstract) 391

—— from Low Percentage Ores, Process for Extracting H. S. Mackay (Patent abstract) 419

——. Method of Producing J. E. Bucher (Patent abstract) 391

—— from their Ores, Improved Process for Separating W. H. Aghan (Patent abstract) 442

——. Rare, in Copper 313

——. Solvent for Precious H. Foersterling 383

——. Some Exact Melting Points of G. K. Burgess 209

——. Sublimation of, at low Pressure .. G. W. C. Kaye and D. Ewan 209

Meter Registers. Venturi 223

Methane Indicators F. Haber 503

Methylene Blue. Determination of Titanium by Titration with B. Neumann and R. K. Murphy .. 435

PAGE
Metric Carat and the Weighing of Diamonds 44
Meyer and Charlton Gold Mine, Analyses of a Cyanide Solution from the H. E. Williams 475
—— ——, Electric Blasting at the 178
Meyer, H., on Assaying Concentrates and Battery Chips 375
—— H., and Steiner, K. A New Carbon Oxide 10
Middelburg Coalfield, Relation of Falls of Roof in Collieries on the, to Weather Changes C. J. Gray 395
Discussion 400, 433
Mill Circuit at the Alaska Treadwell Mill. Tube. 320
——, Conical, Advantages of 216
——, Efficiency of Hardinge A. O. Gates 352
—— Practice, Tube F. C. Brown .. 467
—— Sites H. Lang 212
—— Solutions, Heating 11
—— ——, Rapid Silver Estimation in G. H. Clevenger 200
——, Stamp, Curves and Speeds S. L. Berry 155
—— ——, The Pinder-Berry 89
——, Use of the Tube 349
Milling by Coarse Crushing in Cyanide Solution E. H. Simonds .. 27
—— Costs, Hollinger 361
Mills or Apparatus, Improvements in R. Rodger (Patent abstract) 110
——, Hardinge versus Chilean R. Franke 216
——, L. D. Improvements in Hydraulic Concentration (Patent abstract) 231
—— —— Improvements in Precipitating Apparatus (Patent abstract) .. 47
——, Slow-speed Chilean E. J. Schrader 274
——, Tube *versus* Stamps R. Gascoyne 416
Mine Accidents on the Witwatersrand (Lecture) A. M. Archbold .. 496
—— Management, Mistakes made in, L. Balliet 387 388
—— Manager, Relation of the, to Recent Graduates ....R. Peale 165
—— *and Mill Accounts, Costs and Metallurgical Records, A System of Keeping M. W. Maclachlan* 138
—— *Samples, Notes on the Assay of, Robert Dures. Discussion* 68, 173
Reply to Discussion 263
—— Sanitation W. H. Moulton.. .. 333
—— Supplies, Economy in 34
—— Valuation, Principles of J. R. Finlay 36
—— Valuations, Life Factor in 36
Mineral Deposits W. Lindgren (Review) 389
Mineralogy, Determinative J. V. Lewis (Review) 166
—— of the Rarer Metals E. Cohen and W. O. Wootton (Review) 230
Minerals containing Phosphates and small Amounts of Uranium, Analyses of, C. Lepierre and A. de Carvalho .. 347
——, *Electrical Conductivity of I. Whitby* 170
Discussion 171
Do. Reply to 235
——, Process of Treating, &c. F. C. W. Timm (Patent abstract) 420
Miners and Metallurgists, Pocket Book for, F. D. Power (Review) 381
—— Phthisis Prevention Committee, Interim Report of 98
Mines in Great Britain, Notes on the Effect of Temperature in J. Cadman 388
—— *of the Rand, Ventilation of the G. H. Blenkinsop* 148
Discussion 151, 195, 244, 306
Do. Reply to 402
——, *The Sand Filling of W. A. Caldecott and O. P. Powell* 119
Discussion 131, 264, 303, 368
Do. Reply to 409
—— Wireless Telephony in 333
Mining Abstracts 31, 98, 161, 219, 279, 330, 358, 386, 417, 440, 592
—— Conditions on the Continent 282
—— —— Witwatersrand R. N. Kotze 330
—— *Copper Ores at Messina J. Allan Woodburn* 53
Discussion 192, 244 306
Do. Reply to 401
—— Costs Hollinger 360
——, Economic Efficiency in W. J. Crocker 34
—— International Congress of 334
——, Metallurgy, Engineering and Practical Geology, Sixth International Congress of 393
—— Methods at Cobalt 281
—— ——, Indian Gold H. H. Baker 279
—— —— at Porcupine A. A. Cole 164
—— School Training 165
——, Selective E. J. Renaud 280
—— ——, on the Witwatersrand 104
Miscellaneous Abstracts 39, 107, 165, 223, 282, 333, 361, 388, 441, 468
Missfires 332, 386
Molybdenum in Cyanide Solutions J. E. Clennell 436
—— Detection of E. Kedesdy 414
Moir, Dr. J. The Determination of the Acidity or Alkalinity of Waters: A Study in Indicators 114
Discussion 118, 294
Do. Reply to 404
—— —— on Nitrous Fumes from Firing Sticks 200
—— —— on the Solubility of Iodine in Sodium Iodide Solutions.. 292, 378
—— —— on Ventilation of the Mines of the Rand 151, 246
Moisture in Tube Mill Feed 25, 468
Molasses, Use of, for Reducing Dust 340
Morris Airey, H., and Long, S. H. Electrical Method of Preparing Aqueous Colloidal Solutions of Metals 382
Mortar Boxes, Improvements in T. Rennert (Patent abstract) 47
Moulton, W. H. Mine Sanitation .. 333
Moureu, C., and Lepape, A. Helium in Gases from Springs 10
Moynihan, E. J. on Electric Blasting 249
—— —— *Rock Temperatures* 455

PAGE

Moynihan, E. J. on Union Patents Bill 465
—— —— on Ventilation of the Mines of the Rand 306
—— —— on Witwatersrand Earth Tremors 426
Muller, E., and Fichter, F. Volumetric Determination of Tin with Potassium Bromate by the Method of H. Zschokke 10
Multiple Detonators, Use of 417
Murphy, R. K., and Neumann, B. Determination of Titanium by Titration with Methylene Blue 435

N

Natal Wattle Bark Industry 285
Neill, J. M. Recovery of Black Sand and Floating Particles of Metallic Minerals. Discussion 72
Reply to Discussion 133
—— —— on Vacuum Filtration at the Victoria Mill 67
Neitzel Azides for Detonators 282
Neuhaus, H., at Mining Exhibition Lecture 498
Neumann, B., and Murphy, R. K. Determination of Titanium by Titration with Methylene Blue 435
New Zealand. Treatment of Gold Bearing Scheelite in C. W. Gudgeon 436
Newhouse, R. C. Improvements in Roll Crushers (Patent abstract) 231
Nickel and Cobalt Ores 329
——, The Detection and Estimation of, by means of *a*—Benzildioxime, F. W. Atack 345
—— Industry, The Dr. A. P. Coleman (Review) 335
Nicol, W. J. Side Tip for Conveyor Belt 96
Nipissing, Aluminium Precipitation at E. M. Hamilton 157
Nitrates in Water, The Detection of small quantities of S. Rothenfusser 415
——, The Estimation of, by means of Thiocarbamide and the Interaction of Nitrous Acid and Thiocarbamide in the Presence of Acids of Different Strength M. E. Coade and E. A. Werner 346
Nitrogen from Air, Extraction of E. M. Wiley 272
——, Fixation of—Progress in Scandinavia 346
—— Produced by the Electric Discharge, R. J. Strutt 152
Nitrous Acid, Detection of Leclére 346
—— *Fumes in Firing (Cheesa) Sticks, Note on the Quantitative Determination of Dr. L. Heymann. Reply to Discussion* 118
Correspondence 200
—— Gases, Process for the Absorption of 272
Nomenclature in Ore Dressing G. Caetani 83
——, System of 348
Nordrok, H., and Prelle, G. Improvements in Ore Feeders (Patent) abstract) 110
Notices and Abstracts of Articles and Papers 9, 73, 152, 200, 270, 309, 345, 382, 414, 434, 466, 499
Noxious Fumes, The Control of A. H. Martin 319

O

Obituary Dr. Johnstone Brown 344
—— J. P. Fuller 73
—— Wilfrid Taylor 344
Ocean a Gold Mine, The 351
Oliver, J. Improvements in Tube Mill Liners (Patent abstract) 504
Ore Concentrating Tables, Improvements in J. M. and J. L. Holman (Patent abstract) 419
—— Dressing Improvements R. H. Richards 314
—— ——, Nomenclature in G. Caetini 83
—— Feeders, Improvements in Nordrok and Prelle (Patent abstract) 110
—— —— for Stamp Mills, Improvements in A. G. N. Burden (Patent abstract) 110
——, Free Milling, Treatment of 349
—— Passes and Ore Bins 340
—— Pulp, Improvements in Apparatus for Agitating or Collecting A. James (Patent abstract) 47
—— Pulverising Machines, Improvements in M. Arnold (Patent abstract) 110
—— Sampling, Principles of A. van Zwaluwenburg 19
—— Separating Means E. H. Johnson and A. E. Winterton (Patent abstract) 230
——, Slimy, Cyaniding, by Continuous Decantation H. C. Parmelee 25
—— Treatment, Simplification of Gold, A. W. Allen 348
Ores Amenable to Cyaniding H. A. Megraw 326
——, Apparatus for the Treatment of, L. C. Trent (Patent abstract) .. 391
——, —— —— of, Improvements in, &c. A Argles (Patent abstract) 391
—— for Cyaniding, Grinding W. B. Easton 501
——, Treatment of Refractory Gold and Silver C. Butters 317
Osborn, S. S. Improvements in Pulverizers (Patent abstract) 231
Oxalic Acid, Standard 323
Oxide A New Carbon H. Meyer and K. Steiner 10
Oxidisers in Cyaniding, The Action of M. Green. Reply to Discussion .. 65
Ozone, Rate of Decomposition of, in Aqueous Solution V. Rothmund and A. Burgstaller 75

P

Packs 331
——, Mine, Improved Pole for A. and J. Whitby (Patent abstract) .. 504

PAGE
Palladium from the Noble Metals Gold, Platinum, Rhodium, and Iridium, Separation of M. Wunder and V. Thüringer 414
Palm Oil Industry, The African 166
Pam, Edgar, on Electric Blasting 189
—— on Rock Temperatures 459
—— on the Sand Filling of Mines 131
—— on Ventilation of the Mines of the Rand 151, 248
Papers and Membership 411
——, Personal Allusions in 394
——, Subjects for 107
Paraffin as a Motorcar Fuel R. J. Dunderdale 166
Park, J. Cyanide Process of Gold Extraction (Review) 286
—— —— Text Book of Mining Geology (Review) 46
Parmelee, H. C. Cyaniding Slimy Ore by Continuous Decantation 25
Parral Vat 438
Parsons, C. L. Uranium and Radium Situation, The 107
Parting Apparatus, Description of an Electric F. Wartenweiler 367
Patchin, G., and Bannister, C. O. Detecting Platinum Metals in Beads from Cupellation 435
—— —— *Detection of the Platinum Metals in Cupellation Beads by Means of the Microscope* 478
Discussion 483
Patent Applications, Abstracts of 47, 110, 230, 364, 391, 419, 442, 504
—— ——, Selected Transvaal 47, 111, 168, 231, 287, 336, 364, 392, 420, 442, 504
Patents Bill, The Union A. L. Spoor and W. E. John 296
Discussion .. 302, 379, 408, 428, 465
Payne, A. E. at Annual Dinner 270
Peale, R. Relation of the Mine Manager to Recent Graduates 165
Pebble Loads for Tube Mills 25
—— Weight, Curves of 354
Pebbles, Size of, for Tube Mills 25
Pearce-Low Method of Assay of Tin Ores and Concentrates E. A. Wraight and P. L. Teed 500
Pence, C. M. Potassium Permanganate in the Quantitive Estimation of some Organic Compounds 272
Peroxide as Part Cargo, Danger of Barium P. V. Dupré 345
Pertusi, C., and Gastaldi, E. Detection of Hydrocyanic Acid 152, 203, 347
Petrology, A Manual of F. P. Mennell (Review) 109
Phillips, F. C. Chemical German (Review) 389
—— Sir Lionel, Attempted Assassination tion of 290
Phosphates and small Amounts of Uranium, Analysis of Minerals containing, C. Lepierre and A. de Carvalho .. 347
Phosphorus in Steel, A Rapid Method for the Determination of Haripada Bhattacharyya 347
Phosphorus in Steels containing Vanadium, The Determination of Cain and Tucker 10
Phthisis, Miners', Prevention Committee, Interim Report of 98
Pickard, H. K. on the Effect of Charcoal in Gold-Bearing Cyanide Solutions 341
Piezometer, A 85
Pinder-Berry Stamp Mill S. L. Berry 89
Pineapple Leaves, Utilisation of 109
Pipe-Fitters Underground 339
Pishel, M. A. A Test for Coking Coals 208
Pitchblende 108
Pitot Recorder 227
Platinum in Bullion, Determination of, Trenker 310
—— Metals in Beads from Cupellation, Detecting C. O. Bannister and G. Patchin 435
—— —— *in Cupellation Beads, Detection of the, by Means of the Microscope, C. O. Bannister and G. Patchin* .. 478
Discussion 483
—— ——, *A Method of Assaying Concentrates and Battery Chips for Gold and A. F. Crosse* 373
Discussion 374, 422, 483
—— and its Price Fluctuation 96
——, Rhodium and Iridium, Separation of Palladium from the Noble Metals Gold M. Wunder and V. Thüringer 414
——, Tantalum as a Substitute for 202
Poison, Cyanide, First Aid for W. H. Kritzer 43
Pooler, F. J. on the Effect of Charcoal in Gold-Bearing Cyanide Solutions 371
—— on Personal Allusions in Papers .. 394
—— Responding to vote of thanks for Scrutineers 474
Pope, J. Sampling Coal Deliveries .. 271
Porcupine, Mining Methods at A. A. Cole 164
Portland Cement, Everyday Uses of (Review) 109
—— —— Gripping Power of H. Luftschitz 388
—— ——, Improvements in Manufacture of G. T. Hill and C. G. Stone (Patent abstract) 364
Potassium Bromate, Volumetric Determination of Tin with F. Fichter and E. Muller 10
—— Permanganate in the Quantitative Estimation of some Organic Compounds C. M. Pence 272
——, Rapid Modified Chlorplatinate Method for the Estimation of, W. B. Hicks 271
Powell, O. P., and Caldecott, Dr. W. A. The Sand Filling of Mines 119
Discussion 131, 264, 303, 368
Do. Reply to 409
Power, F. D. Pocket Book for Miners and Metallurgists (Review) 381
Pranke, E. J. Cyanamide in Agriculture 153

PAGE

Pratt, J. R., and Harrison, O. Improvements in means for Controlling the Underflow from Classifiers (Patent abstract) 47
Precipitate, Cyanide, Handling, at Lluvia de Oro — H. R. Conklin . 84
Precipitating Apparatus, Improvements in, L. D. Mills (Patent abstract) .. 47
Precipitation of Cyanide Solutions — C. W. Merrill 97
— at Deloro, Ontario. Aluminium. S. F. Kirkpatrick 346
— Losses 349
— Temperatures — J. Gross 78
— — E. H. Smith 22
— *by Zinc, a New Method of — J. S. MacArthur. Discussion* 1
Reply to Discussion 265
— of Zinc 355
—, Zinc Dust 97
— — — of Gold and Silver — A. M. Merton 317
Preece, Sir Wm. (Obituary) 233
President (Mr. A. Richardson) The, Inaugural Address 51
— —, Remarks by .. .. (See under Richardson, A.) .. ..
—, Inauguration of 485
Price, W. S. V., on Electric Blasting. 190, 251
— on Holding July Meeting 475
— on Prevention of Dust in Underground Workings 459
Primer Clip for Explosives — William Cullen 366
Pring, J. N., and Tainton, U. C. The Electrodeposition of Zinc at high Current Densities 500
Prizes awarded at Third Annual Mining Exhibition 489
— Offered by the South African Red Cross Society 491
Proceedings at Ordinary General Meetings—
Aug. 16, 1913 49
Sept. 20, 1913 113
Oct. 18, 1913 169
Nov. 16, 1913 233
Dec. 13, 1913 289
Feb. 21, 1914 365
Mar. 21, 1914 393
Apr. 18, 1914 421
May 16, 1914 445
(Annual) June 20, 1914 469
Progress in Metallurgy 439
Props and Beams in Mines, Reinforced Concrete — Prof. S. M. Dixon 31
Provay, G. A. P. Improvements in Truck Tipplers (Patent abstract) 110
Pulp, Dewatering Wheel for — R. B. Seeber 276
—, Flow of, Curves of 354
—, Mechanical Stirring of 437
—, Water Ratio in 122
Pulverizers, Improvements in — S. S. Osborn (Patent abstract) 231
Purser, A. Improvements in Valves (Patent abstract) 231
Pyro Boiler Cleaning System 285

R

PAGE

Radium in Radio-active Substances. Determination of — A. L. Fletcher 383
— and Uranium Situation, The — C. L. Parsons 107
Rainfall, Relation of Accident and Death-rate to 395
Ramsay, Sir W. — Argon and its Place in Nature 75
Rand Metallurgical Practice — W. A. Caldecott and Others (Review) .. 166
— Mutual Assurance Co. 113
— Terminology — T. A. Rickard .. 361
Rare Metals in Copper 313
Rate of Solution of Gold in Cyanide Solution — J. Gross 79
Reactions for Gold, Delicate 9
Reagent, Battendorf's, Detection of Arsenic by — L. W. Winkler 9
Recovery of Black Sand and Floating Particles of Metallic Minerals — J. M. Neill. Discussion 72
Reply to Discussion 133
Red Cross Society, South African 50
— Prizes Offered by 491
Reduction Plant, Description of, at Crown Mines 412
— — at Shamva Mine 75
Reef Values, Variation in 69
Referee Assays 367
Refractory Gold and Silver Ores. Treatment of — C. Butters 317
Regeneration, Acidity and 357
— Alkalinity and 357
Registration and Training of Engineers. G. A. Julius 42
Renaud, E. J. Selective Mining 280
Report of the Chief Inspector of Explosives for the Union of South Africa for 1912. Annual — A. B. Denne 417
— of Council, Annual 469
Rescue Appliances 280
— — Lecture on Ambulance and, by Dr. L. G. Irvine 497
Respirators as Dust Catchers 99
Retarding Effects on the Dissolving of Gold and Silver in Aqueous Cyanide Solutions — J. J. Andrejew .. 312
Reunert, T. Improvements in Mortar Boxes (Patent abstract) 47
Reviews and New Books 45, 109, 151, 166, 230, 286, 355, 362, 381, 389, 418, 441, 503
Rhodium and Iridium. Separation of Palladium from the noble metals Gold, Platinum — M. Wunder and V. Thüringer 414
Richards, R. H. Ore Dressing Improvements 314
— — Standard Screens for Screen Analyses 12
Richardson, Alex. (President), at Annual Dinner 267, 270
— — on Annual Report of Council .. 472
— — on Death of Sir Wm. Preece . 233
— — on Detection of Platinum Metals 483
— — on Disaster at Universal Colliery 170
— — on Election of Officers and Council 473

PAGE
Richardson. Alex. on Electric Blasting 192
—— —— on Explosives in Hand and Machine Labour Stopes 244
—— —— on Holding July Meeting 475
—— —— *Inaugural Address* 51
—— —— on Inauguration of President .. 485
—— —— on Iridium Prize Papers and Presentation of Prizes 50
—— —— on Mining Copper Ores at Messina 64, 195
—— —— on Mining Exhibition 51. 421, 488
—— —— on Presentation of Society's Bronze Medal 475
—— —— on Prevention of Dust in Underground Workings 433
—— —— on Sand Filling of Mines 133
—— —— on Seymour Memorial Library 290
—— —— on South African Red Cross Society 50
—— —— on Visit to De Beers 366, 474
—— —— on Witwatersrand Earth Tremors 427
Rickard. T. A. Rand Terminology .. 361
Ridgway Machine, Improved 171
Rietfontein Mines. Iridosmine from the New C. B. Horwood 360
Rittinger's Axioms and Formulæ 87
—— Law 91
—— Sieve Scale 12
Roasting. Efficiency of 31
Roberts, E. on Electric Blasting 190
Robertson, G. A. on Recovery of Black Sand and Floating Particles of Metallic Mineral 72
Robinson, C., and Towne, R. L. Improvements in Processes for Precipitating Metal from Solutions (Patent abstract) 419
Robson. R., and Salters Regulating Feed for Sand Cones (Patent abstract) 504
Rock Bursts 331
—— Drills, Compressed Air *versus* Hydraulic Power for G. A. Denny 163
—— ——. Furnaces for Heating. Improvements in S. T. Tregaskis (Patent abstract) 47
—— ——. Improvements in Supporting. G. H. Chalkley (Patent abstract) 47
—— ——. Small Plugger 164
—— —— Testing 332
—— *Temperatures E. J. Moynihan* .. 455
—— *Values, Waste, Notes on R. Dures* 233
Rocks. Cement Injection for Consolidating M. de Marillac 333
Rodger. R. Improvements in Mills or Apparatus (Patent abstract) .. 110
Rogers. A. Edited by. Industrial Chemistry (Review) 389
Roll Crushers, Improvements in. R. C. Newhouse (Patent abstract) 231
Rollers for Haulage Ropes. Improvements in R. Blane and J. D. Silcock (Patent abstract) 442
—— for Mine Shafts, etc., Improvements in G. Stott (Patent abstract) 364
Rose, T. Kirke The Annealing of Gold 276
Rotary Vacuum Filters T. Breakell 214

PAGE
Rothenfusser, S. The Detection of small Quantities of Nitrates in Water.. 415
Rothmund. V., and Burgstaller. A. Rate of Decomposition of Ozone in Aqueous Solution 75
Rowland. Fred (Secretary) on Ambulance Competitions 169, 289
—— —— On Annual Dinner 169
—— —— on Evening Class Examinations Results 289
—— —— on International Engineering Congress 49
—— —— on Johannesburg Trades School 50
—— —— on Mining Exhibition 169, 393, 421
—— —— on Sixth International Mining Congress 393
—— —— on South African Red Cross Society 50
—— —— on Vote of Thanks to Exhibition Committee 475
—— —— on Mr. Percy Wheeler 50
Rusden, H. On Assaying Concentrates and Battery Chips 374
Rusting of Iron. A New Chemical cause of. Vaubel 271

S

Safe Handling of Missed Holes. The 386
Safety Appliances in Mines 34
—— First Exhibits at Third Annual Mining Exhibition 490
—— Measures in regard to the Use of Explosives (Lecture) W. Cullen 498
—— Winding Gear J. F. Staley and C. F. Jackson (Patent abstract) .. 231
Samplers, Machine 21
Samples, Crushing and Mixing 20
——, *Mine, Notes on the Assay of Robert Dures. Discussion* 68, 173
Reply to Discussion 263
Sampling and Assaying, Coarse Gold in F. White 80
——, Mortar 277
——, Principles of Ore A. van Zwaluwenburg 19
——, Theory of 19
Sand Cones, Regulating Feed for R. Robson and Salters (Patent abstract) 504
——, Definition of 83
—— Filling 331
—— —— A. H. Martin 358
—— —— Costs 305
—— —— *of Mines, The W. A. Caldecott and O. P. Powell* 119
Discussion 131, 264, 303 368
Do. Reply to 409
—— —— Tests 120
—— Pulp. Launder Grades for Dr. W. A. Caldecott 486
—— Pumps, Improvements in J. H. Wilson and R. Craib (Patent abstract) 391
——, Surface Transport of 125
Sanitation. Chloride of Lime in A. H. Hooker 285
——, Mine W. H. Moulton 333

PAGE

Sawdust. Extinguishing Fires with 284
Sawyer, R. E. on the Sand-filling of Mines 303
Scales. Weighing Light Material on Ordinary 334
Scandinavia. Fixation of Nitrogen—Progress in 346
Schafer and Dennstedt. Dangers of Coal 285
—— —— —— Storage of Coal 468
Scheelite in New Zealand. Treatment of Gold-Bearing C. W. Gudgeon 436
Schorne's E., Elutriation Apparatus.. 85
School of Mines Examination Results .. 289
Schrader, E. J. Slow-Speed Chilean Mills 274
Schramm, J. A Simplified Method for Coal Distillation, &c. 346
Schuller, C., and Wunder, M. Determination of Magnesium as Ammonium Magnesium Phosphate and Ammonium Magnesium Arsenate 347
Schwerin, Dr. G. B. Electro-osmotic Extraction of Water (Patent abstract) 231
Scott, A. R. Analyses of Ferro-Titanium 274
Screening, Fine A. McA. Johnston, and Discussion 394
Screens, I. M. M. 12
—— for Screen Analysis, Standard R. H. Richards 12
Scrutineers, Election of, for Annual Meeting 445
—— Report on Election of Officers and Council 473
Secretary, The (Fred Rowland)—on Ambulance Competitions 169, 289
—— —— on Annual Dinner 169
—— —— on Evening Class Examination Results 289
—— —— on International Engineering Congress 49
—— —— on Johannesburg Trades School 50
School 50
—— —— on Mining Exhibition 169, 393, 421
—— —— on Sixth International Mining Congress 393
—— —— on South African Red Cross Society 50
—— —— On Vote of Thanks to Exhibition Committee 475
—— —— on Mr. Percy Wheeler 50
Seeber, R. B. Dewatering Wheel for Pulp 276
Selective Mining E. J. Renaud 280
Sensitiveness of Explosives E. A. Mann 440
Settling, Free 87
Seymour Memorial Library 290
—— ——, New Accessions to .. 363, 418
Shaft Rollers, Bearings for A. H. Guy (Patent abstract) 230
—— ——, &c., An Improvement in A. H. Guy (Patent abstract) 110
—— Sinking 332
Shaking Tables. Improvements in J. Stuart (Patent abstract) 231
Shamva Reduction Plant 75

PAGE

Sharwood, W. J., Dr. Assaying by Iron Nail Method 345
—— ——, and Clark, A. J. Grinding Pans *versus* Tube Mills 95
Side Tip for Conveyor Belt W. J. Nicol 96
Sieving, Fine *versus* Coarse 68
Signal Systems in Mines 34
Silcock, J. D., and Blane, R. Improvements in Rollers for Haulage Ropes (Patent abstract) 442
Silica in Hydrochloric, Nitric and Sulphuric Acids, Solubility of W. Wunder and A. Suleimann 500
Silicon, Uses of Cast 202
Sill, H. A. and R. T. Zinc Dust Feeder 89
Silver, Allowance for 69
—— in Aqueous Cyanide Solutions, Retarding Effects on the Dissolving of Gold and J. J. Andrejew 312
——, Assay for, by Cyanide Regeneration 316
—— in Cyanide Solutions, The Dissolution of Gold and I. I. Andreev .. 383
—— Cyaniding at Tonopah H. A. Megraw 16
—— Estimation in Mill Solutions, Rapid, G. H. Clevenger 200
—— in Gold Bullion 69
—— by Iron Nail Method, Assay of Gold and E. J. Hall and C. W. Drury 415
—— in Litharge 69
—— Ores at Cobalt. Desulphurising, J. J. Denny 323
—— —— Treatment of Refractory Gold and C. Butters 317
—— The Spitting of H. B. Baker .. 467
——, Zinc Dust Precipitation of Gold and, A. M. Merton 317
Sim, A. C. M. on Witwatersrand Earth Tremors 427
Simon, Dr. A. L. on the Effect of Charcoal in Gold-Bearing Cyanide Solutions 341
Simonds, E. H. Milling by Coarse Crushing in Cyanide Solution .. 27
Sinking Operations at the Springs Mines, Notes on B. D. Bushell .. 101
Slime Agitation C. F. Spaulding 76
—— —— for Cyaniding H. A. Megraw 436
—— Settlement. H. N. Spicer 219
Smith, E. E. Acetylene 441
—— E. H. Precipitation Temperatures 22
——, G. Hildick, on Election as Member of Council 473
—— ——, on Vote of Thanks to Secretary 474
—— —— on Witwatersrand Earth Tremors 485
Soda, Analyses of Crude Calcined 238
—— *Deposits of Africa, Natural John Watson* 235
Discussion 241, 377
Do. Reply to 463
—— Lakes of Egypt 235
Sodium Iodide Solution, The Solubility of Iodine in H. W. Gill 290
Discussion 292, 343, 378
Do. Reply to 476
Solder for Aluminium 276

PAGE

Solubility of Iodine in Sodium Iodide Solution *H. W. Gill* .. 290
Discussion .. 292, 343, 378
Do. Reply to .. 476
Solution Control .. 356
—— —— in Cyaniding A. W. Allen 354
——, Fatigue of .. 325
Solutions, Aqueous, Electrolysis of, of the Simple Alkaline Cyanides C. V. Hall .. 416
—— Cyanide, Electrolysis of E. F. Kern .. 384
—— ——, Molybdenum in J. E. Clennell .. 436
——, Heating Mill .. 11
Solvent for Precious Metals H. Foersterling .. 383
Some Observations on the Graphite Deposits of Madagascar and Africa. E. R. Bawden. Reply to Discussion .. 1
Spaulding, C. F. Slime Agitation .. 76
Specific Gravity Bottle and Dilatometer Combination .. 272
Spengel, A., and Fichter, F. Reactions of Aluminium Nitride .. 155
Spicer, H. N. Slime Settlement .. 219
Spirits and Vinegar Act, 1913. Wine, Union of South Africa .. 347
Spoor, A. L., and John, W. E. The Union Patents Bill .. 296
Discussion 302, 379, 408, 428, 465
Sprays in Drilling, Disadvantage of .. 281
——, Use of Fixed .. 339
Springs Mines, Notes on Sinking Operations at the B. D. Bushell .. 101
Stability of Explosives, International Commission for Unification of the Testing Methods of the .. 417
Stacey, D. W. Improved Process and Apparatus for Separating Solutions Containing Metals from Insoluble Substances (Patent abstract) .. 392
Stadler, H. Grading Analyses by Elutriation .. 85
Staley, J. F. Safety Winding Gear (Patent abstract) .. 231
Stamp Mill Curves and Speeds. S. L. Berry .. 155
Stamps *versus* Tube Mills. R. Gascoyne .. 416
Standard Base at Johannesburg, The New 221
—— Screens for Screen Analysis. R. H. Richards .. 12
Stanley, Professor G. H. at Annual Dinner .. 268
—— ——, on Assaying Concentrates and Battery Chips .. 374
—— ——, on Election as President 473, 486
—— ——, on Fine Screening .. 394
—— ——, *Hydraulic Classifiers and Classification* .. 445
Discussion .. 452, 484
—— ——, on Holding July Meeting .. 475
—— ——, on Inaugural Address .. 53
—— ——, on Mining Copper Ores at Messina .. 64, 402
—— ——, on Papers and Membership .. 411
Stanley, Professor G. H. on Personal Allusions in Papers .. 394
—— ——, on Primer Clips for Explosives 367
—— ——, on Vote of Thanks to Scrutineers .. 474
—— ——, on Witwatersrand Earth Tremors .. 426, 466
Stansbie, J. H. Electrolysis of Copper Solutions .. 270
Stansfield, J. F., and Elsden, A. V. Determination of Small Quantities of Lead .. 152
Steel, Determination of Cobalt and Uranium in H. Köning .. 414
—— and Iron, Corrosion of W. H. Walker .. 204
—— Points for Core Drilling. M. L. Martel .. 220
——, A Rapid Method for the Determination of Phosphorus in, Haripada Bhattacharyya .. 347
Steels containing Vanadium, Determination of Phosphorus in Cain and Tucker .. 10
Steiner, K., and Meyer, H. A New Carbon Oxide .. 10
Stem Guides, Improvements in Stamp, W. Daly (Patent abstract) .. 47
Stevens, T. B. on Vacuum Filtration at the Victoria Mill, Waihi .. 171
Stone, C. G., and Hill, G. T. Improvements in Manufacture of Portland Cement (Patent abstract) .. 364
—— Quarrying, Practical A. Greenwell and J. V. Elsden (Review) 166
Stopes, Hand and Machine Labour, Explosives in T. D. Delprat .. 241
Discussion .. 307
——, Watering .. 338
Stoping, Methods of, at Messina .. 59
Stott, G. Improvements in Rollers for Mine Shafts, &c. (Patent abstract) .. 364
Strike, The General .. 337
Strutt, R. J. An active Modification of Nitrogen produced by the Electric Discharge .. 152
Stuart, J. Improvements in Shaking Tables (Patent abstract) .. 231
Students, New 49, 113, 233, 365, 421, 445
——, Training of .. 165
Sugar Mill Residues, Cyanide from C. A. Browne .. 202
Suleimann, A., and Wunder, W. Solubility of Silica in Hydrochloric, Nitric, and Sulphuric Acids .. 500
Sulphide in Natural Water, Determination of Small Amounts of Hydrogen W. Winkler .. 414
Sulphur, Influence of, on Cast Iron H. I. Coe .. 95
——, Rare Forms of .. 329
Sulphuric Acid on Copper, The Action of J. T. Cundall .. 500
Sulphurous Acid, Standardising .. 309
Surr, G. Care in Laboratory Work .. 382
Survey Base Line, New Standard, at Johannesburg .. 221
Surveying, Mine .. 51

T

PAGE

Tailings, Compression Tests of 120
—— Dumps, Treating 415
Tainton, U. C., and Pring, J. N. The Electro-deposition of Zinc at High Current Densities 500
Tanks, Acid Proof Lining for 388
Tantalum as a Substitute for Platinum .. 202
Tappets, Improvements in Stamp W. Daly (Patent abstract) 47
Taps for Burettes, Improved Prof. J. A. Wilkinson 114
Taylor, Wilfrid (Obitnary) 344
Technical Committee of the Burma Corporation, The 502
Teed, P. L. Determination of Water in Coal 152
—— ——, and Wraight, E. A. Assay of Tin Ores and Concentrates .. 499
Telephony in Mines, Wireless 333
Telluride Ores of Gold, Concentration of, H. E. Wood 205
Tellurides, Successful Treatment of 329
Temperature Apparatus, Constant F. H. and P. V. Duprè 272
—— of Decomposition of Dolomite. O. Kallauner 383
—— in Mines in Great Britain, Notes on the Effect of J. Cadman .. 388
—— of Underground Workings, Ascertaining 402
Temperatures of Decomposition of Carbonates K. Friedrich 9
——, Precipitation J. Gross 78
——, —— E. H. Smith 22
——, *Rock* *E. J. Moynihan* 455
Terminology, Rand T. A. Rickard .. 361
Terni, A., and Ciusa, R. A Reaction of Iodine-ions 203
Testing Methods of the Stability of Explosives, International Commission for Unification of the 417
—— Rock Drills 332
Tests of Reinforced Beams 32
Theory and Practice Sir F. Donaldson 39
Thiocarbamide and the Interaction of Nitrous Acid and Thiocarbamide in the Presence of Acids of Different Strength. The Estimation of Nitrates by means of M. E. Coade and E. A. Werner 346
Thiosulphates, Colour Reaction for the Detection of E. Pozzi-Escot .. 73
Thomas, H. T. on Cyanide Practice in India 66
——, W. Losses in Tin Dressing .. 94
Thüringer, V., and Wunder, M. Separation of Palladium from the Noble Metals 414
Tillmans, J., and Heublein, O. The determination of Chlorine in Natural Waters 414
Timber Preservation 286
Timbering 331
Timm, F. C. W. Process of Treating Minerals, &c. (Patent abstract) 420
Tin, Delicate Reagent for C. Zenghelis 75
—— Deposits, Transvaal G. H. Blenkinsop 33

PAGE

Tin Dressing, Losses in W. Thomas 94
—— Extraction Process, Chenall's Patent 278
—— from the Ore, Chemical Extraction of, J. W. Chenall 19
—— Ores, Assay of H. W. Hutchin 499
—— —— and Concentrates, Assay of Pearce-Low Method 500
—— —— ——, —— —— E. A. Wraight and P. L. Teed 499
—— ——, Reduction of 75
——, *Proposed Method of Assay of* *J. Whitby* 170
Discussion 171
Reply to Discussion 292
——, Volumetric Determination of, with Potassium Bromate by the Method of H. Zschokke F. Fichter and E. Muller 10
Titanium, Ferro-, An Analysis of A. R. Scott 274
—— by Titration with Methylene Blue, Determination of B. Neumann and R. K. Murphy 435
—— and its Uses S. J. Johnstone .. 347
Titration Results 354
Tonopah, Silver Cyaniding at H. A. Megraw 16
Toombs, C. Determination of Gold in the Presence of Iridium and Allied Metals in Materials such as Black Sand 4
Discussion 119
Do. *Reply to* 292
Towne, R. L., and Robinson, C. Improvements in Processes for Precipitating Metal from Solutions (Patent abstract) 419
Tregaskis, S. T. Improvements in Furnaces for Heating Rock Drills (Patent abstract) 47
Tremors, Witwatersrand Earth *H. E. Wood* 423
Discussion 426, 466, 485
Trenker, Determination of Platinum in Bullion 310
Trent Agitator 438
——, L. C. Apparatus for Treatment of Ores, &c. (Patent abstract) .. 391
Trinitrotoluol 441
Truck Tipplers, Improvements in G. A. P. Provay (Patent abstract) .. 110
Trucks, Bottom Discharge, Improvements in E. Ullman (Patent abstract) 419
Tube Mill Circuit 23
—— —— —— at the Alaska Treadwell Mill 320
—— ——, Diagrammatic Representation of the Action in a 353
—— ——, Diameter of 23
—— —— for Emulsifying Zinc Dust .. 207
—— —— Feed, Moisture in 25
—— ——, Grinding, Diagram Illustrating 23
Length of 23
—— —— Liners 17, 468
—— —— ——, Improvements in J. Olivier (Patent abstract) 504
—— —— Pebbles, Size of 25
—— —— Power 24

PAGE

Tube Mill Practice. F. C. Brown 467
—— —— Speed 468
—— —— Use of the 349
—— Milling. The Open Circuit in 24
—— ——. The Ordinary or Closed Circuit in 23
—— Mills. Diameter of Discharge Opening of 468
—— ——, Dimensions of 29
—— —— *versus* Grinding Pans. Clark and Sharwood 95
—— ——. Improvements in. W. Fennell (Patent abstract) 231
—— —— —— in Linings for the Ends of. W. B. Dodds and D. C. Bowen (Patent abstract) 364
—— ——, Load and Pebbles for 467
—— ——. Moisture of Feed for 468
—— ——, Pebble Load for 25
—— ——. Rapid Discharge System for .. 501
—— ——, Selection and Operation of. A. M. Merton 23
—— ——, Speed of 24
—— —— *versus* Stamps. R. Gascoyne 416
Tube Wells. Improvements in. J. Ashford (Patent abstract) 110
Tuberculosis in Underground Workings, Risk of Direct Infection by 100
Tucker and Cain.—Determination of Phosphorus in Steels containing Vanadium 10
Tungsten. Applications of Ductile. C. G. Fink 205
——. The Effect of, on the Ammonium Molybdate Assay of Lead. H. Lavers 466
—— Furnaces 206
—— Gauze 206
Tyler Co., W. S. Standard Screens .. 13

U

Ullman, E. Improvements in Bottom Discharge Trucks, &c. (Patent abstract) 419
Underground Workings, Ascertaining Temperature of 402
—— ——, *The Prevention of Dust in. B. C. Gullachsen* 337
Discussion 433, 459 483
Union Observatory, Visit to 169
—— *Patents Bill. A. L. Spoor and W. E. John* 296
Discussion .. 302, 379, 408, 428, 465
Universal Colliery, Disaster at 170
Universities. The Chemical Industries and the. M. T. Bogert 311
Uranium. Analysis of Minerals containing Phosphates and Small Amounts of. C. Lepierre and A. de Carvalho .. 347
—— and Radium Situation. C. L. Parsons 107
—— in Steel. Determination of Cobalt and. H. Köning 414

V

Vacuum Filtration at the Victoria Mill. W. Macdonald. Discussion 67. 171
Reply to Discussion 430

PAGE

Valuation, Mine, Principles of. J. R. Finlay 36
Valuations, Mine, Life Factor in 36
Valves, Improvements in. A. Purser (Patent abstract) 231
Vanadium, Determination of Phosphorus in Steels containing. Cain and Tucker 10
Vaubel.—A New Chemical Cause of Rusting of Iron 271
Veasey, J. H. on Explosives in Hand and Machine Labour Stopes 307
—— —— on Mining Copper Ores at Messina 306
—— —— on Sand Filling of Mines 370
—— —— on the Ventilation of the Mines of the Rand 197, 248
Velocity of Fall, Relation Between Average, and Diameter of Quartz Particles 88
Ventilation of the Mines of the Rand: the Problem of obtaining Healthier Conditions. G. H. Blenkinsop 148
Discussion 151, 195, 244. 306
Do. Reply to 402
Venturi Tubes 223
Vinegar Act, 1913. Wine, Spirits and. Union of South Africa 347
Viscosimeter. Comparison of Redwood's with Engler's. W. Meissner .. 201
Viscous Resistance, Law of 87
Visit to Crown Mines, Ltd. 412
Volatilisation, Extracting Gold by. B. Howe 14
—— Gold Recovery by. S. Croasdale 315
—— Process. The Howe. B. Howe .. 385
Volumetric Determination of Gold. V. Lenher 309

W

Walker, W. H. Corrosion of Iron and Steel 204
Walters, E. A. at Opening of Mining Exhibition 488
Walton, A. J. on the Ventilation of the Mines of the Rand 196
Wannenburg, G. J. L. A Dust Preventer (Patent abstract) 391
Wartenweiler, F. Description of an Electric Parting Apparatus 367
Washoe Concentrator, Launder Data of the 320
Waste Rock Values, Notes on .. .. R. Dures 233
Water Blast, Sketch of 462
—— in Coal, Determination of. P. L. Teed 152
——. The Detection of small Quantities of Nitrates in. S. Rothenfusser 415
——. Determination of small Amounts of Hydrogen Sulphide in Natural. W. Winkler 414
——. The Estimation of Traces of Chlorides in. M. Lombard 383
Water-Gauge, The 219
——, Improvements in Apparatus for Filtering. M. Deacon and W. Gore (Patent abstract) 442
—— Purification. Solubility of Calcium Carbonate and its Bearing on. W. T. Burgess 435

PAGE
Water Supply in Mines 339
Watering Development Places 338
Waterproofing of Concrete Structures 282
Waters. The Determination of Chlorine in Natural J. Tillmans and O. Henblein 414
——, Potable, Alkalinity of 295
——, *The Determination of the Acidity or Alkalinity of, A Study in Indicators* *Dr. J. Moir* 114
Discussion 118, 294
Do. Reply to 404
——, *W., Cullen, W., and Donaldson, T. Electric Blasting* 174
Discussion 189, 248, 308, 375
Do. Reply to 431
—— —— on Electric Blasting 251
—— —— on Explosives in Hand and Machine Labour Stopes 243
Watson, F. W. on Assaying Concentrates and Battery Chips for Gold and Platinum Metals 422
—— —— on Holding July Meeting of Society 475
——, John on Annual Dinner 290
—— —— on the Assay of Mine Samples 68
—— —— on the Determination of the Acidity or Alkalinity of Waters.. 294
—— ——, *Natural Soda Deposits of Africa: with some Notes on the Alkali Trade* 235
Discussion 241, 377
Do. Reply to 463
Wattle Bark Industry, The Natal.. 285
Weather Changes, Relation of Falls of Roof in Collieries to *C. J. Gray* 395
Discussion 400, 433
Weighing Light Material on Ordinary Scales 334
Weights, Standard 206
Werner, E. A., and Coade M. E. The Estimation of Nitrites by means of Thiocarbamide and the Interaction of Nitrous Acid and Thiocarbamide in the presence of Acids of different strength 346
Weston, E. M. on Electric Blasting 190, 308
—— —— on Explosives in Hand and Machine Labour Stopes 243
—— ——, *Gold Production in Relation to Humanity; a Study of the Question: Has Gold depreciated in Value? Discussion* 64, 136
Do. Reply to.. 252
—— —— on Holding July Meeting 475
—— —— on Mining Copper Ores at Messina 401
—— —— on Prevention of Dust in Underground Workings 461
—— —— on Sand Filling of Mines 409
—— —— on the Ventilation of the Mines of the Rand 199
Whaling Industry, The South African .. 286
Wheeler, Dr. R. V. Inflammable Dusts 39
—— ——, and Jones, D. T. The Composition of Coal 435
Whitby, A. on Assay of Mine Samples 173
—— —— on Natural Soda Deposits 241
Whitby, A. *Proposed Method of Assay of Tin* 170
Discussion 171
Do. Reply to 235
—— —— on Referee Assays 367
—— A. and J. Improved Pole for Mine Packs (Patent abstract) 504
White, Franklin. Coarse Gold in Sampling and Assaying 80
—— H. A. on Assay of Tin 171
—— —— on Detection of Platinum Metals 483
—— —— on Hydraulic Classifiers and Classification 452
—— —— on Sand-filling of Mines 264
—— —— on Seymour Memorial Library 290
—— —— on Union Patents Bill 379
—— —— on Visit to Union Observatory .. 169
—— —— on Vote of Thanks to Exhibition Committee 475
—— —— on Witwatersrand Earth Tremors 426, 485
Whitehouse, J. on the Ventilation of the Mines of the Rand 195
Wiley, E. M. Extraction of Nitrogen from Air 272
Wilfley Tables 314
Wilkinson, Prof. J. A. on the Acidity or Alkalinity of Waters 118
—— —— at Annual Dinner 270
—— —— on Assaying Concentrates and Battery Chips 374
—— —— on Improved Taps for Burettes 114
—— —— on Natural Soda Deposits of Africa 377
—— —— on the Solubility of Iodine in Sodium Iodide Solutions 292
—— —— on Union Patents Bill 428
Williams, H. E. *Analyses of a Cyanide Solution from the Meyer and Charlton Mine* 475
Williams, John R. on Annual Report of Council 472
—— —— on Inaugural Address 53
—— —— on Iridium Prize Papers 50
—— —— on Natural Soda Deposits 241
Wilson, J. H., and Craib, R. Improvements in Sand Pumps (Patent abstract) 391
Wine, Spirits and Vinegar Act, 1913, Union of South Africa 347
Winkler, L. W. Detection of Arsenic by Bettendorf's Reagent 9
——, W. Determination of small Amounts of Hydrogen Sulphide in Natural Water 414
Wireless Telephony in Mines 333
Weston, E. M. on Electric Blasting 190, 308
—— —— on Explosives in Hand and Machine Labour Stopes 243
Witwatersrand Earth Tremors, H. E. Wood 423
Discussion 426, 466, 485
—— Mining Conditions R N. Kotze 330
Wöhler and Krupko.—Sensitiveness to Light of Azides 270
Wood, H. E. Concentration of Telluride Ores of Gold 205
—— —— *Witwatersrand Earth Tremors* 423
Discussion 426, 466, 485

PAGE

Wood. Ignition of 389
—— Steamer. Artificial Coal from a, E. Heuser 334
Woodburn, J. Allan,—Mining Copper Ores at Messina 53
Discussion 192, 244, 306
Do. Reply to 401
—— —— on Ventilation of the Mines of the Rand 247
Work. Care in Laboratory G. Surr .. 382
Working Costs in Canada 360
Wraight, E. A., and Teed, P. L. Assay of Tin Ores and Concentrates .. 499
Wunder, M., and Schuller, C. Determination of Magnesium as Ammonium Magnesium Phosphate and Ammonium Magnesium Arsenate 347
Wunder, M., and Thüringer, V. Separation of Palladium from the Noble Metals 414
—— W., and Suleimann, A. Solubility of Silica in Hydrochloric, Nitric and Sulphuric Acids 500

Y

Yates, Prof. J., on Mining Copper Ores at Messina 64

Z

Ziebarth. Accident with a Benzole Locomotive at the Rossleben Potassium Salt Mines 360
Zenghelis, C. Delicate Reagent for Tin 75
Zinc and Aluminium Methods, Comparative Cost of 161
—— Box Products, Tests on 78
——, Consumption of 2
—— Dust, Chemical and Physical Characteristics of Good 209
Zinc Dust, Composition of 210
—— ——, Consumption of 161
—— —— Feeder H. A. and R. T. Sill 89
—— —— Feeders 207
—— —— Precipitating Plant, Diagram of 318
—— —— Precipitation 97
—— —— —— Equipment D. F. Irvin 206
—— —— —— of Gold and Silver A. M. Merton 317
—— ——, Sampling 210
—— ——, Specifications and Tests for A. M. Merton 209
—— ——, Storage of 210
—— ——, Use of, in Mexico 84
—— Fouling Solution 158
—— at High Current Densities, The Electrodeposition of J. N. Pring and U. C. Tainton 500
—— and Lead in Ores 328
—— Losses 356
——, Mercury Destructive of 328
——, Metallic, Determination of 210
—— Oxide, Determination of 210
—— as Precipitant, Alloy of C. W. Merrill 205
——, Precipitation of 355
——, —— *by A new Method of J. S. MacArthur Discussion* 1
Reply to Discussion 265
—— as Zinc Ammonium Phosphate and as Zinc Pyrophosphate, Estimation of T. M. Finlay and A. C. Cumming 346
Zirconium 73
Zoutpan (North of Pretoria), Soda Deposits 236
Zschokke, H. Volumetric Determination of Tin with Potassium Bromate by the Method of F. Fichter and E. Muller 10
Zwaluwenburg, A. van—Principles of Ore Sampling 19

TN
1
S66
v.14

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